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The Position of Electrochemistry in the Chemical and Metallurgical Industries.

When the story of the industrial evolution of the United States will be written by some future historian who must be both a philosopher and an engineer, one of his brightest chapters will deal with the glorious history of electrochemistry at Niagara Falls. On Aug. 26, 1895, the Niagara works of the Pittsburgh Reduction Company started the manufacture of aluminium. On Oct. 19 of the same year the current was turned on at the works of the Carborundum Company. In quick succession came calcium carbide, artificial graphite, phosphorus, ferro-alloys, made in the electric furnace, and metallic sodium, caustic and chlorine, chlorates, made by electrolysis. In the span of 15 years these pioneer electrochemical industries and others that followed have become so firmly established in the industrial world that their products are well-nigh indispensable in present civilization. We have forgotten that not so long ago aluminium, carborundum, calcium carbide were unknown. The revolutionary features of the early inventions are gone. What remains are splendidly flourishing industries developing along logical lines into ever-extending fields of usefulness. We have almost learned to look at the pioneer Niagara works as the older generation of electrochemical industries. This is said in a spirit of filial pride. For if Germany may be justly proud of having nursed and raised the coal-tar chemical industries, the United States can claim applied electrochemistry as distinctly American. The bulk of these electrochemical industries was not imported, it was made, born, nursed and raised at Niagara by such pioneers as Acheson and Hall and by those who worked with them or followed them. So far they represent the most distinctly American phase in the history of applied chemistry. But from a broader viewpoint—that of the conservation of our natural resources about which many talk so much and think so little—electrochemists have done more at Niagara. They have made at least a start in modifying popular conception of beauty in the direction expressed in Lord Kelvin's remark that "the great cataract will never be beautiful until it has ceased to be such an awful and unlovely example of waste." There is dramatic beauty in the kinetic energy of the Falls, but there must be and will be as much beauty in the transformation of this kinetic energy into the potential energy of innumerable electrochemical products for distribution for the welfare of mankind all over the land.

Aside from these specifically electrochemical industries, electrochemical methods are exerting an ever-increasing and broadening influence in general metallurgical and chemical industries. Of course, there is nothing essentially new in this development. Electroplating is one of the oldest electrical arts, and electrolytic copper refining, carried out on a large scale since the start of the Balbach refinery in 1883, has now assumed enormous proportions and has become indispensable in modern copper metallurgy, both on account of the necessity of pro-

ducing very pure copper for electrical work and of the easy recovery of the precious metals. But, after all, these were applications of electrochemistry to isolated though important problems. The more general tendency towards electrochemical methods did not manifest itself until the success of the aluminum, carborundum, calcium carbide and graphite industries had attracted general attention. To some extent it is the relation of effect and cause. But that is a somewhat superficial view. It cannot be merely incidental that practically simultaneously theoretical electrochemistry came into existence by the foundation of the electrolytic dissociation theory through physicists rather than chemists in Germany and the large electrochemical industries were founded through American engineers rather than chemists. Clearly the scientific as well as the industrial atmosphere was supersaturated and those two nuclei were the first to form and to grow crystallike. Naturally, others followed quickly. So there came into existence the tendency to try and apply electrochemical methods to all possible problems of general chemical and metallurgical engineering. The immense possibilities of industrial electrochemistry became apparent as well as its limitations. Electrolytic refining was extended to silver, gold, and lead. In gold metallurgy there are now on one hand the various attempts to improve the efficiency of the amalgamating plate by making it a cathode in an electrolytic cell and, on the other hand, the proposition to substitute cyanamide with electrolysis for cyanide. In zinc there is great activity on one hand in the development of electric furnace processes, and, on the other, in dry chlorination and electrolysis. In the iron and steel industry the peculiar local conditions of California, Canada, Sweden and Norway have encouraged large-scale and well designed tests of reducing pig iron in the electric furnace; but the largest and most important industrial development is the appearance of the electric furnace for steel melting and refining on a commercial scale. In chemistry the electrochemical fixation of atmospheric nitrogen in form of nitric acid and nitrates or in form of calcium cyanamide affects the explosive and fertilizer industries. Thus there is rapid evolution. The fact that it is still rapid in many lines proves that equilibrium has not been reached, that we are still far from finality. But this evolution is concentrated on distinct promising problems and proceeds along distinct logical lines. The most characteristic feature is perhaps the broadening of the viewpoint from electrochemistry to physical chemistry. Thus in the fixation of atmospheric nitrogen we have in cyanamide manufacture the problem of separating nitrogen from oxygen by cold, in Haber's ammonia synthesis the increase of reaction velocity by certain catalysers with unusually high atmospheric pressures. Thus this evolution has had an immensely broadening effect on all the workers. The rise of the chemist above the pure analyst, the emancipation of the chemical engineer, the dissociation of the metallurgical from the mining engineer are logical concomitant phenomena; and chemical and metallurgical engineering becomes essentially applied physical chemistry.

The Metal Markets in Retrospect and Prospect

It can be well said that just at present the metal markets are slowly recovering from a neurasthenic condition. Now, excessive introspection is equally bad for anybody in such a state. It is usually better to work than to think about the "whys" and "wherefores." Yet, "if we must philosophize, we must philosophize, and if we must not philosophize we must still philosophize." Accordingly, with the New Year's chimes of Trinity Church still ringing in our ears it may not be amiss to look over the metal markets of the past year and also to see what are the signs for the future. Considering the metallurgical industry from the standpoint of a few years back we see that the great increase in production and its concomitant consumption starting in 1898 continued with more or less momentum for 10 years, until the panic of 1907 and the industrial depression that followed it put effectively the brakes on the movement of expansion; that subsequent to this the metal industries recovered very quickly in tone, until the summer of last year, when a second retardation was felt, followed now by a period of liquidation, quiescence and perplexity.

The reasons for this are many—for, indeed, in what dominion of human knowledge is there not a multiplicity of causes? But mainly and chiefly the present unpleasant condition can be attributed to the fact that we recovered entirely too quickly from the period of retrenchment and liquidation of 1907, starting too soon on an error of extravagance and inflation. Secondarily, we have as minor causes the fear of more legislation that is hostile in intent to capital and the uncertainty about the judicial inquiries into the commercial ethics of the trust question and the expounding by the Supreme Court of the eternal verities of the law to changed conditions. Naturally, this has induced a spirit of unrest in the investing public. Direct in its application to the metal markets is the refusal of the railroads to buy in their usual quantities. In quite a human way the railroads do not care to purchase unless they are allowed to raise freight and passenger rates as compensation for the increased wages given their employees. Furthermore, we have the everywhere-present demands of the wage-earning classes for continual increase of pay. In one intense aspect, we have a strong inhibiting cause. That is the high price and uncertain market for cotton, the staple which means so much to us as an exporting nation and so much more to us as a manufacturing nation. All the above enumerated causes act as mental depressants on all industry and though exaggerated in point of fact are none the less potent in their effect on the minds of business men. In short, things are unsettled and no sure foundation for such an expanding industry, such as normally is the metal and chemical industry, can be made until things are more definite.

Let us consider briefly in this light the several special markets aside from the iron and steel market, which is discussed in another column. Like our steel men the copper producers have had also their pleasing prandial conferences. Possibly the dire straits of some of these convivial spirits has moved to compassion the industry and helped to hold up the price of the red metal. At any rate, production has been curtailed. The electrification of the railroads is proceeding much slower than had been anticipated by the large copper interests. The development of the large porphyry ore beds of Utah, Nevada, New Mexico and Arizona has led to the existence of mines that can make copper at an extremely low price. The exactitude with which the cost of development and plant and the cost of producing the metal can be prognosticated has attracted capital rapidly to this kind of mines. The net result is a largely increased and still increasing productive capacity which can sell copper at a profit at 10 cents a pound.

This has effected an over-production and we have seen the price of copper held down to an average of less than 13 cents with a large surplus built up in years of industrial depression. We are not so bearish on the copper industry, especially as enforced economies are reducing the production costs of the older producers. Besides, the low price has caused the use of copper in many new minor ways in the building trade. In this age of electrical development there cannot long be too much copper. It is interesting to note that the high prices of copper in 1906 and in the first half of 1907, due quite largely to the short-sighted policy of the producers in exaggerating the runaway market for the metal, is a cause of this overproduction. For this high price stimulated the promotion and development of these large low-cost producers that now threaten the market. "As ye sow, so shall ye reap." In the steel industry a broad policy has certainly been pursued, even if cynical Wall Street sees nothing but glittering generalities in the pleasant dinners of the gentlemen directing the policy of the different steel concerns. If a similarly broad policy like that observed in the steel industry had been pursued with copper things would have been better for the producer to-day. It is not unnatural to expect in the course of time the consolidation of the several larger concerns into one concern or trust that will control 50 per cent of total production. This will probably be better than the proposed large central selling agency. But the psychological moment for the birth of the copper trust has hardly yet arrived. A logical inference is that the definite character of the ore reserves of the porphyries will lend the stability to the concern that is needed to enlist the support of the banking interests. It would almost seem that the conditions now are exactly analogous to those existing 11 years ago when the great iron ore beds of the Mesaba range assumed an increasing industrial importance.

Turning to the zinc business, we find that it has only been satisfactory to the mining end of the business. The market for spelter has fluctuated between fairly high limits, but the smelting margin has been so low that money alone was made by the smelting concerns possessing great power and hold on the market. The final decease of Iola as a smelting center after years of a prolonged lingering illness is expected this winter. The tendency is to build new spelter plants in the Illinois coal fields. The sensational finds of carbonate ore in Leadville and the reduction to a commercial basis of the Huff electrostatic separator will, among other factors, ease in the ensuing year the position of the smelters. While it is so cheap to build competitive plants, and small plants can be so efficiently operated in the zinc industry, the idea of a zinc trust is chimerical.

In the lead market restricted production of ore, due to the approaching exhaustion of certain large mines, has unquestionably brought the market into a position much brighter from the viewpoint of the producer. Statistical evidence is wanting, but it is probable that the price for 1911 will average higher than the price for 1910.

Silver has been absorbed by the Far East to quite an extent. There has been of late a generally healthy tone to the silver market and a continuation of this can be legitimately anticipated. It is the one metal save gold—for which, as many an alluring gold mine prospectus has told us, Uncle Sam is an unlimited customer—about which there is little perplexity.

In all the other markets the chief feature is the perplexity and doubt on the part of the consuming public. Could an intelligent despot by his fiat take the stand and say "all metals will be sold as per a certain schedule for a period of five years," and could he make the statement with means to back it up: "I will buy copper in any amount for 13⅞ cents and I will sell copper in any amount at 14 cents," consumption would increase at once. But fortunately no one man possesses the power or the means or the intelligence needed to be such a despot. And the only cause for the perplexed human consumer is to pursue a careful course.

In conclusion, we might say that metals are produced now more efficiently than ever before and that each year civilization finds new uses for them. This is a general encouraging feature. In particular a bright feature is that our crops are unrivaled in size and prices lower. Of course, we will always have social agitation, for the extravagance of the foolish must always be expected and a certain class of people will be discontented and agitate for their alleged rights. Questions that perplex the metal markets to-day will not perplex them next year. Then there will be a new set of troubles. We do not now think that the metal industry is in such a bad way. For when things are bad, they are never really as bad as people imagine. Just one thing is certain. Those in the metal business who work hard and intelligently at their own affairs will have little time for morbid introspection and what philosophizing they do will be pleasant.

Iron and Steel in 1910.

The story of what occurred in 1910 in the iron and steel trade is soon told. Why it occurred is, as our friend Kipling would tell us, "another story," and, we might add, a much longer story, if, indeed, it can be told at all. The year opened with the United States producing at the rate of about 31,000,000 tons of pig iron a year, and finished products in proportion. Three years, 1906, 1907 and 1909, had shown a production between 25,000,000 tons and 26,000,000 tons, but no calendar year had crossed the 26,000,000-ton mark. From April, 1909, to February, 1910, there was a steady gain in production. After February production declined, somewhat rapidly during the second quarter, very slowly during the third quarter, and very rapidly during the fourth, so that the rate at the end of the year was not much above 20,000,000 tons, in terms of pig iron.

Prices of pig iron declined almost steadily, but very slowly, through the year. Prices of finished steel products declined rather easily from early in January until the forepart of August, total declines in finished products outside of rails and tin plates averaging not far from $3 a ton. The prices of rails and tin plates did not decline during the year. In August the nominal prices were shaded in some cases, whereas at the close of the year the same nominal prices prevailed and were not being shaded. The circumstantial evidence is that a "controlled market" came into being.

The steel mills entered the year with a very large, though not unprecedented, volume of business on their books. Uniformly shipments exceeded new bookings, and month by month the accumulation decreased until by the close of the year the

mills were shipping almost wholly on current orders. A natural sequence to the continued expiration of old contracts was an increase, beginning in August, in new bookings, but as the new buyers thus appearing were merely those whose contracts had expired the actual demand for material did not increase thereby

We estimate the production of pig iron in 1910 at 27,150,000 ± 150,000 gross tons, comparing with previous years as follows:

1902, 17,821,307	1906, 25,307,191
1903, 18,009,252	1907, 25,781,361
1904, 16,497,033	1908, 15,936,018
1905, 22,992,380	1909, 25,795,471

1910, 27,150,000 ± 150,000.

The calendar year is altogether too long a period to show the real vicissitudes of pig iron production. While neither 1906 nor 1907 showed as much as 26,000,000 tons of pig iron produced, the twelvemonth ended June 30, 1907, showed 26,200,000 tons. The calendar year 1908 fell fairly well in the trough of reduced production, but 1909 and 1910 fell one upon the ascent and the other upon the descent of the curve. Maximum production was in February, 1910, and the twelvemonth of greatest production was that ending July 31, 1910, it covering a production of no less than 29,800,000 tons.

The explanation of why a year opening with such promise broke previous production records and yet constituted on the whole a bitter disappointment would be written differently by different observers, according to their viewpoint. One, full of old time ideas of the importance of railroad buying, would attribute the recession in buying to the refusal of the government to allow the railroads to advance their freight rates, at least without the serious consideration of the Interstate Commerce Commission, which is still in progress. It was early in April that it became apparent that the rate advances could not be made without a contest, and about that time railroad orders became light Another, accustomed to regard political developments as exerting one of the most serious influences upon business, would cite the change in political sentiment in the country as the cause of the slowing down in the industry A third, irrevocably given to the open-market method of transacting business, would ascribe the reversal in the iron industry to the recently devised system of price control, being responsible, first, for an unhealthy boom after the great price break of February, 1909, following a period of 15 months' price control during a period of slack demand, and, second, for the abstention of buyers in the closing months of 1910 when price control was again in evidence.

Probably all these causes contributed toward making 1910 what it was. In the upward movement in the second half of 1909, after prices had found the lowest level for nine years, producers frankly stated they were advancing prices to choke off buying, but the more they asserted that the more buyers were encouraged to buy, in some cases making contracts for forward delivery, in other cases actually taking out the material and adding to their stocks. To support the position in which the iron and steel industry found itself at the opening of 1910 there was required a general and marked improvement in the general consumptive capacity of the country. This was experienced, perhaps, to a limited extent during the early months of the year, but when railroad buying was largely shut off the industry had to run on its accumulation of railroad orders. The accumulation decreased rapidly toward the close of the year, while the general requirements of the country probably underwent some diminution also.

At the close of the year the paramount question was: Is the liquidation still running its course, or, as the revival a year earlier went too far, has the recession also gone too far? A very few months of the new year will determine, and we think it will determine in favor of the latter alternative. To the close-range view the matter of prices of finished steel products is an element, there being a sentiment widely prevalent among buyers and not absent entirely from the ranks of producers that finished steel prices should be reduced. This we consider a detail which the more vital elements will settle. If business is in position to revive in a marked way, the present system of price control will yield. If there is no hope for radical improvement in any case, or if there is good promise of improvement without price reduction, prices will hold.

Western Steam Coal.

Chemists and engineers long have recognized important differences between Eastern and Western coals. These differences are so marked that methods of proximate analysis devised for Eastern coal are not always suited to Western coal, or at least the results of such analysis on Western coal are not capable of the same interpretation. Again, methods of boiler firing and types of boiler construction which are useful and efficient elsewhere have been found deficient in the West. In many sections of the West the coal available for steam purposes is a lignite containing 40 per cent to 48 per cent fixed carbon, 32 per cent to 37 per cent volatile matter, 5 per cent to 23 per cent moisture and 2 per cent to 14 per cent ash. The calorific value is about 11,000 b.t.u. The high percentage of volatile matter is the distinctive feature of this coal. If consumed under ordinary conditions, this same volatile matter, which contains about 17 per cent of the calorific value, is also the cause of the greatest loss in efficiency, so that even though the coal may have the same calorific value as an Eastern coal it will give much lower efficiency when burned under the same conditions. This is due to the fact that the volatile matter is easily driven off from a freshly fired charge of coal at a temperature low enough to distil it, but not sufficiently high to bring about combustion. The gases thus escape from the zone of action unburned and reduce the efficiency of the operation.

These peculiar conditions have caused Western engineers to devise special furnaces and means of firing, whereby the efficiency of Western coal may be greatly increased over former practice, and made to approximate closely that of the East. A thick fuel bed, mechanical stoking and forced draft, with preheated secondary air, have been found valuable in burning this fuel. This idea has been applied not only to boilers for steam-raising purposes, but also to metallurgical furnaces, such as roasters and reverberatory smelters. Forced draft has been applied in various ways and improvement has been marked in this line. Using steam direct for forcing the draft has consumed about 18 per cent of the steam raised. When using fans driven by steam engines the steam consumption has been reduced to about 12 per cent, while more recent devices embodying small turbines and fans are credited with a consumption of

only 4 per cent to 6 per cent of the steam raised. A difference of opinion exists as to the value of injecting steam with the draft, although this probably refers more to the quantity used and manner of application than to its total elimination. Another interesting phase of the use of Western coal is the improvement which is noted where coal is bought on specification, with a graded schedule of prices for fuel value. One notable instance is that of an institution using slack of a certain specification. As long as little attention was paid to the grade of the coal it was of poor quality and the efficiency of the steam plant was low. As soon, however, as chemical control was instituted, the grade of the fuel promptly rose with corresponding efficiency in the power plant.

National Faults in Manufacturing.

The American people have been styled often by Europeans "a nation of shopkeepers." This remark certainly has a modicum of truth in it. While it is perfectly true that we display at times an intense striving for the immaterial and for the spiritual, yet all of us are more or less of the time quite nimble dollar-chasers. Possibly in our desire for acquiring money we overreach the mark, but we are not the only ones. American visitors in England and continental Europe have remarked on a certain convulsive grasping after the last farthing, centime, or pfennig that suggests that Europeans criticism of our weakness for money is founded on the critics' ability to see the mote in the American eye and to disregard calmly at the same time the Continental beam.

However that may be, concisely stated, Americans are business people and are fond of the business game. Now this tendency is often too intense and the vision focused so closely and so continuously on the intermediate object in view as to make the American business man myopic. A quite prominent manufacturer of special steels was discussing recently an intricate phase of the working of the automobile with a metallurgical engineer in a New York club. The former showed such an intellectual grasp of the broad principles of a business foreign to his own that the latter ventured to ask him whether he was considering the introduction of the electric-steel furnace in one of his plants. Quite to the surprise of the metallurgist this man, broadly intelligent outside of his own field, was only narrowly intelligent in his own field. He confessed never to have heard of the electric-steel furnace as being anything more than a toy. Now, for years this journal has been reciting the potential advantages of electric steel making and also the practical achievements of the several electric-steel furnaces in this country and in Europe. Recently also the general technical press has not been backward in following the lead of METALLURGICAL AND CHEMICAL ENGINEERING. Furthermore, the largest steel company in the world has demonstrated the commercial utility of the electric-steel furnace. And yet this manufacturer of special steels, so keen in looking at the automobile business in a philosophical manner, had centered his mind so closely to the actual carrying out of his operations from day to day that he could not see that his concern would be saved from the ever vexatious labor question by the introduction of the electric-steel furnace. Verily, as the sage remarked, "some people cannot see more than six inches beyond their nose."

Our metallurgical friend had the tact to awaken the manufacturer slowly and also the sagacity to test the sincerity of his remarks. For some people assume an attitude of ignorance in order to gain expert knowledge without paying for it. The steel manufacturer simply had the inability to see the architectonics of his manufacturing structure. He had exaggerated the value of salesmanship (that side of the subject he understood perfectly) and he laid too much stress on the value of practical and resourceful foremen. These details are, of course, well-nigh essential and always exceedingly important. But, but, and but, they are not the sum total of high-class manufacturing.

The three prime elements of successful manufacturing are, first, to have a clear view of the commercial and financial aspects of the situation; secondly, to pay close attention to the practical and engineering sides of the subject; and thirdly, to have ever in mind an imaginative, yet reason-tempered conception of the many possible applications of physics, chemistry, and other pure sciences to the special class of manufacturing on hand. This analysis is but a rough one, just as an analysis of concrete as an aggregate of cement, sand and crushed stone without further chemical or physical determinations is a rough, general analysis. But though rough, we believe it sufficient and practical for pragmatic purposes. It is accordingly only too evident that the American manufacturers exaggerate the importance of the engineering and strictly commercial sides of business affairs.

To attain perfect sanity in an individual a proper balance of the physical and mental and emotional energies is necessary. Without these co-ordinated properly, a departure from the highest normality is made, varying, according to the disharmony of the forces and their magnitude, from everyday folly and extraordinary silliness, till eccentricity, incompetence and finally insanity is reached. Just so in the practical world of business to attain the fullest measure of success there must be a co-ordinated balance of the three different components. The average American manufacturer has overdeveloped two sides and neglected the third side. When he turns to the use of the projected imagination he often shows a queer form of activity as, for instance, the desire to get gold out of seawater, or the willingness of a banker to capitalize an attempt of making silver from lead. Probably his imagination is so seldom used that when it is used it is stimulated to such an extraordinary degree that his otherwise sound judgment is paralyzed.

While American manufacturers do not pay attention enough to the possibilities of applied science, German manufacturers perhaps pay too much attention to pure science. In many large German works a few less doctors of philosophy and a few more hard-headed Yankees would improve things. It is not necessary for us to criticise much the policy of our Teutonic confrères. They have developed their business to meet the actual conditions confronting them. But they do not know all the possibilities of the game any more than do our own manufacturers. Both have characteristic points of weakness. Each country can learn and, we are glad to say, is learning from the other. For there is no tariff on the interchange of ideas. It is grandly true that in the realm of ideas reciprocity expands freely by laws made not by man but by God.

Perkin Medal

At a meeting of the Perkin Medal Committee, consisting of representatives of the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society, the Perkin medal for the present year was awarded to Mr Chas. M. Hall, of Niagara Falls, for his pioneer work in the aluminium industry

The formal presentation of the Perkin medal to Mr. Hall will be made at the next meeting of the New York Section of the Society of Chemical Industry.

The Western Metallurgical Field

British Columbia.

The investigation of the zinc resources of this province, which began about five years ago, is still being prosecuted by the Dominion government under the direction of Dr. Eugene Haanel, director of the Department of Mines. Experiments are now being conducted at McGill University for the purpose of solving some of the fundamental difficulties in the reduction of low-grade zinc ores by electric methods. When this is accomplished it is proposed to erect an experimental plant at Nelson, where tests will be conducted on a larger scale. Electric smelting is only one of the processes being investigated by the commission having the matter in charge. Ore has been shipped to the university from a number of the properties having zinc ore, and the possibilities of treatment are being thoroughly investigated by the commission, which includes Dr. A. Stansfield and Mr. W. R. Ingalls.

The new Portland Canal district was recently visited by the provincial mineralogist for the purpose of issuing a preliminary official report. As yet there is but one producing property, although a number of owners are prosecuting development work. The Portland Canal Mining Company, of Victoria, has erected a 75-ton concentrator, which is designed to be one unit of a larger mill if the developments warrant. Ore is conveyed by aerial tramway, and concentrated about 4 into 1, with a clean separation and good saving

Alaska.

Sluicing and dredging in the Nome district were brought to a close earlier last year than usual, and as a result of this and other causes the output for 1910 will be from $250,000 to $500,000 less than for 1909. The production from the Seward Peninsula as shown by the Customs Office was about $3,500,000 for 1910, as compared with slightly over $4,000,000 for 1909.

Other conditions contributing to the reduced output were the early closing of the navigation season, a late spring, and the partial exhaustion of some of the pay streaks. Nineteen dredges were in operation during the year, but of this number 16 closed unusually early.

About 10 new dredges were built in the peninsula last season, and these, together with others to be built next year, will make a total of 24, most of which will commence operations as soon as possible next spring.

Mexico.

The recent demonstration against the government has not amounted to as much as United States papers intimated, and personal advices from the southern republic indicate that the revolution has never been serious. In view of the large metallurgical interests controlled by American capital in Mexico, it is gratifying to know that there is no immediate prospect of a disturbance there. There can be no doubt that the growing supremacy of American capital and industry in Mexico is distasteful to a part of the Mexican population. On the other hand, the officials have recognized the value of the foreign element and have encouraged their work by offering favorable concessions.

The Pacific Smelting & Mining Company, a New York concern is laying plans to become a strong custom smelting company in northern Mexico. It has recently purchased a concession for erection and operation of a custom smelter at Mazatlan, Sinaloa. Two other plants are under construction by this company, viz., one each at Fundicion and Guaymas, Sonora, and both will be in operation early this year. With three plants drawing from the mining districts of Sonora, Jalisco, Tepic, Sinaloa and Durango the company will be a strong factor in Mexican metallurgy.

The Cananea Consolidated Copper Company, of Cananea, Sonora, will treat the concentrates produced by the new mill of the Miami Copper Company, Arizona, delivery to begin about January, 1911. Special bin and furnace equipment have been installed to accommodate this increased supply of ore

Nevada.

One of the less prominent districts of Nevada is known as Seven Troughs. Although not so well known as some others, it produces some rich ore and has some mills that are reported to be doing good work. The Seven Troughs Coalition Mines Company has a 10-stamp amalgamation-concentration mill in which the low-grade ore is treated with a good recovery. The tailing loss is expected to be recovered by cyanidation in the future, and tailings are being impounded in anticipation of installing that process.

The most pretentious plant in the district is that of the Darby Ore Reduction Company, at Mazuma. Ten stamps reduce the ore to pulp, which then passes over plates, and later, over Deister concentrating tables. The table tailings are classified into sand and slime by Dorr classifier, the sand being ground in tube mill and amalgamated before joining the slime for cyanide treatment. The latter process is carried out by agitation in tanks and filtration on Oliver continuous filters

One of the newest districts in Nevada is known as Jarbidge, situated in the northern part of the State. The developments have proceeded far enough to warrant one company, the Pavlak, erecting a cyanide mill, which is now under construction. In order to aid the pioneer companies as much as possible the county commissioners have ordered a road built into the camp, making good connections with the Western Pacific Railway at Deeth.

Utah.

The recent report of the Utah Consolidated Company, which showed such marked discrepancies between the supposed and actual state of the property, was a great surprise. It had been generally supposed that, while the Highland Boy mine was not in the promising condition which it once enjoyed, it was, nevertheless, in possession of three to five times the reserve which the last report showed. This opinion was supported not only by the previous reports on the property made less than a year ago, but by the fact that the International smelter was built at Tooele ostensibly relying on the ore from the Utah Consolidated for its main supply. At present it appears that the anticipated ore supply will not be forthcoming, and that another year will see the finish of shipments from the Utah Consolidated to the International.

This has necessitated a campaign for other ore supply, and one of the possibilities mentioned is that the Giroux company will ship to Tooele instead of building its own smelter in Nevada. This would involve an adjustment of freight rates over several railroads, but in view of the difficulty in securing a large copper tonnage under present conditions, shipments might be made even if arrangements were not as good as the best. The Tooele plant is our newest and most complete copper smelter, as has been outlined previously in these columns, and favorable rates would undoubtedly be made rather than have it idle wholly or in part.

In the meantime conditions at the Garfield plant of the American Smelters Securities Company are more flourishing than before, both as to capacity and ore supply. A sixth reverberatory furnace has been built, and the converting capacity has been increased by additional Peirce-Smith basic lined converters. All of the reverberatories and three out of four

blast furnaces are running. These changes have been made necessary by the increased output of the mills of the Utah Copper Company at Garfield, and the combination bids fair to make Garfield the greatest copper concentrating and smelter center in the country.

Experiments looking toward improved zinc concentration at the Daly-Judge mill, at Park City, have been under way for some time. Retreatment of the mill slime on tables is being tested, and in addition to this tests are being made with the MacQuisten tube in an endeavor to improve the quality of the zinc product

Colorado.

The recent completion of the Newhouse tunnel in the Gilpin district, and the connection of the Roosevelt tunnel at Cripple Creek with the El Paso shaft, are regarded as two events which will have an important bearing on the immediate future of Colorado mining and metallurgy. The Newhouse tunnel is 21,968 ft. long, and provides drainage and transportation facilities for a mineralized area which has been very productive and which still contains large quantities of ore suitable for preliminary concentration or final reduction in the mills of the district. The Roosevelt tunnel has been driven about 16,000 ft. and has 14,000 ft. yet to be driven before its objective point is reached. In the latter portion of its course it will effect the drainage of the area which has a common water level and which contains numerous gold-bearing veins. In view of the undoubted efficacy of the cyanide process in the treatment of these ores the completion of the tunnel will undoubtedly open the way for the erection of new mills and the creation of a new metallurgical era for Cripple Creek.

The Non-Ferrous Metal Market.

A general state of depression has pervaded the market for non-ferrous metals, and the feeling regarding the future is divided. Interest in copper is centering about the efforts being made to realize an agreement among the large producers, for it is believed that some plan of co-operation will be necessary for the good of the industry. Prices for all metals have been almost stationary or reduced since our last monthly report.

Copper.—Although the producers are endeavoring to hold the market at the advance which it experienced in November it is apparent that purchasers are not inclined to buy at the high figure, and as a consequence the prices are being shaded for most transactions. Lake copper has been offered freely at 12¾ cents to 12⅞ cents, and electrolytic at 12.5 cents to 12.6 cents.

Lead.—Considerable business has been transacted on contracts, but the market is quiet. New York quotations are 4.5 cents, and St. Louis 4.35 cents to 4.37½ cents.

Tin.—The advance in the foreign market and the apparent control of that market by the bull interests has resulted in a slight advance at home, although the volume of business is light.

December tin was quoted in New York at 38½ cents.

Spelter.—In common with the other metals, spelter has been depressed and has lost its favorable position reported last month. Inquiries for metal are few and competition among the sellers has been keen. Prices have steadily declined from week to week, and the metal is quoted at 5.47½ cents to 5.52½ cents, St. Louis, and 5.62½ cents to 5.67½ cents, New York.

Other Metals.—Aluminium has remained quiet at 22½ cents to 23 cents for No. 1 ingots. Antimony is unchanged at 7 cents to 7⅝ cents for various brands. There is a fair business in quicksilver at $42 per 75-lb. flask at New York and $41.50 at San Francisco.

Camp Bird, Ltd., Ouray, Colo., operated its mill 30 days in September last, treating 6690 tons of ore and recovering 6281 oz. of bullion and approximately 761 tons of concentrates. The monthly report for September shows about $85,703 applicable to profit.

The Iron and Steel Market.

The stagnation which developed in the iron and steel market late in November was intensified in December, and further and sharp reductions in output were made in the steel industry. In the merchant furnace trade some curtailment of output occurred, but obviously not enough to take up the slack, and December witnessed an increase in the accumulation of pig iron stocks which has been practically continuous all year

Following the general meeting of steel manufacturers in New York on Nov. 30, a series of meetings of producers in individual lines occurred, the meetings covering in turn rails, structural shapes, sheets, tin plates, merchant bars and plates. These meetings were held not for the purpose of fostering a definite price agreement, but rather for the purpose of interchanging information and encouraging manufacturers to curtail output.

Many plants closed entirely for the last fortnight in the year, and others closed for the last week. Altogether there has been the greatest holiday slowing down in the industry that has occurred for many years. At the close of 1903 and again at the close of 1907 there was sharp restriction of output, but in those cases the definite movement began in October or earlier, while last month the curtailment was so great, following the very gradual curtailing tendency which had been in evidence, that it partook of the nature of a separate movement.

Prices of finished steel products have been very steady, there being less shading of openly recognized prices, as a rule, than in October or November. The precise cause of the steadiness cannot be stated absolutely, it being apparently the resultant of the meetings which had been held in the trade and the complete indifference of buyers, which removed quite largely the temptation to invite business by cutting prices.

In the first week or two of December production of pig iron was at the rate of about 22,500,000 tons annually, contrasting with a maximum rate of 31,600,000 tons, reached in February last. Considering, however, that the merchant furnaces curtailed their output much less than the steel interests, it may be estimated that the steel industry in the forepart of December was running at only two-thirds the February gait. Toward the close of December the reduction was so sharp that no precise estimate can be made. The prospects of resumption are uncertain. Some plants are definitely scheduled to resume operations the first week in January, but in other cases there will probably be a wait for some indications of the character of business in the new year.

Among buyers the impression is widespread that finished steel prices will be lower within a few weeks, either through an orderly reduction or an open break. It has been generally reported that the chief desire of the steel interests was to hold prices until after the turn of the year, so that if lower prices were to be sanctioned the point could be reached by an impressive and formal reduction.

Pig Iron.

The market has continued quiet, without any important changes in prices. Demand has been light and it is evident that melters aim to have their obligations at a minimum. Several meetings of blast furnace interests have been held, chiefly at Philadelphia and Cleveland, looking to the exerting of influences for a better market, but no definite conclusions have been reached. There was a particular desire to bring about better conditions in the basic pig iron market in the Central West, but the work so far seems to have been fruitless. Basic iron had, however, firmed up slightly before the movement was definitely proposed, and is quotable at $13.50, valley furnaces, while as low as $13 was done in November. A sale of about 2000 tons of Bessemer was made for December delivery at $15, valley, the regular market since late in August Foundry iron has gone at $13.75 to $14, valley, and one interest has been ready to do $13.50 on particularly attractive business

Steel.

The movement for regular prices on billets and sheet bars was in the first flush at the time last report was written, and since then the understanding has had to yield a trifle at two points. At the time of last report Bessemer billets were quoted at $23 and sheet bars at $24.50, both f. o. b. Pittsburg, plus freight from Pittsburg to destination. The idea now is $23 for billets and $24 for sheet bars, at Pittsburg or Youngstown, with freight to destination from the nearer point. A concession of 50 cents in the spread between billets and sheet bars was made and the separate basing point at Youngstown was re-established. However, the open-hearth steel market seems to have firmed up to the Bessemer level, whereas in November some sales were made at less than the going price for Bessemer. Rods are slightly lower, at $27.50, Pittsburg.

Finished Steel.

The following prices, f. o. b. Pittsburg, except where otherwise stated, are being quite generally observed:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3-in. to 15-in., zees, and angles 2 x 3 and larger.

Steel bars, 1.40 cents, base; iron bars, 1.35 to 1.40 cents, Pittsburg; 1.30 cents, Cleveland; 1.30 to 1.35 cents, Chicago.

Wire nails, $1.70, base, per keg; plain wire, 1.50 cents; painted barb wire, 1.70 cents; galvanized barb wire, 2 cents.

Black sheets, 28-gage, 2.20 cents; galvanized, 3.20 cents; painted corrugated roofing, $1.55 per square; galvanized, $2.75.

Tin plates, $3.60 for 100-lb. cokes.

CORRESPONDENCE

Presidency of American Electrochemical Society.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In view of the nomination of officers of the American Electrochemical Society for the year 1911, we, the undersigned, suggest for the office of the president Dr. William R. Whitney.

The abilities of Dr. Whitney as an electrochemist are too well known to need to be stated here. His recognition as a chemist is best shown by the fact that he has been elected to the presidency of the American Chemical Society. His able services as a president of a national society have been shown by his conscientious work as president of the American Chemical Society.

He was one of the two prominent candidates for president at the election of the American Electrochemical Society last year, but declined on the ground that he was then president of the Chemical Society; the reason for his declination last year, therefore, now no longer exists.

We feel that with Dr. Whitney as president of the American Electrochemical Society the interests of that society will be in very able and conscientious hands, and that the duties of that office will be well taken care of.

F. G. Acheson, L. H. Baekeland, H. R. Carveth, Francis A. J. FitzGerald, Carl Hering, F. A. Lidbury, F. R. Taylor.

Heat Loss Through Furnace Walls.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—My attention has been called, by Mr. F. T. Snyder, to a typographical error in formula (4), page 218, Vol. XIV of the *Trans.* Amer. Electrochem. So. in my paper on "Heat Conductances Through Walls of Furnaces," and copied with the same error in the advanced copies of my paper on "Method for Determining Thermal Conductivities" read at the recent Chicago meeting, where it is (6) on page 9. The letter "l" occurring twice in that formula should be the numeral "1."

I take this opportunity to also call attention to the fact that the simpler formula (4) in the latter paper given for a sphere, and (2) in the earlier one, is also rigidly correct for any rectangular furnace and not merely for a cubical one.

Philadelphia, Pa. Carl Hering.

Heat Resistance Units.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In your issue for December appears an article by Mr. Carl Hering on "temperature drop a true measure of thermal resistance," which to my mind is much to the point. Not only are Mr. Hering's proposals logical from a scientific standpoint, but they are directly applicable to everyday practice.

I would propose that instead of using the English inch as the unit cube for thermal resistivity and conductivity the centimeter cube be used. This would give to these specific properties of materials a multiple relation to the corresponding C. G. S. figures. I would further propose that the thermal unit for resistance and conductance be named not thermal ohm and thermal mho, but that as abbreviation of these "thohm" and "mhoth" or "themho" be used instead.

Is it not eminently proper for the American Electrochemical Society to study now the question of nomenclature, as the above, and ratings, etc., and attempt to standardize terms that would be accepted by the profession?

Hartford, Conn. Woolsey McA. Johnson.

Metallurgy of Zinc.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I have read with interest the editorial in the December number, entitled "Condensation of Zinc Vapor," in which the work of Mr. Hixon, Mr. Johnson and others is spoken of. The general trend of the editorial seems to be an encouragement of improving the metallurgy of zinc, the "unusual" metal. It also infers that the present metallurgy of zinc is primitive and not in keeping with twentieth century methods. This attitude is certainly correct.

The serious error in the editorial, however, is the failure to recognize the advance of the dry-chloridization of complex zinc ores, originally invented in this country by Messrs. Chas. E. Baker and Arthur W. Burwell, prefaced in Norway by Alf-Sindig Larsson, and in England by Swinburne and Ashcroft. The process has been further improved in this country by Mr. Chas. E. Baker and others at various times connected with him, such as Mr. William Koehler, once associated with Dr. Hoepfner, Chas. A. Diehl and J. L. Malm. Mr. Malm is at present manager of a works in Georgetown, Col., known as "The Western Metals Co.," licensed under the Baker-Burwell patents by Mr. Baker, owner of the patent rights for Colorado.

It is not my purpose in this letter to describe the process, as such has already been done, both in your journal and in a paper by Mr. Baker before the American Electrochemical Society. It is my purpose to call your attention and that of your readers to the error in your editorial, that while you congratulate Mr. Hixon and Mr. Johnson you omit the Baker-Burwell process, which, I believe, is much nearer commercial fulfillment than any.

The process is particularly adapted to ores carrying zinc-lead-iron sulphides with small quantities of copper, silver and gold, and recovers all the values. As such ores lie on dumps in the West in unlimited quantities, the value of the process as a factor in the "conservation of our natural resources" is of great magnitude. Either spelter or zinc and zinc-lead pigments may be obtained, as desired.

Chicago, Ill. Howard Baker.

[As the editorial in question was entitled "The Condensation of Zinc Vapor," its scope was clearly restricted to smelting processes. That, even with greatly improved smelting processes, there would remain a distinct profitable field for a chlorination process, when properly reduced to practice, there can be no doubt.—Editor.]

The Training of Chemical Engineers.*

BY M. C. WHITAKER.

Much attention is properly being centered upon the education of men in the scientific development and management of agricultural production, and the Government not only gives direct financial support to a large number of schools for such training, but also maintains a Department of Agriculture, the function of which is largely educational.

The annual production of the chemical industries is almost equal in value to the agricultural production of the country, and it is, therefore, proper to compare the facilities now being provided and the means now being adopted for the education of men to develop and manage the widely diversified units of chemical manufacture with the methods and facilities in other educational fields. Obviously the general educational and technical equipment of men to develop and direct processes or works depending upon the applications of chemistry, and usually involving a knowledge of power and mechanical and electrical machinery, is much greater than the technical equipment required for men in agricultural production. While agricultural education has grown and developed at a rate commensurate with the producing power of that industry, and has been fostered by private, state and federal funds, education in the equally important and essential field of chemical manufacture has been, in some institutions, entirely ignored, in others only indifferently provided, and in no single case supported and equipped on a basis comparable with the wealth-producing power of the industry.

Efficient production and the economic management of our manufacturing plants are essential features to our commercial development, and it is in this field that the greatest results are to be attained in the conservation of our natural resources. The accomplishment of these results obviously depends upon the application of scientific knowledge to the solution of all problems, both great and small, in the development, the direction and the management of our factories.

The chemical industries very properly look to the schools for men qualified to produce these results, to improve their production and manage their works. The best evidence that more and better trained men are needed in the chemical industries is the fact that this Institute of Chemical Engineers, composed of men of experience and men now engaged in the direction of great industries, meets twice a year and devotes a large portion of its time to the study of the problem of how to obtain trained men to assist them in the development and administration of their work.

Every employer of chemical talent knows that at best the young graduate is only a vague "prospect" and that much has been left undone in his training.

The problem of proper training and correct methods of teaching chemical engineering is one which must be carefully analyzed and studied from its very foundation and is not one to be remedied by a few trivial changes in existing curricula—by dropping this and substituting that. The subject must be opened up and examined to its foundation, and if it is found that the present structure is built on sand we should put some "concrete" under it.

Necessarily the problem must be solved by the co-operation of the teachers with the employer of chemically trained men, and the writer, having served a term of practically 10 years in each of the above fields, begs to submit a few observations and conclusions for the consideration of the institute committee.

The chemical industries require for their development and management two classes of men of somewhat different training, natural qualifications and range of service.

First.—Research chemists who are qualified naturally and by training to originate and to develop in new fields. The training of the research man is more advanced, or supplements that of the chemical engineer, and the research results furnish the foundations for the industries which the chemical engineer organizes and administers. In other words, the research chemist works in the new and undeveloped fields of chemical knowledge and produces results calculated to contribute new knowledge and new arts.

Second.—The *chemical engineer* works in the organization, operation and management of existing or proposed processes with a view to building up a successful manufacturing industry. He uses his classified knowledge of chemistry and the allied engineering branches in developing, perfecting, organizing and administering a plant or a process for the production of a marketable and useful product at a profit to the investor.

Such a field as this requires a man of knowledge, originality and resourcefulness. His fundamental training in chemistry, physics, mathematics, etc., must be thorough and must be combined with a natural engineering inclination and an acquired knowledge of engineering methods and appliances. His originality and imaginative capacity seem to me to be essential and a combination result of natural qualifications and training. Resourcefulness, however, comes as a result of experience in application and the ability to adapt a wide range of methods and practices to new conditions. A man with the natural qualifications, supplemented by the proper training, may become a successful chemical engineer.

The success of the research chemist is measured by his scientific achievements, whereas the success of the chemical engineer is measured by his commercial results. While their work bears a certain mutual and cyclic relation it must be apparent that their natural and educational equipment may be entirely different.

The development of specialized training in certain selected lines of manufacture is not a proper function for an engineering school, but should be left to the trade schools or to the industries themselves. Specialized research, on the other hand, should be carefully eliminated from the regular engineering training and provided for in graduate courses.

For convenience in discussion I have divided training subjects for the chemical engineer into three general classes: fundamental, associated and supplementary. No attempt is made to discuss the details of the curriculum, the general purpose being to get broad subdivisions as a starting point. The question of arrangement of subjects into a curriculum and the proper allotment of time for the course depends upon what is finally determined as proper training for the men under discussion:

Chemistry Physics Mathematics Allied subjects	Fundamental training.
Electrical engineering Mechanical engineering Civil engineering General engineering Business economics	Associated training.
Study of the *applications* of the fundamental and associated training in laboratories equipped with the "tools of the trade" and with working plants. Training by "contact" and by "example" in a laboratory managed as an approved business, in the principles and practices of efficient organization and administration.	Supplementary training.

Fundamental Training.

The training in the fundamental subjects need not be materially changed except perhaps more attention should be given

*A paper read before the American Institute of Chemical Engineers at the meeting in New York City on December 7.

to laboratory facilities and to the selection of instructors for the fundamental work. Profound scholarship and a record for original research work are too often made the sole determining factor in the selection of teachers for these subjects.

Research qualifications are of secondary importance in the selection of men to give the fundamental training to our chemical engineers. Such teachers might better be chosen from men physically, temperamentally and educationally qualified to present their subjects clearly, logically and enthusiastically to the student, and to present them in such a way as to quicken and hold his interest.

Too often our undergraduates become muddled, discouraged and even give up a subject because of its dry, disconnected presentation or because the instructor "talks over their heads" or lacks the personality and the power to interest them. Research instructors are often permitted to emphasize their "hobbies" or teach their special experiences without regard to the purpose of the course and its relation to the fundamental training. Our instructors should always be chosen to fit the course requirements and should not be permitted to divert the work into special fields not contemplated in the basic formulation.

The training in the fundamental branches should be more thorough and carefully proportioned than at present, keeping in mind that the chemical engineer's knowledge must be classified, complete and accurate in the broad principles and not statistical, encyclopediac or of the class of deep profundity required for research in new and original fields. The chemical engineer's knowledge is to be used as a basic "tool" of his trade and may be given as such without detracting one iota from its value as a mental training.

Associated Training.

Associated training is for the purpose of placing in the hands of the chemical engineer a working knowledge of the other engineering branches so that he may utilize them in the development and management of his particular problem. Courses designed primarily as foundational for future advanced study in the same field are not likely to be proper courses for this associated training. Our associated courses should be broad and general, and need not carry the student beyond the foundational principles of the subjects with a working knowledge of the existing design, construction, materials, operation and use of engineering appliances.

The existing methods of teaching by lectures, demonstrations and laboratory work, combined with a liberal use of text and reference books is probably the most effective way of reaching the fundamental and the associated subjects.

Assuming that the student now has a thorough fundamental foundation in the basic principles of his profession and a working knowledge of the associated engineering branches schools should be equipped for *directed* study of their *applications* to the problems of manufacture and production. This applied study is classed as "supplementary training."

In the education of teachers, in the education of doctors, in the study of agriculture, and in the education of all other engineers except chemical engineers, training in the *application* of the respective fundamentals, under the direction and with the assistance of experienced teachers, is considered essential. Medical schools without hospitals, teachers' colleges without practice schools, schools of agriculture without experimental farms, or engineering schools without laboratories and shops would hardly be dignified by classification as schools. Such institutions invariably invest large proportions of their resources in facilities for demonstrating the proper practical applications of the fundamental theories.

Is there any reason why chemical engineering should not and could not be taught by the tried methods of the normal schools, the medical schools, the agricultural schools and other engineering schools? Only with proper chemical engineering laboratories, equipped with real working models of standard appliances to illustrate the basic applications of the industries and a proper curriculum arrangement to permit the student to study and use these appliances and the principles involved in their application will we be on a basis comparable with the methods and the facilities now existing in other educational fields.

But we can accomplish still more with proper laboratory facilities. We can so construct and administer these laboratories that the student will be surrounded by an "atmosphere" of manufacturing and business efficiency, where he will learn by example, by daily contact, by "attrition," the modern methods and practices of office and works management. Chemical engineers should have this knowledge. It is fundamental to the proper fulfilment of their job. Instances where failure has been converted into success and industries saved by a knowlege of proper administrative practices, and the application of business and works efficiency methods, are too numerous to mention. Refinements in chemical processes and ingenious mechanical devices are all ineffective and often useless if inefficient organization and management are permitted to absorb the earnings. Successful chemical manufacturing is based upon three general principles: First, chemical and physical facts; second, mechanical applications; third, organization and management.

The first principle is definite and basic, and is, therefore, not capable of variation, but the second and third are variable and may be far from perfection. The greatest opportunity for increased profit therefore lies in the possibility of mechanical improvements and improvements in the efficiency of the organization. Instruction is provided in a greater or less degree in existing schools in the two first principles mentioned, but the third equally important and fundamental principle is not even undertaken.

The "contact" system which I am urging for adoption as a method for teaching the principles and the applications of chemistry to manufacture, and for imparting a knowledge of the principles and practices of efficiency in organization, management and administration, are the same as the methods which have always obtained, by force of circumstances, in country schools, where the small children in the A, B, C classes unconsciously absorb geography, history, arithmetic, etc., by hearing, seeing and being in the "atmosphere" of the more advanced work. This method can be adopted in the training of chemical engineers and can be made infinitely more effective than in the country school.

The "contact" or "frictional" method would require a laboratory plant larger and more comprehensive than anything yet established, and would have to be managed on somewhat different lines than those now obtaining in engineering schools.

If it is proposed to direct the student in the study of the applications of fundamentals of factories, factory appliances, factory processes and manufacturing business by the "frictional" method, the laboratory should be equipped, organized and administered as a live manufacturing proposition. There should be ground space enough to permit of the erection of one or more complete operating plants in addition to the general laboratories for the study of great applications, such as distillation, evaporation, filtration, wet reactions, high temperature oxidation, electrochemistry, etc.

The complete plants should be chosen from some of the important chemical processes of great common interest, which at the same time illustrate a wide diversity of chemical principles, appliances and products—for example, the manufacture of gas.

These plants should be designed and built from the foundation to the roof so that the structure itself, materials used, equipment and arrangement would illustrate the best known practice and permit of the most economical management and operation. Every structural detail should be provided with a view to educating by example. Office appliances, office sys-

tems and facilities should be provided, and the equipment include books, trade journals, patent literature, samples, advertising matter, house organs, etc., of the business. Conference facilities should be provided in each laboratory for lectures, discussions, demonstrations of the chemical, mechanical and business features of the process. All such work should be conducted in the special laboratories, and the student from the time he enters until he leaves should be made to "stumble over" the tools of his art and become "saturated" with the "atmosphere" of the business from the raw material to the market.

The classes should "go the rounds" of the laboratories in convenient sized sections for conferences and for "manning" the work.

In connection with the chemical engineering laboratory there should be a "shop organization" consisting of competent mechanics, helpers and laborers necessary for the upkeep of the plant, and this organization should be so administered that our students would gain actual experience in dealing with workmen. Office experience, correspondence, dictation, filing and recording, fundamental costing, etc., may all become familiar knowledge to the student by having these facilities arranged for his use and by incorporating them in his daily "business."

Technical Advisers.

These laboratories of engineering chemistry should be organized with a system of technical advisers to be selected from representative experts in each typical field of industrial chemistry. These advisers should be specially qualified experts or active works' managers of live concerns. Their connection with the laboratory should not be merely nominal, but should carry with it a responsibility for the proper equipment of the special laboratory of the industry which they represent. They should also give some lectures upon their specialty, not altogether from the chemical standpoint, which presumably can be covered by the regular instructors, but from the business, organization, and relative industrial standpoint.

The personal contact and acquaintance between adviser and student is an obvious mutual benefit. The attitude of the leading industries toward such a chemical engineering school would become one of personal interest instead of one of criticism.

The technical advisory system, if properly initiated and administered, will ensure modern equipment, methods and instruction in chemical engineering, and will give, through the lectures, first hand and accurate general information on the scope and magnitude of the typical industries. Furthermore, it will definitely dispose of the frequent reference to obsolete and antiquated methods and equipment. By relying upon the judgment of these experts and managers of live and paying industries we would avoid the equally dangerous extreme of becoming loaded up with impractical, untried appliances and uncommercial processes.

Industrial Research.

The largest industrial corporations, such as the General Electric Company, the United Gas Improvement Company, the National Electric Lamp Association, etc., have established and are maintaining research laboratories. These laboratories are directed in their chemical and physical research by able and high-priced men, and are equipped and maintained at great expense.

Such laboratories are being established because there are no existing equipments, on a suitable scale, for solving the problems necessary for their industrial advancement. The corporations have not undertaken the burden and expense of academic research from choice, but from necessity. Our scientific institutions have utterly failed to provide the facilities for industrial research, and as a consequence are being "elbowed" aside from their natural position of technical and scientific leadership.

Under existing conditions the small manufacturer who cannot afford to equip and maintain a research laboratory has no prospect of solving his industrial problems. Unless the scientific institutions develop modern chemical engineering laboratories, equipped with the tools of the industries, and place their facilities within the range of the small manufacturers, industrial development will be limited to the few concerns whose means will permit the establishment and maintenance of laboratories for private research.

On the other hand, with such laboratories as I have outlined, equipped with every facility for experimentation on a large scale, factory managers, technical laboratories and individuals could be interested in availing themselves of these facilities and in establishing researches for solving industrial problems.

Frequent reference has been made to the cost of such a laboratory of engineering chemistry as has been outlined We are dealing with a great industry and a great problem, and we should therefore exercise our sense of proportion. It is estimated that the minimum net profit of chemical manufactures is $300,000,000 annually. It is also estimated that the combined investment in chemical engineering equipment in all of the educational institutions of the country is not over $30,000, or one hundredth of 1 per cent of the net profit of the industry for one year. A $30,000 investment, or even one hundred times that, is an absurdly small equipment to be devoted exclusively to instruction in the application of chemistry for such an enormous and profitable industry, not to mention the facilities for research development. If the plan outlined will produce men better qualified to enter this field of industry, and men who can "deliver the goods," the question of cost will never prove an obstacle to the development of such laboratories of chemical engineering.*

Columbia University.

North Dakota Lignite has been tested as a fuel for power purposes at the plant of the Wiliston irrigation project of the United States Reclamation Service. The boiler used was of the standard Stirling water-tube type, consisting of three steam drums, one mud drum and three nests of tubes. The conclustions drawn from the test are that steam can be made with a fuel efficiency of 55 per cent to 58 per cent of the heat in the coal, and no difficulty is experienced in getting the full capacity of the boiler. The results compare favorably with those obtained in the average plant using a good grade of bituminous coal, when the heat available to the boiler is considered. Reduction of the moisture in the coal seems to improve economy, but the tests were not sufficient in number to determine definitely the condition of the fuel and time required for weathering to secure the best results. The steam blower for the ashpit is inefficient and there is no gain in supplying superheated steam to it. A saving in steam with equally good results could probably be obtained by substituting a fan. A complete statement of the test is contained in *Bulletin No. 2* of the Bureau of Mines, Washington, D. C.

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*At the close of this lecture Prof. Whitaker presented a series of concrete questions which will be submitted to all members of the American Institute of Chemical Engineers for a mail vote. These questions are as follows:

(1.) Is the definition of a chemical engineer and the outline of his requirements substantially correct? (2.) Is the prevailing system of education adequate to produce the chemical and engineering talent needed in the industry? (3.) Does the present method of training chemical engineers produce men comparable in their respective qualifications with civil, mining and mechanical engineers? (4) Do factory managers find our graduates qualified to assist in building up and developing the business or are they simply "prospects" who must be "educated" at the expense, in time and money, of the industry employing them? (5.) Are men who receive their experience in the application of the fundamentals in the works where they are employed narrow in their perspective and limited in resourcefulness? (6.) Is there any reason why chemical engineering should not be taught by instruction in the applications of the fundamentals, in a similar manner to other educational systems? (7.) Are the subjects included under "supplementary training" desirable or essential, and is the "frictional" method suggested a proper way to present them? (8.) Would a laboratory of engineering chemistry, equipped and managed as outlined, maintaining a rigid standard for entrance and for graduation, come nearer fulfilling the needs of men in the chemical industries? (9.) Would manufacturers avail themselves of the facilities of such a laboratory for industrial research? (10.) Should such a laboratory be organized and administered, both in its relation to the students and to its work, as a live, up-to-date business?

What Is "Cyanamid"?

By John Woods Beckman.

At the present time there are three different substances that go under the name of "cyanamid." But only one of these has a legitimate right to the name, the two others are illegitimate sisters to this one

The substance with the right to the name is one whose chemical formula is $H_2N{-}CN$, and the name of cyanamid indicates to the chemist that an amid (NH_2) group is combined with a cyanogen (CN) group forming cyan-amid.

Cyanamid is produced in the following manners

Cyanogen chloride is introduced into a solution of ammonia in ether, and cyanamid is formed according to the following formula.

$$CNCl + 2\,H_3N = CN.NH_2 + NH_4Cl.$$

If tio-urea is treated with the oxides of lead or mercury in an aqueous solution, cyanamid is formed, and the reaction can be written in following manner:

$$NH_2CS.NH_2 = CN.NH_2 + H_2S.$$

Also if a cyanamid salt be treated with an acid under special conditions cyanamid will be produced.

Pure cyanamid is a white, crystalline, delinquescent mass, soluble in water, alcohol and ether. It melts at 40° C. One characteristic feature of cyanamid is its tendency to polymerize, forming di-cyandiamid, and even higher products of polymerization, such as melam. The polymerization of cyanamid takes place when a solution of cyanamid is being concentrated or being kept for any length of time. If cyanamid be heated to 150 C. it polymerizes with great violence.

Chemically cyanamid can be either classed as weak acid or as a weak base. As an acid it will form salts with metals, such as silver, calcium and others. Of these salts the calcium salt is the most important. As a base, cyanamid will form a number of salts; with acids these salts as well as most of the metal salts are very unstable.

Cyanamid, as well as a number of its salts, is an unstable chemical complex that very readily transforms if conditions are right in two different directions, polymerizing forming dicyandiamid, or hydrolizing forming urea.

The substance that is most generally known by the name of "cyanamid" is the product obtained in the process of fixing atmospheric nitrogen with calcium carbide. "Cyanamid" is the name most generally used for this compound, but a trade name of "lime-nitrogen" for this product is being used. The only right "lime-nitrogen" has to the name of "cyanamid" is that one of the components of the product is a calcium salt of cyanamid.

The chemical formula for "lime-nitrogen cyanamid" can only be expressed in the following manner.

$$xCaCN_2 + yCaO + zC + uCaC_2.$$

From this formula it will be readily seen that this compound is a heterogeneous substance and of no definite chemical composition, nor do its components appear in the same ratio all the time.

To explain the composition it is necessary to give, in brief, the chemistry of the formation. If calcium carbide is heated in an atmosphere of nitrogen, the carbide combines with the nitrogen with great avidity, according to the following formula:

$$CaC_2 + 2\,N = CaCN_2 + C.$$

This reaction explains the existence of free carbon in the graphitic modification in the "cyanamid," while the free CaO is an unavoidable admixture The carbide used for the fixation of the nitrogen contains a considerable amount of free CaO as diluent, and this naturally will appear in the finished "cyanamid." A small amount of carbide is unavoidable, as it is impossible with the present state of the art to totally bind all the carbide within reasonable time

The chemical nature of "cyanamid" or lime-nitrogen" is a very complex question, and even more so than for the pure cyanamid NH_2CN, on account of the unavoidable CaO. An Italian agricultural chemist by the name of Ulpiani has made some very extensive researches on "lime-nitrogen," and has found that "lime-nitrogen" is permanently changing it chemical structure, even only by storing same and under the influence of ordinary atmospheric conditions. What part the air-slacking of the lime may play in this chemical rearrangement of the chemical structure of "lime-nitrogen," or, more correctly, the calcium salt of cyanamid, is very hard to determine. The calcium cyanamid is of an unstable structure and forms with great ease polymerized as well as hydrolyzed compounds

In treating the "lime-nitrogen" with water, even with small amounts, still more complex phenomena take place, basic as well as acid calcium cyanamid salts are formed, with the free lime present; urea is formed; calcium cyanamid carbonate may be formed, and a quantity of others, including dicyandiamid and free ammonia.

Ulpiani has made very extensive researches regarding the action of "lime-nitrogen" with various different acids and in various solutions, and these are able to spread a good deal of light over an exceedingly complex chemical problem, which is growing more and more important as new uses for "cyanamid" are developed.

These researches of Ulpiani have shed some light on the fertilizing value of "cyanamid," and he proves quite conclusively that it is not cyanamid or dicyandiamid that is the direct fertilizing ingredient in "lime-nitrogen," but it is the result of hydrolization, the urea, which is acted upon by the bacteria, and ultimately transformed into a nitrogen compound that can be absorbed by the plant.

The third product that claims the name of "cyanamid" bears it only as a trade name. "Lime-nitrogen" is mixed with a certain per cent of sodium nitrate and with a low-grade nitrogeneous material as a diluent to regulate the nitrogen contents of the finished product, and is then called "cyanamid." This material is compounded exclusively for the fertilizer trade and is of no chemical importance.

It is apparent from the above that the name of "cyanamid" is one that has lost its original meaning and has come to indicate altogether different substances to different groups of people.

The name "cyanamid" should stand for the chemical compound known by the following formula, H_2CN_2.

"Lime-nitrogen," although a meaningless name, as meaningless as if gypsum were called "lime-sulphur," may remain the name for the product now generally known as "cyanamid," while for the third product a name like "cyanamid compound" or "lime-nitrogen compound" would probably answer, thus clearing up a question of names that is becoming more and more complex the wider the field stretches in which "lime-nitrogen" can be used.

Niagara Falls, Canada.

Copper-Aluminium Alloy.—A new alloy, believed to consist essentially of copper and aluminium, is being placed on the market under the trade name of cupsor by the Cupsor Company, of 50 Church Street, New York City. It has a color like gold, polishes readily and acquires a brilliant luster and is, therefore, especially adapted for jewelry (watch cases, etc.), and all ornamental purposes. It is further claimed to be highly resistive to chemical influence. Not only is it said to be immune from the attack of the ordinary atmosphere, from salt water and salt air, but it is claimed that sulphuric acid, hydrochloric acid, lemon juice, tomato juice, vinegar, etc., have no effect on cupsor. Cupsor can be readily machined and is easily rolled and drawn into wire. "The tensile strength of cupsor in the casting is estimated to be at least 80,000 lb. to the square inch. The No. 20 gage hard-drawn wire broke in a recent test at 138,550 lb. to the square inch." The specific gravity is 8.11.

Thermal Resistance and Conductance; the Thermal Ohm and Thermal Mho.

By CARL HERING.

In the preceding issue of this journal (page 676) the writer showed that the degree of temperature and the watt may be conveniently and correctly used to express and measure thermal resistances and conductances, respectively; also thermal resistivities and conductivities. Moreover, they are true mates, as they are numerically the true reciprocals of each other. Hence we have at our disposal two simple, well-known and named units which can be used in such thermal calculations.

The only objection to them is perhaps best shown by the better known electrical analogy. Suppose the ohm had not been named as a separate and independent unit of electrical resistance, and that we therefore were compelled to express and measure resistances in our calculations, in volts, that is, in terms of the number of volts which would be required to cause 1 amp to flow through that resistance. The unit of resistance (equal to the one now called the ohm) would then naturally be 1 volt, hence numerically equal to the unit of e.m.f. This would, of course, be quite correct, and calculations would give the same results as they do now with the ohm, but it would be confusing and more cumbersome; it is mentally disturbing to measure a resistance in terms of volts, because we know that volts represent a different physical quantity than resistances. It unquestionably gives us a greater mental comfort to give this same unit of resistance a name of its own, and one which is different from the unit of voltage, because it is a different physical quantity.

Ohms are equal to the ratio of the volts to the amperes; to measure a resistance in volts is, therefore, making the denominator of this ratio unity so that the numerical value of this ratio is then equal (numerically, but not physically) to the numerator. It is, therefore, a correct expedient to use when we have nothing better. But in electrical calculations it has been found very convenient to create something better, namely, the ohm; it changes nothing, as the unnamed unit is equal to the ohm; it is, therefore, merely adopting a name.

The above analogy applies very perfectly to thermal resistances. It was shown in the article referred to that thermal resistances can be correctly expressed and measured by the drop of temperature for a unit flow of heat (1 watt) just as electrical resistances could be measured by the drop in volts for a unit flow of current (1 amp). But why should we involve in thermal calculations the mentally disturbing features of measuring a thermal resistance in degrees of temperature when we know that physically they are two different quantities? The same is true of watts and conductances. Why not do as was done in electrical mensuration, give these same units names; it will not change anything else; quantitatively they will remain the same.

The purpose of this article is to urge that this be done, namely, that a name be given to the unit of thermal resistance which was shown in the previous article to be the most rational unit, and one which gives the simplest possible relations, namely, unit relations, just as in electrical calculations. Calculations of thermal flows would then be precisely as simple as the electrical ones made with Ohm's law; there would be no reduction coefficient to remember or to use, as it would be unity, like in Ohm's law. It would simplify the conception of a unit of thermal resistance.

To select a new name, like that of Peclet, so as to honor those who taught us the fundamental principles of this branch of science, would be following a very good custom. But the number of such new names is becoming great and requires them to be defined. It, therefore, seems a better expedient to adopt a half-way measure, and name this unit the "thermal ohm." This term was originally sugggested by Dr. A. E. Kennelly.

The thermal ohm, as applied to the quantity above mentioned, is then that thermal resistance which will require a drop of temperature of 1° C. for 1 watt of heat flow.

Ohm's simple law then applies numerically to calculations of thermal flows. If R is the thermal resistance in thermal ohms, W is the heat flow in watts and T the drop in temperature in centigrade degrees, then the thermal Ohm's law is

$$W = \frac{T}{R}$$

which may then be unhesitatingly used in calculations, as the units are then such as will avoid all coefficients and reduction factors.

Specific thermal resistances (that is, resistivities) are then expressed in terms of thermal ohms for 1 in. (or centimeter) cube.

The numbers expressing such resistances in thermal ohms will be the same as the numbers expressing the drop in temperature for a unit of heat current, as explained in the previous article; the only change is in the name; instead of saying that the resistance is so-and-so many degrees, it would be said to be the same number of "thermal ohms."

The specific resistance of copper is about 0.1 thermal ohm for an inch cube; that of graphite about 0.3; carbon about 1.0; iron about 0.5; hot firebrick about 44. The total thermal resistance of the walls of a well constructed furnace of about 100 kw might be of the order of about 0.15 thermal ohm, hence according to the thermal Ohm's law the loss through the walls for a temperature of about 1500° C. would be $W = 1500 \div 0.15 = 10{,}000$ watts. Or conversely, if a 10 per cent loss (= 10,000 watts) is to be allowed for such a furnace, then as $R = T/W$, the thermal resistance of the walls must be designed to be equal to $1500 \div 10{,}000 = 0.15$ thermal ohm.

According to this same system thermal conductances should be expressed in "thermal mhos" instead of in watts; they will again be numerically the same as the number of watts. Moreover a thermal conductance expressed in thermal mhos would be equal to the reciprocal of the thermal resistance in thermal ohms, just as the electrical mho is the reciprocal of the electrical ohm.

The definition of a thermal mho then is that conductance which will allow 1 watt of heat flow to pass when the difference of temperature is 1° C.

Conductivities will then be expressed in terms of thermal mhos for 1 in. (or centimeter) cube. They will be numerically equal to those heretofore expressed in watts, and are about as follows for a 1-in. cube: Copper, 10 thermal mhos; graphite, 3.3; carbon, 1; iron, 2; hot firebrick, 0.023. The conductance of the above-mentioned furnace will be about 7 thermal mhos.

These units are, therefore, of a very convenient size to use in calculations. The thermal ohm and mho are, of course, independent of whether inches or centimeters are used, just as with the electrical ohm and mho. In stating resistivities and conductivities, however, it is necessary to state whether the numbers of ohms or mhos refer to an inch cube or to a centimeter cube.

Thermal conductivities and conductances are often stated in books and tables in terms of a unit based on the gram calorie per second, the cubic centimeter, and the centigrade degree; this unit is generally called the c.g.s. unit, and is generally the one understood to be used by physicists when they fail to state which one they are referring to. The reciprocal of this is then necessarily the c.g.s. unit of thermal resistances.

To call these the c.g.s. units is, however, an error; they are not really the true and correct c.g.s. units, or absolute units, notwithstanding their common usage as such by physicists. A transmission or flow of heat, whether by conduction, radiation or convection, is energy (work or heat) per second and is, therefore, a physical quantity of exactly the same nature as power, like watts, horse-power, etc., just as the foot and the meter are both measures of the same physical quantity, namely,

a length But the absolute or c.g.s. unit of power is the erg per second, that is, the dyne-centimeter per second, and not the calorie per second, as assumed by physicists who use the term as above described. This unfortunate error, which is so commonly committed, may, therefore, cause confusion unless an author describes specifically what he means. This confusion is another good reason for abandoning both of them in practical work and using the practical units recommended in this article.

These true c.g.s. or absolute units are, moreover, a very inconvenient size to use in practice The practical units here suggested are the most convenient to use in all calculations involving electrically produced heat, as in electrical furnaces. The common though incorrect c.g.s. unit—that is, the gram calorie unit—is the most convenient to use in practice when the heat is expressed in calories; it is also the most convenient one for the mathematical physicist under the same circumstances. When, however, the physicist's problem involves electric heat, or, more broadly, a flow, radiation or other transmission of energy of any kind, then the most convenient unit for the mathematical physicist is the true and correct c.g.s. unit described above and based on the erg.

The following tables are a complete set of all the conversion factors and relations between these several units, that is, the new units, the gram calorie units and the true c.g.s. units; the reciprocal values of all these factors are given, so as to avoid all divisions by them; the approximate values are within 2 per cent. They have been carefully checked and cross checked. The unit of thermal conductance referred to in the writer's former papers as "watts" is identical with the one herein called the "thermal mho." And the resistance unit referred to in a previous paper as the "degree" is herein called the "thermal ohm."

Table of Conversion Factors.

(Note.—A flow or radiation of heat may be measured in watts, in gram calories per second, or in ergs per second. In this table the term "c.g.s. unit," that is, the absolute unit, refers to the true ones based on the erg per second; those often called "c.g.s. units" and based on the gram calorie are here given under the latter term. The centigrade degree is understood to be meant in all these units.)

Thermal Resistance.

Thermal ohms
= centigrade degrees ÷ watts = degrees per watt.
= 1 ÷ thermal mhos.

1 thermal ohm:
= a resistance requiring 1° C. per watt of flow.
= 4.18617 gram calorie units; approx. 50/12.
= 0.0000001 c.g.s. unit of thermal resistance

1 gram calorie unit:
= a resistance requiring 1° C. per flow of a gram calorie per second
= 0.238882 thermal ohms: approx 24 100.
= 2.38882×10^{-8} c.g.s. units of thermal resistance.

1 c.g.s. unit of thermal resistance:
= a resistance requiring 1° C. per flow of an erg per second.
= 10,000,000 thermal ohms.
= 41,861,700 gram calorie units

Thermal Conductance

hermal mhos.
= watts ÷ centigrade degrees = watts per degree
= 1 ÷ thermal ohms.

thermal mho
= a conductance permitting a flow of 1 watt per degree
0.238882 gram calorie unit; approx 24/100.
= 10,000,000. c.g.s. units of thermal conductance.

1 gram calorie unit
= a conductance permitting a flow of 1 gram calorie per second per degree.
= 4.18617 thermal mhos; approx. 50/12.
= 41,861,700. c.g.s. units of thermal conductance.

1 c.g.s. unit of thermal conductance
= a conductance permitting a flow of 1 erg per second per degree.
= 0.0000001 thermal mho.
= 2.38882×10^{-8} gram calorie units.

Thermal Resistivities.

Note.—These are all the reciprocals of the corresponding conductivity units.

1 gram calorie, cubic centimeter, unit:
= 0.3937 gram calorie, cubic inch, unit; approx. 4/10.
= 0.238882 thermal ohm, cubic centimeter, unit; approx 24/100.
= 0.0940478 thermal ohm, cubic inch, unit; approx. 2/21
= 2.38882×10^{-8} c.g.s. units of resistivity.

1 gram calorie, cubic inch, unit:
= 2.54001 gram calorie, cubic centimeter, units; approx. 10/4.
= 0.606762 thermal ohm, cubic centimeter, unit; approx. 3/5
= 0.238882 thermal ohm, cubic inch, unit; approx. 24/100.
= 6.06762×10^{-8} c.g.s. units of resistivity.

1 thermal ohm, cubic centimeter, unit:
= 4.18617 gram calorie, cubic centimeter units; approx. 50/12
= 1.64809 gram calorie, cubic inch, units; approx. 5/3.
= 0.393700 thermal ohm, cubic inch, unit; approx. 4/10.
= 10^{-7} c.g.s. unit of resistivity.

1 thermal ohm, cubic inch, unit:
= 10.6329 gram calorie, cubic centimeter, units; approx. 21/2
= 4.18617 gram calorie, cubis inch, units; approx. 50/12.
= 2.54001 thermal ohm, cubic centimeter, units; approx. 10/4
= 2.54001×10^{-7} units of resistivity.

1 c.g.s. unit of resistivity:
= 4.18617×10^{7} gram calorie, cubic centimeter, units
= 1.64809×10^{7} gram calorie, cubic inch, units.
= 10^{7} thermal ohm, cubic centimeter, units.
= 3.93700×10^{6} thermal ohm, cubic inch, units

Thermal Conductivities.

Note.—These are all the reciprocals of the corresponding resistivity units.

1 c.g.s. unit of conductivity:
= 2.54001×10^{-7} thermal mho, cubic inch, unit.
= 10^{-7} thermal mho, cubic centimeter, unit.
= 6.06762×10^{-8} gram calorie, cubic inch, unit.
= 2.38882×10^{-8} gram calorie, cubic centimeter, unit

1 thermal mho, cubic inch, unit:
= 3.93700×10^{6} c.g.s. units of conductivity.
= 0.393700 thermal mho, cubic centimeter, unit; approx. 4/10.
= 0.238882 gram calorie, cubic inch, unit; approx. 24/100.
= 0.0940478 gram calorie, cubic centimeter, unit; approx. 2/21

1 thermal mho, cubic centimeter unit:
= 10^{7} c.g.s. units of conductivity.
= 2.54001 thermal mho, cubic inch, units; approx. 10/4.
= 0.606762 gram calorie, cubic inch, unit; approx. 3/5.
= 0.238882 gram calorie, cubic centimeter, unit; approx 24/100.

1 gram calorie, cubic inch, unit:
= 1.64809×10^{7} c.g.s. units of conductivity.
= 4.18617 thermal mho, cubic inch, units; approx. 50/12.
= 1.64809 thermal mho, cubic centimeter, units; approx. 5/3
= 0.393700 gram calorie, cubic centimeter, unit; approx. 4/10.

1 gram calorie, cubic centimeter, unit:
= 4.18617×10^{7} c.g.s. units of conductivity.
= 10.6329 thermal mho, cubic inch, units; approx. 21/2.
= 4.18617 thermal mho, cubic centimeter, units; approx. 50/12
= 2.54001 gram calories, cubic inch, units; approx. 10/4.

The following table of values will show the convenient size of these two units. It has been calculated from the determinations of S. Wologdine given by A. L. Queneau in this journal, September, 1909, page 383. They are recent determinations of the materials of commerce and do not refer to extra pure or inferior grades. The temperature was about 1000° C. Further details and the method by which they were determined are described in that article. The present writer's opinion is that

some of the heat entering the samples in those tests is likely to have been lost before it was measured. If so, the conductivities given would be too high or the resistivities too low. The figure given above for firebrick was taken from recent careful determinations of Clement and Egy; it gives the conductivity about half as great or the resistivity about twice as great as in the following table; the temperatures were about 400° C. to 800° C. The figures given above for electrode materials were taken from the recent determinations of the writer.

	Resistivities in Thermal Ohms		Conductivities in Thermal Mhos	
	cb. in.	cb. cm.	cb. in.	cb. cm.
Graphite brick	3.76	9.56	0.266	0.105
Carborundum brick	4.06	10.3	0.246	0.097
Magnesia brick	13.2	33.7	0.0755	0.0298
Chromite brick	16.5	42.0	0.0606	0.0239
Firebrick	22.4	56.9	0.0446	0.0176
Checker brick	24.1	61.2	0.0415	0.0163
Gas retort brick	24.7	62.8	0.0404	0.0159
Building brick	26.9	68.2	0.0372	0.0147
Bauxite brick	28.5	72.4	0.0351	0.0138
Glass pot	34.8	88.6	0.0287	0.0113
Terra cotta	40.9	104.0	0.0244	0.0096
Silica brick	46.9	119.5	0.0213	0.0084
Kieselguhr brick	52.3	133.0	0.0191	0.0075

As an illustration of the simplicity of the calculations when these units are used, let the inside of a small furnace be 6 in. cube; let the wall consist of a 4-in. layer of silica brick on the inside and 4 in. of firebrick on the outside. The true average cross-section* of the layer of silica brick perpendicular to the heat flow (namely, the geometric mean, that is, the square root of the product of the extreme sections) for the whole furnace will be 504 sq. in. If its resistivity, from the above table, is 47 thermal ohms, its resistance will be

$$R = \frac{r \times l}{S} = \frac{47 \times 4}{504} = 0.37 \text{ thermal ohm.}$$

A similar calculation for the outside layer of firebrick having a resistivity of 22 thermal ohms gives its resistance as 0.048 thermal ohm. Hence, the total is the sum of these two, which is 0.42 thermal ohm. Incidentally, it shows what a very much smaller insulating effect this much larger outside layer has.

Suppose the drop of temperature through the wall to be 1500° C. Then, according to the thermal Ohm's law, the loss of heat will be

$$W = \frac{T}{R} = \frac{1500}{0.42} = 3580 \text{ watts, or about } 3\tfrac{1}{2} \text{ kw.}$$

This seems high, but the writer fears the resistivities of Wologdine are too low. The heat resistance at the joint, which has not been taken into account, will reduce this loss somewhat. If the arithmetical average cross-section had been used, as is more usual, the calculated loss would have been nearly 50 per cent greater, hence greatly in error.

If it is desired to know the temperature at the joint, then according to Ohm's law the drop of temperature in the inside layer will be

$$T = WR = 3580 \text{ watts} \times 0.37 \text{ thermal ohm} = 1325°.$$

Hence the temperature will be 1325° less than in the inside. As a check the drop in the outside layer will be 3580 watts × 0.048 thermal ohm = 175°, making the total 1325 + 175 = 1500°. This again shows how much smaller the insulating effect of the outside layer is.

To have used conductivities in the above calculations instead of resistivities would have made them quite tedious and much longer.

Philadelphia, Pa.

*"Heat Conductances through Walls of Furnaces." Hering. Trans. Amer. Electrochem. Soc., Vol. 14, 1908, p. 215. This Journal, Vol. VI, p. 495.

The Tooele Smelter.*

BY C. H. REPATH AND A. G. MCGREGOR.

The Tooele smelter is situated about four miles east of Tooele City, Tooele County, Utah, and by rail 41 miles from Salt Lake City. The smelter is connected by its own railroad (the Tooele Valley Railroad) with the San Pedro, Los Angeles & Salt Lake Railroad, the junction being about seven miles from the smelter site. The smelter is located upon a hillside, which makes it possible to a large extent for the level of delivery of the product of one building to be the level of charge floor of the next succeeding one.

The Ores.

The ore treated comes principally from the mines of the Utah Consolidated Mining Company in Bingham Canyon. These mines in Bingham Canyon are connected with the smelter by an aerial tramway, four miles long, having a capacity of 125 tons per hour. Custom ores from other mines are received over the San Pedro Railroad and the Tooele Valley Railroad.

FIG. 1.—TOOELE SMELTER. BLAST-FURNACE BINS AT LEFT, SAMPLING AND CRUSHING PLANT IN CENTER, ROASTED ORE BINS AT RIGHT.

Receiving Bins.

The receiving bins are constructed of steel, and are fireproof throughout. The receiving bins have a capacity of 10,000 tons of ore and coal. The bins are so located and arranged that they may be served either by the 50-ton electric larry cars, which deliver the ore from the tramway terminal, or by standard railroad cars.

Sampling Mill and Crushing Plant.

This building is of steel and concrete throughout. It is 40 ft. x 84 ft. in plan and is five stories high. The mill is composed of two sections and each section has a crushing and sampling department. The ore from the receiving bins is automatically fed upon belt conveyors which convey it to shaking grizzlies that discharge the coarse ore into the crushers. The ore from the crushers is elevated to the top of the building by bucket elevators. In the sampling department the ore is cut four times by Brunton automatic samplers, cutting out one-fifth of the amount each time and discarding four-fifths so that of each ton of ore crushed a sample weighing 3.2 lb is obtained.

The sampling department of each section of the mill contains besides a 12-in. x 24-in. Blake crusher, which is common to the crushing and sampling department:

One O-A sampler.
One 20-in. x 10-in. crusher.
One 1-A sampler.

*A paper read before the Utah Society of Engineers.

One 48-in. x 12-in. rolls
One 2-A sampler
One 26-in. x 15-in. rolls
One 4-A sampler

The discard from the sampling machines drops into screens of the crushing department, from which it is conveyed to the roaster bins. The final sample is quartered by a quartering shovel, the resulting sample is then dried on a steam dryer, ground in an Englehardt sample grinder, and again ground in a Braun pulverizer, so as to pass through a 100-mesh sieve

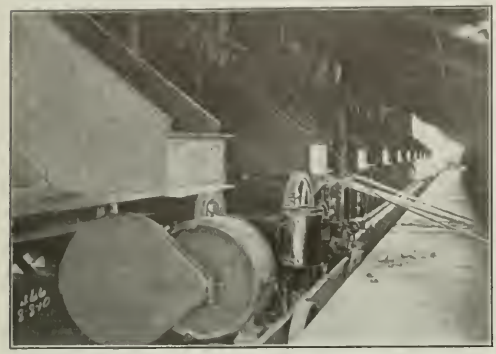

FIG. 2.—CONVEYOR FEEDING MACHINE.

The 100-mesh product is put up in three sample packages, one of which goes to the smelter laboratory for analysis, one to the owner of the ore, and one is filed away for use in case of a dispute.

The crushing department of each section of the mill contains:

One 12-in. x 24-in. Blake crusher.
Two 15-in. x 9-in. crushers
Two 48-in. x 14-ft. screens.
Two 48-in. x 12-in. rolls.

From the crushing plant the crushed and sampled ore is conveyed by belt conveyors to the McDougal roaster steel receiving bins, which have a capacity of 5700 tons. If it is desired the coarse ore after being sampled may be conveyed to the blast furnace steel receiving bins, which have a capacity of 3500 tons.

A blast furnace has not yet been installed, but provision has been made for it.

Roaster Plant.

From the roaster ore bins the ore is automatically fed upon a conveyor system which conveys it up to and discharges it directly into the McDougal furnace charge hoppers. The ore in transit from the bins to the roaster plant passes over a Blake-Denison automatic and continuous weighing and recording machine, which accurately weighs all the ore delivered to the roaster plant.

The roaster plant consists of two buildings, each 64 ft. x 162 ft. Each building contains 16 McDougal calcining furnaces of the Evans-Klepetko type. These furnaces have six hearths 16 ft. in diameter and are 18 ft. high. They have revolving water-cooled shafts and arms, driven by a suitable gearing from the bottom. The rabbles are so set as to move the material from the circumference to the center and vice versa on alternate hearths until it finally drops into the calcine hoppers immediately over the tracks of the electric tramming system for transportation to the charge floor of the reverberatory building. No fuel is used other than the sulphur in the crushed ore, the burning of which furnishes sufficient heat to do the calcining. The gases are taken through flues into the large brick and steel dust chamber, which is 120 ft. x 140 ft. x 40 ft. high. This dust chamber is so arranged that the bottom forms a series of hoppers. The entire contents of the dust chamber can be drawn into flue-dust cars of the electric tramming system and conveyed directly to the reverberatory furnaces for smelting. Each of the 32 McDougal furnaces has an approximate capacity of 45 tons in 24 hours.

Reverberatory Plant.

The reverberatory plant consists of three buildings joined together; the reverberatory charge building, containing the ore and coal hoppers over the furnaces, 66 ft. x 280 ft.; the reverberatory furnace building, covering the larger portion of the furnaces, 82 ft. x 280 ft., and the boiler house, 36 ft. x 326 ft. The latter at the present time contains four 746-hp waste-heat boilers and three 350-hp hand-fired boilers. The boilers are all of the Stirling water-tube type. The reverberatory plant contains five coal-fired reverberatory furnaces, the hearth dimensions of which are 19 ft. in width by 102 ft. in length, with a grate area of 7 ft. x 16 ft. These furnaces have a maximum capacity of 300 tons of calcine in 24 hours on natural draft. The fuel used is Diamondville coal, shipped from the mines in Wyoming owned by the Washoe Copper Company. The coal is dumped from large railway coal cars or from cars of the electric tramming system into hoppers that have five points of discharge, directly over the firebox. The flame after leaving the furnace passes through a 746-hp Stirling boiler, which reduces the temperature of the gases going to the main flue to about 600° Fahr. By this means approximately 600 boiler horse-power are obtained from each furnace from the waste heat. The ashes from the furnace firebox fall into hopper cars of the electric tramming system and are hauled away to the ash dump. The slag is skimmed from the reverberatories twice in eight hours. It is allowed to accumulate until its depth is from 3 in. to 4 in. above the skimming plate in the front of the furnaces, and then skimmed into slag cars having a capacity of 225 cu. ft., which are hauled to the slag dump over the electric tramming system. The matte is tapped from the side of the furnaces through a copper tap-hole plate and run through cast-iron launders directly into the converter building.

One of the waste-heat boilers referred to above has been

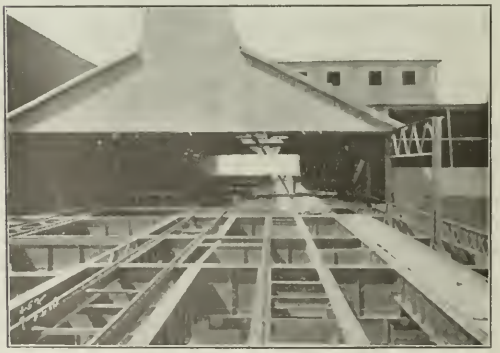

FIG. 3.—REVERBERATORY CHARGE TRACKS.

equipped with a superheater. The superheater is just at the rear of the boiler. In case it proves satisfactory the other waste-heat boilers can each be equipepd with one.

Converter Plant.

The main converter aisle of this plant is 65 ft. x 408 ft., and the casting shed is 52 ft. x 255 ft. The main converter aisle contains five converter stands for 96-in. x 150-in. converters and the converter-lining department. The converters used are of the horizontal barrel type and are 96 in. in diameter by 150 in. in length. The convereters are electrically operated. The main aisle is served by one 60-ton electric traveling crane.

The casting department is served by a 30-ton electric traveling crane. As stated previously, the matte is received in launders directly from the reverberatory furnaces. The end sections of these launders are pivoted so that the matte may be poured

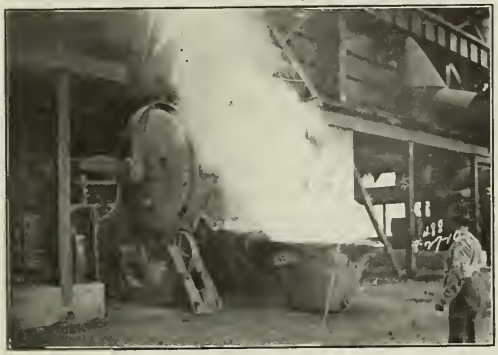

FIG. 4.—SKIMMING FIRST CONVERTER.

directly into the converter opposite a reverberatory furnace, or the launder may be turned so that the matte will fall into a ladle and be transferred to any of the other converters in the building. The slag from the converters is poured into unlined cast-steel ladles and transferred to the reverberatory furnaces by means of overhead cranes. There are two of these cranes, of 12½-ton capacity each. The blister copper is poured into a ladle and transferred by the crane and transfer car to the casting department. In the casting department the ladle is handled by a crane and its contents emptied into anode molds, which completes the operation of producing pig copper. The pig copper contains 99 per cent copper and the gold and silver. It is shipped East, where it is further refined and the gold and silver extracted.

Converter Lining.

The converters are lined in the main converter building, but the lining material is prepared in a building, 24 ft. x 40 ft., adjacent to the ore-crushing plant and receiving bins. The lining material contains a high percentage of silica, having gold and silver values. The material is crushed by the crushers in the crushing plant, and conveyed to a 7-ft. dry pan and pug mill in the clay and silica-mixing and grinding plant. The product of the pug mill drops directly into a bin over the electric tramming system, from which it is transported in cars to the converter-lining plant. The lining plant at the converter contains a 7-ft. grinding and mixing pan. From the grinding pan the material is transferred to the converter, where it is tamped around a cast-iron form by a special Ingersoll-Sergeant tamping machine, 5 in. diameter by 20 in. stroke. The tamping machine is supported by a revolving jib crane, the vertical motion of which is controlled by an electric hoist. After lining the converter is removed to its stand, where it is dried.

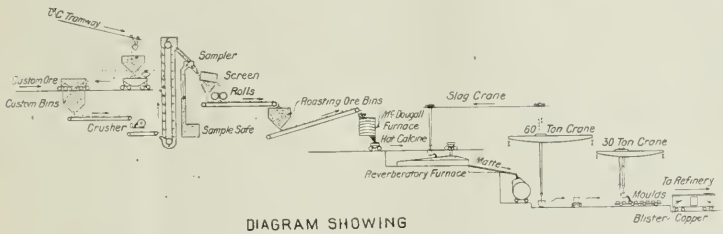

DIAGRAM SHOWING
FLOW OF MATERIAL THROUGH TOOELE PLANT

Smelter Power Plant.

The smelter power-plant building is of brick and steel, the main aisle of which is 52 ft. x 240 ft. It has a lean-to 30 ft. x 90 ft. for the steam auxiliaries, and one on the other side, 30 ft. x 60 ft., for the switchboard and transformer. The building contains the various power engines, blowing engines, compressors and auxiliaries and is equipped with an overhead electric traveling crane.

The building contains:

One 16-in. x 32-in. x 36-in. Corliss engine direct-connected to a 250-kw, 500-volt, direct-current generator.

One 15-in. x 30-in. x 36-in. Corliss engine direct-connected to a 250-kw, 500-volt, direct-current generator.

Two 18-in. x 26-in. x 40-in. x 24-in. vertical, triple-expansion engines, each direct-connected to one 750-kva, 2200-volt generator.

One 15-in. 30-in. x 36-in. x 42-in. blowing engine.

One 26-in. x 52-in. x 52-in. x 48-in. blowing engine.

One 13½-in. x 26-in. x 26-in. x 15-in. x 36-in. steam-driven air compressor.

One 26¼-in. x 15¼-in. x 18-in. motor-driven air compressor.

One 200-kw motor-driven direct-current generator.

One 50-kw motor-driven exciter generator.

One engine-driven 50-kw exciter generator.

Two feed pumps having a total capacity of 678 gal. per minute.

One fire pump of 750 gal. per minute.

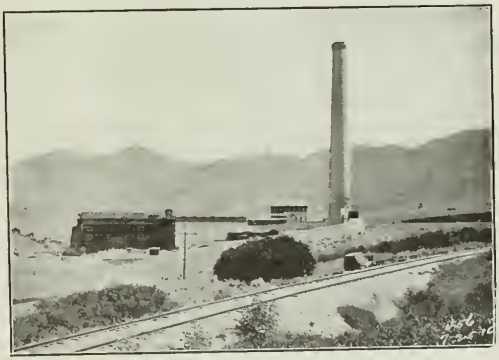

FIG. 5.—FLUES AND STACK.

Besides the above the plant contains the necessary condensers, vacuum pumps, feed-water heaters, traps, etc. The steam for these engines is furnished by the waste-heat boilers and hand-fired boilers adjacent to the reverberatory building.

The engine-driven generators, in addition to furnishing power for the various motors about the plant, will also furnish power for driving air compressors and for hoisting and pumping purposes at the mines of the Utah Consolidated Mining Company.

A cooling tower of the natural-draft type has been constructed for cooling the water from the power-plant condensers and for cooling the water for the McDougal roasting furnaces. The tower is 49 ft. high, and at the base is 20 ft. wide by 140 ft. in length.

Electric Tramming System.

The equipment of the electric tramming system consists of three 7½-ton and two 18-ton locomotives and 50 cars.

Flues and Stack.

The main flue which conveys the gases from the reverberatory furnaces and converters is 20 ft. wide x 18 ft. high x 1360 ft long. One section of the converter flue is 8 ft. wide x 12 ft. deep and 248 ft long; the other section has a cross-sectional area of 132 ft. and is 181 ft. long. Both sections are provided with hopper bottoms and chutes for removing the flue dust. The roasting-plant flue is 16 ft. x 16 ft. and is 255 ft. in length. The main and roasting-plant flues are rectangular in section and of brick and I-beam construction, while the converter flues are constructed entirely of steel.

The stack is 25 ft. inside diameter at the top by 350 ft. in height above the base.

The water for various purposes in the smelting plant is obtained from Pine Canyon. The water is conveyed by gravity from a dam in the canyon through a 12-in. pipe approximately 5000 ft. to a standpipe of 50,000-gal. capacity at the plant. The water is distributed from the standpipe to the various departments.

The Status of Metallurgy in Colorado in 1910.

Editorial Correspondence.

Progress in metallurgy in Colorado during 1910 was not marked by any notable innovations, although it is probably true that foundations were laid on which a prosperous industry can be raised in the next few years. The principal subject attracting attention late in the year was the possibility of reviving the mining industry in the State, from which it may be properly inferred that mining and its allied industry metallurgy have been declining for several years past and reached so low an ebb during 1910 as to attract the serious attention of those most concerned.

The decline and low ebb have been the result of a combination of causes, no one of which can be assigned as the sole cause of the condition. The decrease in quantity and quality of smelting ore, coupled with the failure of mine owners to anticipate this outcome and seek new and cheaper methods of treatment ere it arrived, may be given as the principal causes contributing to the present condition of the industry in Colorado. That there are still great possibilities in mining, however, and that metallurgy will play a large part in the success of the future may be gathered from the following presentation of the situation.

Smelting.

Smelting was the pioneer metallurgical process used in Colorado and constituted a thriving industry until the last decade, during which time it has steadily declined. In 1910 the American Smelting & Refining Company had in operation four Colorado plants: Globe (Denver), Pueblo, Arkansas Valley (Leadville) and Durango. These were operated at about half capacity throughout the year. Lead ore was largely brought in from the Coeur d'Alene district, Idaho, while quite a large tonnage of silicious silver ore was shipped into the State from Cobalt, Canada. The smelting of the latter has given rise to some interesting problems, such as the study and treatment of the cobalt speiss which is formed in the smelting process. It is understood that this work is still in progress, and no information is available. A new addition to the Globe plant was completed last year, viz., an arsenic plant for the treatment of flue dust. The process employed is the usual one of roasting and condensation in flues. Local flue dust is treated, as well as some flue dust that is shipped to the plant from Utah.

A number of independent smelting plants have operated during 1910, some of them leading the usual precarious existence of such institutions, and others apparently prospering as well as the status of the industry will permit. Unfortunately for the industry itself, some of these ventures are ill advised, being the creations of promoters who lack information on the basic principles of the smelting business and proceed along lines which almost necessarily lead to failure. It has been observed also, in some instances, that the greater the hazard the less competent the management and the more sudden the termination of operations. Only by the employment of the most skilful technical talent, combined with the most painstaking preliminary investigation, can such misfortunes be avoided.

The Argo plant of the Boston & Colorado Smelting Company, near Denver, was permanently closed in March, 1910. The company had enjoyed a successful career of 40 years' duration, during a part of which time it operated a secret process of refining which made it independent of the larger corporations which sprang up with the introduction of lead smelting. Argo was operated on a copper basis, and owing to the lack of suitable copper ores in Colorado was forced to secure supplies from other States, often at great cost and little profit. The plant has now been completely dismantled by wreckers, who took charge after a thorough clean-up.

The plant of the Modern Smelting & Refining Company, near Denver, which was blown in during 1909, was closed in May, 1910, after less than a year of operation. During this time the plant was occasionally closed for various reasons and finally abandoned as an unprofitable venture. There have been occasional rumors of a revival of activity, but nothing definite has transpired.

In April last, the smelting plant at Golden, which had been purchased and remodeled by the North American Smelter & Mines Company, was placed in commission. With the exception of a few interruptions this plant has been in steady operation, and the prospects are good for continued success. The company has pursued a sound policy, securing excellent technical and business management, and has endeavored to assure a suitable supply of ore by purchasing good mines. The latest acquisition of the company is the Santiago group of mines in the Clear Creek district. As there are apparently ample funds behind the management it is probable that the plant will be operated for an indefinite period. The matte produced is sold to the Globe plant of the A. S. & R. Company.

Still another matte smelter was put in operation at Alma about the middle of the year. Some of the details of this plant were given in our issue for November, 1910, page 610. It is probable that operations will be irregular this winter owing to the difficulty of getting suitable ore supply.

The Salida smelter of the Ohio & Colorado Smelting & Refining Company has been in operation as usual, although at reduced capacity. This plant is the only custom lead smelter in the State which is not controlled by the A. S. & R. Company. It has the usual standard equipment with the addition of the Dwight-Lloyd sintering machines for roasting and agglomerating fine ore and concentrate. The other smelters use the Huntington-Heberlein system of roasting.

An innovation in smelting furnaces was introduced in Colorado last year, known as the Partridge smelter. The change from the standard type of furnace consists in the use of a low-pressure blast of large volume, preheated by the slag flowing from the furnace. An annular opening between crucible and shaft is substituted for tuyères, and the blast is heated in a chamber into which the slag is discharged. A complete description of the furnace and its operation will be found in our issue for December, 1910, page 671. One of these plants has been erected at Como to treat the ore of a mining company hitherto shipping to custom smelters. Other plants are projected for similiar properties, notably in the San Juan region, where, by reason of isolation, it is difficult to ship ores at a profit which might be concentrated by fire in a simple and efficient furnace like the Partridge.

Zinc smelting in Colorado is in about the same status as in 1909. The plant of the United States Zinc Company at Pueblo has been in operation, with six furnaces and 1440 retorts. The manufacture of zinc dust at this plant was one of the important features of the year. The production of zinc in Colorado has increased greatly during the past year, so that the outlook for this industry is excellent.

On the whole the outlook for the general smelting industry is not particularly attractive, for reasons already given, but it is equally certain that other branches of metallurgy will take the place of smelting and tend to expand the business of ore treatment. The decrease in production of ore suitable for smelting in Colorado is shown by the following figures compiled by Franklin Guiterman, manager for the A. S. & R. Company in Colorado:

	1900.	1909.	Decline per cent.
Gold oz.	1,400,000	1,061,000	24.21
Silver oz.	20,300,000	8,900,000	56.16
Lead lb.	82,137,000	32,360,000	60.60
Copper	No change.		
Zinc	Increasing.		
Value	$50,314,000	$30,196,000	38.55

Chlorination.

The mills of the Portland Gold Mining Company and United States Reduction & Refining Company, both at Colorado City, have continued to use the chlorination process on roasted Cripple Creek ores, following this with cyanidation of the tailings. At the former the chlorine is generated from bleaching powder and sulphuric acid, and at the latter by an electrolytic plant of 100 McDonald cells. The barrel charge at the Portland consists of 20,000 lb. of ore, 300 of acid and 100 of bleach (30 per cent available chlorine). At the other plant the ore charge is the same, with the addition of chlorine solution of suitable strength.

It was expected and earnestly hoped by Colorado metallurgists that the dry chlorination plant of the Western Metals Company, at Georgetown, would be tested during the year. Owing to delays incident to the promotion of a new process the plant has not yet been completed and the initial test is delayed. The process to be used has been described in the technical press,* and is usually referred to as the Malm process. Mr. John L. Malm has made what are expected to be vital improvements on the work of earlier investigators, Swinburne, Ashcroft, Baker and Burwell, whereby the process can be profitably applied to base sulphide ores. The plant at Georgetown will afford an excellent opportunity to test the efficiency of the process, as the Clear Creek district has an abundance of such material.

Cyaniding.

There was an increase in cyaniding in this State during 1910, principally on Cripple Creek ores. It has been predicted from time to time that cyaniding ultimately would supplant chlorination, but as noted above it has not yet fully done so. It is true that chlorination has steadily lost favor with Colorado metallurgists, and it is probable that but for the disinclination to abandon a working plant which represents a considerable investment the present chlorination plants would not be in operation and would be superseded by cyanide works.

Probably the most important event in cyaniding last year was the completion and operation of the Cripple Creek mill of the Portland company. The full details of the process have been kept secret, but it is a direct cyanidation of raw ore, with the addition of a substance or mixture, probably a halogen cyanide and an oxidizing substance. The process was worked out by the metallurgical staff of the company. After coarse crushing the ore is ground in cyanide solution in Chilean mills and concentrated on Wilfley tables. The concentrate is shipped to the old mill at Colorado City. The tailing is classified, the sand being rejected and the slime concentrated on Card tables. The Card concentrate is shipped to the Colorado City mill, and the tailing thickened, agitated and filtered. Zinc dust precipitation according to Merrill's process is used. The plant has a number of new devices, such as the Akins classifiers and Rothwell & Akins agitating tanks, and Portland continuous revolving slime filters. The mill is in steady operation on low grade mine ore, and its successful operation will point the way to the erection of other local mills.

Stratton's Independence mill, which was started in 1909, was in operation last year at about 50 per cent increased capacity This mill is treating dump ore of an average value of $3.60 at a cost of about $1.50 per ton. The ore is crushed and ground in cyanide solution, then concentrated to remove the sulphotellurides, after which the tailing is cyanided.

The Colorado City mills of the Portland, Golden Cycle and United States Companies were in operation throughout the year, although not always at full capacity. The Golden Cycle uses cyanidation exclusively, while the other two use cyanide only on the tailings from the chlorination plants. The Golden Cycle has probably the best roasting plant in the State, comprising a battery of Edwards' roasters, each having a capacity of 125 tons of Cripple Creek ore daily. The roasted ore is ground to 25 mesh in cyanide solution, passed over blankets and then classified in Dorr classifiers. The sand is leached and the slime agitated and filtered in a vacuum system which was newly installed last year Precipitation is on zinc shavings

The only other district in the State where cyaniding has been practised regularly is the San Juan, where the Camp Bird at Ouray and the Liberty Bell at Telluride have been using the process as in the past. There has been a tendency on the part of engineers to extend the use of the cyanide process to other ores and concentrates and during the past year some successful experiments were carried on at Breckenridge and in the Clear Creek district. In connection with this work some new machines were investigated, notably the Bonnot pulverizer and the A. Z. agitator, both of which were found well suited to the work required of them.

The Clancy process of cyaniding was disclosed to the public too late in the year to permit of any tests being made on Colorado ores, but it is certain that the process will be applied to some Cripple Creek ores this year, in an endeavor to test its efficiency. A complete statement of the process will be found in our issue for November, 1910, page 623.

Ore Dressing.

There was probably more activity and new business in this branch of metallurgy than in any other in the State during 1910. The greatest progress was made in the concentration of zinc-lead-iron sulphides, and several new plants were erected and operated. The systems used vary with the nature of the ores, but may be broadly classified into two, viz., those in which roasting is or is not used prior to magnetic separation. Again, the systems employing roasting may use that process either before or after the accompanying wet dressing on reciprocating tables. When roasting is employed low intensity magnetic separators are used, while the high intensity separators are used on raw zinc-iron middlings from wet concentration. The low intensity machines attract the strongly magnetic iron in the roasted mixture, while the high intensity machines attract the weakly magnetic zinc from the raw mixture.

The mills of the American Zinc Extration Company, Leadville; Wilson Mining Company, Robinson; Kokomo Metals Company, Kokomo; Eagle Mining & Milling Company, Red Cliff; United States Zinc Company, Pueblo, and Western Chemical Manufacturing Company, Denver, all use a system of roasting, magnetic separation and wet dressing. With the exception of the Kokomo mill they all use the following general scheme: Crushing, grinding, screening, drying, roasting, magnetic separation of the iron and table concentration of the lead and zinc. At the Kokomo mill the wet dressing comes first, producing lead concentrate, tailing and zinc-iron middling, the last being dried, roasted and magnetically separated.

A difference of opinion exists among metallurgists regarding the relative places of the roasting and wet dressing in the general scheme indicated. Those favoring wet dressing

*West. Chem. & Met., Vol. 5, No. 3; Vol. 6, No. 5.

first and roasting with magnetic separation last argue that they immediately get rid of the tailing and save a lead concentrate and zinc iron middling, the last alone requiring roasting and magnetic treatment. They contend that they thereby avoid unduly loading the drying, roasting and magnetic units with an excess of material that does not require that treatment and consequently reduce the plant required. On the other hand it is contended that slime losses in preliminary wet dressing, and the difficulty in drying the middling, are points against that procedure. Further, that table dressing following roasting is somewat improved owing to the effect which roasting sometimes has of widening the range of specific gravity of the various minerals. It is also claimed that if iron carbonate is present its separation from the zinc is better effected by wet dressing after roasting than before. The two processes have not been in operation long enough to compare their efficiencies or to prove the claims of their adherents.

Several types of roasters are in use at these mills, but the preference is in favor of the multiple superimposed hearth type. A Wilfley vertical furnace was installed at the Kokomo mill, and revolving furnaces at the Leadville plant. At the Western Chemical Manufacturing Company, where sulphuric acid is made from the roaster gases, the furnaces are of the Herreshoff type. The new Wilfley roaster, which was described in our issue for November, 1910, page 646, has not been installed at any plant.

Three types of magnetic separators are used in these mills, viz., the Dings, Cleveland-Knowles and Fairchild. The first two are well known, but the last is a recent device of O. H. Fairchild, superintendent at the Wilson mill, where the machines are in use. There are three sets of magnets of increasing strength arranged transversely to the shaking feed plate. The magnetically separated iron is removed by traveling belts which move around the magnets. In its travel on the belt the iron comes into fields of alternate polarity which cause the particles to change their position and thereby release any mechanically entrained mineral which should not be removed.

Wilfley and Card tables are almost universally used in Colorado, and each has its ardent supporters. At the Leadville mill mentioned above there are some Ford tables in use. In this connection it is interesting to note that during the past summer the Mine & Smelter Supply Company, owning the Wilfley patents, presented its *prima facie* case against the Hendrie & Bolthoff Company, controlling the Card table, in a suit for damages for infringement. The case was heard before a special examiner, but was not finished. Similar suits were instituted against the Deister and Dunham tables.

In the Creede and Clear Creek districts lead-zinc concentration is by wet processes only, as there is but little iron in the ore. Jigs of the Harz type, and Card and Wilfley tables are used. The zinc concentrate from these districts is of excellent quality, ranging from 40 per cent to 55 per cent. The lead is also of high grade, running from 60 per cent to 72 per cent.

The plant of the Empire Zinc Company, at Canon City, has been in operation throughout the year. Raw zinc-iron middling or crude ore is treated at this plant, which is equipped with Wetherill magnetic separators of the high-intensity type. It is understood that the company contemplates the erection of a new plant embodying roasting, magnetic separation and wet dressing. The company has maintained an elaborate testing plant for some time, and has been active in experimental work. The testing plant contains an Elmore unit, Sutton dry table, electrostatic separator and Wilfley roaster.

Tungsten concentration in Boulder County was the principal metallurgical industry of that district last year. The chief difficulty experienced in this work is the tendency of the friable tungsten minerals to slime, thereby causing high losses. The perfection of the Monell slimer last year is said to have reduced these losses materially. The machine is a canvas belt vanner with end motion, and side feed and discharge of tailing. The concentrate is recovered at the end as usual. The principal feature of the machine is its unusual length and large flat concentrating surface on which the mineral particles have ample time to settle with the least possible disturbance in the succeeding washing.

Miscellaneous.

The Colorado School of Mines testing plant was put under construction last fall. This plant, when finished, will be a complete testing mill for ore dressing and metallurgical processes. The fact that the School authorities plan to permit the use of the plant by professional engineers materially widens its field of usefulness. The School will thus perform a service to the State, and the students will have opportunity to witness and assist in many commercial tests on a variety of ores.

The vanadium industry in Colorado has not grown as fast as was expected. The Primos Chemical Company has a mill in operation at Newmire producing vanadate of iron by first roasting the ore with an alkali salt, then leaching and filtering and precipitating the vanadium with ferrous sulphate. Considerable experimental work was done at Boulder last year by Mr. Warren F. Bleecker in an endeavor to improve the process used at present. The results have been gratifying, and our readers will have an opportunity to secure the results of Mr. Bleecker's work by consulting his series of articles which began in our issue for December, 1910.

Portland cement is made in Colorado by two companies, the Ideal and the United States, both operating in Fremont County, at Portland and Concrete, respectively. The shale and limestone used are quarried from beds exposed in the valley of the Arkansas River. Shale and limestone are mixed in the proportion of 2200 lb. limestone to 5800 lb. of shale, making a mixture of about 73 per cent $CaCO_3$ at the mill, or 64 per cent CaO in the finished product. Ball and tube mills are used to grind the mixture preliminary to clinkering, which is done in 120-ft. kilns, using powdered fuel. Fuel consumption is about 140 lb. to 400 lb. of cement. A plaster of Paris plant is also operated by the Ideal Company at Portland. The United States plant is the newer and said to be thoroughly modern.

The discovery of zinc carbonate at Leadville will undoubtedly give metallurgists an opportunity to display their ability in devising treatment for the low-grade ore which cannot be shipped directly to the smelters. Although metallurgical works have lowered the limit on these ores to 20 per cent zinc, there is a vast tonnage that will not run higher than 15 per cent, and there will undoubtedly be a demand for a method of treatment.

The necessity for co-operative work in Colorado metallurgy is becoming more and more apparent. Many districts have an abundance of low-grade ore, which can be mined and treated only by a concerted effort on the part of many small owners. The time has arrived when the mine owner must work out his own metallurgical salvation if his operations are to prove profitable, and if his own property does not of itself warrant a metallurgical plant, he may well consider the advisability of co-operating with his neighbors who are similarly situated. The wisdom of this course has been demonstrated and it should be more widely considered.

Silver Assay.—In addition to being influenced by the temperature of cupellation, the accuracy of the silver assay is also affected by the quantity of litharge used in the fusion. It has been shown that an excess of litharge tends to lower the silver recovered, the litharge probably carrying some of the silver into the slag along with impurities which are dissolved. A flux containing 12 parts each of soda, potash and litharge, 2 of borax glass and 1.1 of flour will give good results if about 80 grams of the same are taken for ½ A.T. of ore.

The Cyanamide Process in the Metallurgy of Gold.*

BY JOHN COLLINS CLANCY.

After the wide publicity given to the Clancy process by the Moore Filter Company, I feel it incumbent upon me to present the following paper for consideration and discussion.

This process is more immediately concerned with the treatment of refractory ores, these ores generally having their values associated with compounds in chemical combination, which are not susceptible to the action of solvents until roasting or some other method of oxidation has disassociated the said compounds and liberated the precious metals. Roasting is by all means the cheapest oxidation process so far known; any attempt to oxidize the base constituents by chemical means is obviously of a very expensive nature, therefore, impracticable.

It is known that, in some cases, profitable extraction of the precious metals from base ores may be obtained, depending on the rapidity of the action of the cyanide upon the finely divided gold in preference to its action upon the base constituents; that is to say, before the base metals are attacked the gold has gone into solution. Sometimes by keeping down the alkalinity of the solution to almost neutral point in "protective" alkalinity a good extraction may be obtained, as the absence of an excess of alkali will protect the metallic sulphides from decomposition and allow the cyanide to work directly upon the precious metals. Of course, these results are only capable of accomplishment when the precious metals are not in chemical combination, unless a solvent for the chemical compounds is present with the treatment solution. The Clancy process to be described in a measure includes the embodiment of these principles.

Preliminary Experiments.

As a preamble, a little history relating to my earlier experiments on refractory ores may not be out of place. The ores of Cripple Creek, Col., may be taken as an example of so-called refractory ore. I need not dwell upon the past or present metallurgical treatment of these ores, but will only state in passing that straight cyanidation on the raw undressed ore yields but a trifle over 60 per cent extraction. It will be apparent, therefore, that a process to extract from about 90 per cent to 95 per cent of the values without roasting or concentration, and at a reasonable cost, would be a boon to the district of Cripple Creek, as well as other districts in different parts of the world. To this task I set myself some years ago, and my improvements may be seen from the different specifications, which show the line of thought and gradations from earlier processes.

The object of my first experiments on Cripple Creek ores was to find a cheap alkaline solvent for the telluride of gold which occurs in these ores, the tellurides existing principally in the form of (AuAg) Te_2, the minerals calaverite and sylvanite being approximately represented by this formula. The minerals calaverite and sylvanite are often found alone, but more frequently associated with pyrite; this close association with the pyrite increases the difficulty of treatment by chemical solutions as well as separation by mechanical means.

I discovered that the alkaline hypochlorites, hypobromites, hypoiodites (NaClO) (NaBrO) (NaIO) dissolved tellurium readily and also attacked the tellurium in the gold telluride crystals, while their calcium salts—the analogues of bleaching powder—were not solvents. Sodium hypochlorite being the cheapest, therefore, there was a possibility that, by crushing the ore fine, the tellurium would be dissolved off and allow of gold dissolution by cyanidation. The equation representing this reaction may take place after the following:

$$Te_2 + 4\,NaClO + 2\,H_2O = 2\,H_2TeO_3 + 4\,NaCl.$$

The tellurous acid so formed would dissolve in the excess of alkali as

$$H_2TeO_3 + 2\,NaOH = Na_2TeO_3 + 2\,H_2O.$$

A preliminary treatment with NaClO, followed by a cyanide treatment on ore crushed to 100 mesh, invariably gave an extraction of from about 80 per cent to 85 per cent of the gold contents, and frequently a higher extraction when the gold telluride in the ore was not closely crystallized with the pyrite.

The preliminary treatment, however, had its drawback, inasmuch as the remnant NaClO left in the pulp after treatment destroyed cyanide unless washed free therefrom. This washing caused extra filtering and meant two sets of solution treatments, and for that reason, together with the alleged high cost of producing NaClO, it was not continued at this time.

My next process was the employment of a soluble persulphate, such, for example, as ammonium persulphate, as a preliminary treatment to dissolve the tellurium contained in the ore. This solvent $(NH_4)_2S_2O_8$ was superior to NaClO, owing to the fact that the iron sulphides did not become oxidized to any appreciable extent while the tellurium was undergoing dissolution.

Another advantage was that the remnant persulphate did not act to destroy the cyanide on subsequent treatment; on the contrary, it hastened dissolution of the gold, after the following equation:

$$4\,KCN + 2\,Au + (NH_4)_2S_2O_8 = 2\,KAu(CN)_2 + (NH_4)SO_4 + K_2SO_4.$$

The high cost of the persulphate, together with the necessity of using two sets of solutions, was the reason for not continuing this method.

I next tried if it was possible to dissolve the tellurium and gold simultaneously by using a mixture of cyanide and persulphate. This scheme was found to work satisfactorily on gold telluride crystals, but when applied to the ore the method was prohibitive on account of the large amount of persulphate required, due principally to the presence and formation in the solution of ferrocyanide, which caused the action of the persulphate to be completely nullified.

The thought then occurred: Could some substance be used to prevent the action of the ferrocyanide upon the persulphate? Reflecting on the behavior of the alkaline hypo-halogen compounds toward KCN, I recalled the weak action exhibited by sodium hypoiodite upon KCN in comparison to that exhibited by sodium hypochlorite and sodium hypobromite; the latter compounds almost immediately converted the KCN into KCNO in alkaline solution—in acid the solution formed cyanogen bromide, cyanogen chloride, respectively.

Guided by this process of thought, the idea of using a soluble iodide, together with the persulphate, in the cyanide solution at once suggested itself. I forthwith tried the effect of potassium iodide in conjunction with persulphate and cyanide solution on gold tellurides, with the result that the tellurides underwent complete dissolution. The nature of the reaction at once explained itself: There being no alkali present in any of these tests, other than the alkali occasioned by the hydrolysis of the cyanide solution, it was simply the persulphate liberating the iodine in presence of the cyanide solution to form cyanogen iodide. The following equations may represent the action shown in two stages:

$$2\,KI + K_2S_2O_8 = 2\,K_2SO_4 + I_2,$$

then

$$I_2 + KCN = ICN + KI.$$

From the study of these reactions I conceived the idea of liberating iodine while the ore was in contact with the cyanide solution by the use of a substance yielding nascent oxygen. Any oxidant might be used which has not a too high velocity reaction, and one which does not unduly increase the alkalinity of the solution so as to prevent the formation and existence of cyanogen iodide in the said solution. Persulphate apparently filled these conditions, as it introduced no free alkali while giving off nascent oxygen, but, on the contrary, neutralized alkali owing to its anxiety to part with one of its sulphions (SO_4) as:

$$K_2S_2O_8 + 2\,KOH = 2\,K_2SO_4 + (OH)_2,$$

*A paper read before the New York section of the American Electrochemical Society.

or we may write the equation in the following manner:

$$2\,KSO_4 + 2\,KOH = 2\,K_2SO_4 + H_2O_2.$$

It will be seen from the above description that the solutions would contain free cyanide together with cyanogen iodide and the oxidizing agent (persulphate), the proportion of cyanogen iodide to cyanide depending upon the speed of the reaction and ratio of persulphate to potassium iodide.

To apply this solution to the treatment of ore and maintain the chemical equilibria necessary for producing cyanogen iodide required very skillful manipulation. For example, the ore had to be made alkaline by applying the necessary protective alkali to the cyanide solution, if excess of alkali was added the persulphate would still liberate iodine, but the iodine so liberated immediately formed alkaline hypoiodite, the hypoiodite formed was rapidly destroyed by the sulphides contained in the ore, thus rapidly using up the oxidizing energy of the oxidizing agent

The amount of alkali to be added depended, firstly, on the acidity of the ore; secondly, on the proportion of persulphate added, the persulphate requiring alkali in proportion to the removal of its SO_4; thirdly, there was influence of time on the rate of progress of the reaction; fourthly, there was the influence of atmospheric oxygen during agitation accelerating the oxidation of sulphides, which developed acidity and required alkali. To control this alkalinity sufficient alkali had to be added, but not enough to destroy the existence of cyanogen iodide; again, the cyanogen iodide formed would, in the presence of alkali and sulphides, form sulphocyanides, representing a direct loss of cyanide.

Nor is this all; when agitation treatment was completed, the solution after leaving the ore preparatory to filtration would drop, i.e., show a greater cyanide consumption than when in contact with the ore. This consumption of cyanide was probably due to the cyanogen iodide formed, together with the remaining persulphate, acting upon the cyanide solution in absence of the retarding influence of the reducing agents in the ore.

When it was necessary to add alkali to the solution in order to secure the proper precipitation in the extractors, a further consumption of cyanide took place after the equation:

$$CNI + 2\,KOH = KCNO + KI.$$

I discovered that this consumption of cyanide could be completely stopped, and the cyanide which had combined to form cyanogen iodide regenerated, by the simple addition of sodium sulphite (Na_2SO_3) and lime according to the equation:

$$CNI + Na_2SO_3 + H_2O = NaCN + NaI + H_2SO_4,$$

while the lime added neutralized the acid produced. The addition of sodium sulphite acted perfectly at this point, but it did not take care of the decomposition of cyanogen iodide to alkaline cyanate produced by the necessary presence of alkali in the ore-treatment process (a measure, of course, not allowable in the treatment tank).

It is obvious that the above process would require a skilled manipulator to obtain results at a reasonable cost. In a word, the process is not foolproof However, if the market price of persulphate would drop to a reasonable figure and a cheap salt were used which would undergo hydrolysis, i.e., give hydroxyl ions in solution to the extent of taking up the acidity produced by the oxidation means employed, then the process would require no more attention than the ordinary straight cyanide process

Ozone as an Oxidizing Agent.

In view of the high cost of persulphate, my attention naturally turned to the use of a cheaper oxidizing agent, and one that would not introduce, as before stated, alkali in the solution. Ozone, therefore, claimed my attention, as previous experiments with ozone gave interesting results. As there were several ozonizing machines on the market at that time, I began to look around for the most economical machine which produced the greatest amount of ozone at the lowest cost. One machine in particular was guaranteed and backed by a reputable, well-known firm to produce from 100 grams to 150 grams of ozone per kw-hour, and figuring the cost of electrical energy at 1 cent per kw-hour, it seemed that I had now struck the right nail on the head as regards a cheap oxidizer.

An ozonizer of the above description was immediately purchased and installed in my laboratory, whereupon I at once set to work to test the action of the ozone on the treatment of ores with cyanide solution containing soluble iodides, using the ozonized air produced by the ozonizer to liberate the iodine from the necessary chemicals while in contact with the ore.

It would take up too much space to give in detail the results of the numerous tests made on different samples of ore. In short, the ozonized air acted as a first-class oxidizing agent and gave excellent results. The only thing against this scheme was that it consumed too much cyanide unless very carefully watched, it being difficult to establish the proper equilibrium necessary not to allow of rapid oxidation to cyanates. The equation representing the action of the ozone upon the iodide was as follows.

$$O_3 + 2\,KI + H_2O = O_2 + I_2 + 2\,KCH,$$

and, of course, the iodine combined with the cyanide to form cyanogen iodide, as already shown in one of the foregoing equations, the excess ozone forming with the cyanide KCNO after the equation

$$O_3 + KCN = KCNO + O_2.$$

The question now was how to prevent this formation of KCNO

Learning from previous tests on ores containing large amounts of reducing agents that the consumption of cyanide was not so great as with ores which did not contain reducing agents, the thought suggested itself to add a reducing agent to the cyanide solution. I knew that potassium sulphocyanide, although in a sense a reducing agent, had no action on straight cyanide treatment, i.e., the atmospheric oxygen was not absorbed by it. Experiments showed that when sulphocyanide was subjected to ozonized air, cyanide was generated according to the equation:

$$KCNS + H_2O + 3\,O_2 = KCN + H_2SO_4$$

The alkali present, of course, neutralized the acid as produced, and the cyanide so formed was not decomposed as long as the sulphocyanide was present.

This action opened up a field for thought, and in the ordinary course of thought I tried the action of ozonized air on sulphocyanide solution containing an iodide. Upon introducing the ozonized air into the vessel containing the sulphocyanide and the iodide solution free iodide was at once liberated. When this experiment was repeated, using part of the same original solution, but with a small addition of alkali, on again passing the ozonized aid through the mixture, the solution immediately assumed the characteristic iodine coloration.

This solution was exceedingly active on gold leaf and on telluride of gold, the presumption being that the solution contained cyanogen iodide and free cyanide. The formation of cyanogen iodide by this means, I presume, may be represented by the equation:

$$KCNS + KI + 4\,O = ICN + K_2SO_4$$

Following up this wealth of discovery, as shown by the above equations, I proceeded with my tests, using the sulphocyanide to prevent the direct formation of cyanate, and was rewarded with most satisfactory results, the consumption of cyanide being very much reduced. However, the amount of cyanide consumption which did take place over and above that consumed by the ore as compared with straight cyanide was probably occasioned by the cyanogen iodide formed being oxidized slowly to cyanate in presence of alkali; the speed in this reaction was not great and probably occurred after the following equation:

$$5\,CNI + 6\,KOH = 5\,KCN + KIO_3 + 2\,I_2 + 3\,H_2O.$$

Potassium hypoiodite (KIO) being formed as an intermediate product, the excess of iodine acting upon more alkali forming

hypoiodite would destroy the KCN formed and eventually become oxidized to KCNO.

Here was a field for the expansion of the process on the large scale, but there is almost always a "but." While the production of ozone was apparently cheap enough as regards the cost of electrical energy, the cost of the machine (installation) to produce enough ozone to treat, say, 1000 tons of ore per day was staggering. The machines required the supervision of trained experts, and for a 1000-ton per day plant a battery of about 500 unit ozonizers would have been required, beside various other mechanical objections too numerous to mention.—"Another hope blasted."

The production of ozone by electrical means is yet in its infancy, and it is only within the last few years that ozonizers have been put on the market to give ozonized air of high concentrations. In the very near future it is probable that ozonizers will be built to give large volumes of ozonized air at high concentration with a small installation of ozonizer units. Meanwhile, I had to look around to find a substitute. How was it to be found? I had already exhausted almost the whole list of economical oxidizers known to chemistry.

Electrolysis.

The employment of the electric current claimed my attention as a means of producing the oxidation necessary while the solution was in contact with the ore. Electrical means for the treatment of ore labors under a stigma, inasmuch as there have been unfavorable criticisms of the use of electrolytic processes, no doubt caused by the numerous failures, among which may be mentioned the electrical precipitation of gold requiring the installation of an enormous plate surface, electrode difficulties and the consequent destruction of the cyanide by the use of the current.

One of the most severe criticisms relates to electrolytic processes in which the direct preciptation of the values from the ore pulp is attempted, the objections probably being due to the scouring of the plates by the circulating ore and the consequent loss of finely divided gold amalgam in the ore pulp.

Much more could be said on this subject, but the Clancy process is not concerned with any electrolytic amalgamating scheme or deposition of the gold on plates or depending upon the dynamical theory of the gold being brought against the cathode electrode and there meeting with a nascent hydrogen or the ore being brought against the anode electrode and meeting its nascent oxygen. The process thus differentiates itself into a method of using the electrical energy to disassociate the chemicals employed and by their agency doing the necessary work.

Electrodes.—Obviously the use of electrolytic methods requires proper anodes. Numerous substances to act as anodes have been tried, such, for example, as lead peroxide, carbon, iron, graphite, etc., but all these substances have been more or less inefficient

Anodes of lead peroxide under the influence of high current densities break down, i.e., disintegrate. Iron when used as an anode is oxidized by nascent oxygen liberated and forms hydrated oxides of iron, and should the cyanide solutions contain halogen salts the corresponding iron halogen compounds are formed. It is needless for me to describe the many difficulties which I have been up against to find a suitable anode. All the common metals, together with different alloys and compounds, were experimented upon to find a substance which would withstand the action of the halogen compound, i.e., some substance that would not combine with either chlorine, bromine or iodine under the action of the current, so as to allow the liberated halogen or the nascent oxygen to do the useful oxidation work in the solution.

Graphite (extruded) stood the electric current fairly well. i. e., makes a good anode while it lasts, but, under the influence of high current density, it disintegrates and cannot be used for any length of time in circulating solutions containing ore in suspension, as it is worn away by the attrition or friction of the ore particles against the said electrodes. Nor is this all; when it is understood that graphite will not stand a much higher density than, say, 10 amp per square foot of surface, it requires no calculation to figure out the amount of electrode surface required on a working scale.

The only metal which would stand up under the electrolysis of cyanide-bearing solutions containing halogen compounds was platinum. But on a commercial scale the employment of platinum could not be entertained for one moment.

After trying out every known electrode which I thought might prove suitable, I found that oxide of iron which had been melted in the electric furnace was the only thing on this side of Jupiter that would stand up under high current densities and not combine with the halogen compound contained in the cyanide solution. The area of anode surface of this material to give the necessary current density is reduced to a fraction of the amount of surface required in the case of graphite.

In general, these melted iron oxide electrodes used as described are particularly suitable for the electrolysis of cyanogen-bearing solutions and present the following advantages over all other anodes, including platinum:

First, they are cheaper, they have an unlimited life in comparison with the other anodes already mentioned. Second, they furnish nascent oxygen free from carbonic acid, which is of great advantage in the electrolysis of cyanide solutions. Third, they do not disintegrate under high current densities and are so hard that they do not wear away by the atrition of the ore pulp in the agitating tank.

Armed with the necessary weapons in the shape of these highly inert refractory electrodes, confidence in my ability based on previous experiments with platinum electrodes, I felt sure that I could do anything in the shape of electrolysis of cyanogen-bearing solutions containing halogen compounds. I could now use high current densities and liberate chlorine, bromine or iodine in the said solutions without worrying about my anode breaking down.

It will be remembered in my earlier experiments that I was confined to the use of iodine or soluble iodide, as no other oxidizer, even ozone, would liberate bromine or chlorine in the alkaline cyanide solution. Taking advantage of these new-found electrodes I experimented with cyanide, sulphocyanide and a soluble bromide, and obtained excellent results on the most refractory ores by keeping the solutions at the neutral point so as to allow of the formation of bromocyanogen while in contact with the ore.

The use of a bromide, as above described, although giving results equivalent to the use of soluble iodide, was more difficult to control owing to the necessity of maintaining the solutions at the neutral point, and required more electrical energy to do the same work as iodide. Again, by keeping the solutions at the neutral point the formation of cyanate was much more rapid, consequently the free cyanide did not coexist very long in presence of the cyanogen bromide, owing to its high speed of reaction. The bromates produced by the oxidation action would accumulate in the solutions and were apparently not reduced to bromides even after passing through the zinc boxes.

In the case of using a soluble iodide any iodates formed were at once reduced by the action of the sulphocyanides. This reaction between sulphocyanide and an iodate is very striking, so much so that if we take a solution of an iodate and add to it a solution of sulphocyanide, on acidifying this solution, immediate liberation of iodine takes place, or, again, if we take a solution of iodate containing sulphocyanide and electrolyze same, iodine is immediately liberated at the anode.

With bromates this is not the case. Bromates are reduced more or less by the nascent hydrogen evolved at the cathode, but this nascent hydrogen has to do other work, in fact, it has no time to look after the complete reduction of bromates, as during electrolysis a thin pellicle of calcium hydrate is produced on the cathode. The formation of this pellicle is occa-

stoned by the presence of calcium salts in the solution. This pellicle scales off, but is immediately reformed, thus more or less preventing the hydrogen gas in its atomic state from coming into contact with electrolyte and will cause a union of the atoms to molecular hydrogen. When the hydrogen has once assumed the form of molecules then its reducing power is at an end. Bromates, therefore, have a chance of partial reduction in the treatment tank.

The advantages possessed by the use of iodine over bromine are the following. Iodine is easily liberated from its compounds at a very low voltage; it does not react violently upon the cyanide solution; it is continuously reduced from its higher oxidation state by the sulphocyanides present in the solutions; it can be liberated from its compound in alkaline solutions, which fact prevents the loss of valuable iodine due to volatilization.

The fact that iodine forms cyanogen iodide in a slightly alkaline solution is of extreme importance in the treatment of ores, as by this fact sufficient free cyanide coexists during the formation of cyanogen iodide. The decomposition of cyanogen iodide to cyanate in the presence of alkali takes place slowly, thus allowing ample time for the precious metal's dissolution.

This method of using cyanide sulphocyanide and a halogen compound in conjunction with the electrolysis of the solution gave excellent results on most of the so-called refractory ores. The cyanide could have been regenerated from the KCNS (i.e., the KCNS which had formed due to the action of the cyanogen iodide upon the sulphides during treatment) but for the presence of the halogen compound.

The reaction $KCNS + 3O + 2KOH = KCN + K_2SO_4 + H_2O$ takes place only to a limited extent. The presence of the halogen salt handicapped this reaction, i.e., it finally became more or less a reversible one after the following equations:

$$KCNS + KI + 2KOH + 4O = KIO + K_2SO_4 + KCN + H_2O,$$

$$KIO + KCN = KCNO + KI,$$

$$KIO + KCN + K_2S = KCNS + KI + K_2O.$$

In a word, the formation of KIO oxidizes the KCN to cyanate, and when regenerating the solution in the presence of ore the KIO acts upon the sulphides in the presence of alkali, producing KCNS.

It will be seen that it is difficult to establish the correct ratios of alkali to obtain an equilibrium to allow of the formation of KCNS and prevent the formation of cyanate. How could this formation of cyanate be prevented? How could the cyanogen-bearing solution be regenerated?

Numerous experiments made with all kinds of additions conducted upon scientific principles with a view to reducing the cyanate to cyanide gave most discouraging results. As far as I am aware, there is no record in technical literature in reference to the regeneration of cyanide from solutions of cyanate. Therefore, I had to launch out into a sea of speculation.

My first experiment was to try the action of the electric current upon a chemically pure solution of potassium cyanate. The action of the current upon straight cyanide (as is well known) results practically in the formation of cyanate. I, therefore, tried the effect of continued effect of electrolysis upon a pure solution of KCNO.

It was not my intention to chase any scientific butterflies, but only the question as to what were the products of electrolysis. I forthwith proceeded with my experiment. On examination of the cathode product NH_3 revealed its presence by its action on the olfactories. At the anode I collected CO_2, the solution meanwhile increasing in alkali. These products for a start were by no means encouraging.

However, as a preliminary to an intended systematic research and to see if there was any possibility of some reduction to cyanide, I withdrew a portion of the liquid from the containing vessel and floated some gold leaf upon the surface of the solution and set it aside for observation, meanwhile continuing electrolysis of the remaining solution. After about 10 minutes this solution became very hot; I was about to throw it out as a spoiled experiment, but, thinking there might be some intermolecular change due to the increased temperature—having in mind among other things Wohler's discovery, viz., conversion of ammonium cyanate into urea—I withdrew a second portion of the solution and floated gold leaf upon its surface, placing it alongside the first experiment.

I watched both solutions carefully. No. 1 experiment, which had already stood 15 minutes, showed no signs of dissolving; No. 2 experiment after 10 minutes surprised me; the gold was surely being dissolved; in 30 minutes it was completely dissolved. No. 1 experiment, after standing all night, showed no signs of dissolution.

Thinking that the reason why No. 1 did not dissolve gold leaf was owing to its not having the same amount of current as No. 2, I set to work and made two experiments under precisely the same conditions, with the exception that one was kept cold by immersion of the containing vessel in a freezing mixture, the other one was allowed to become warm as before. After passing through both the same number of amp-hours of electricity I floated gold leaf on both solutions. On 30 minutes' standing the one that was allowed to become hot dissolved gold leaf completely; the one which was kept cold showed no signs of dissolution even after three days' standing. Platinum electrodes were used in both experiments.

What was the cause of the difference between the electrolyzed hot and cold solutions? Was it the formation of urea? A solution of potassium cyanate containing a small portion of urea I at once electrolyzed, adding a small amount of caustic soda to increase the conductivity; by this means the solution kept at about room temperature. On completion of electrolysis gold leaf was floated upon the surface of this solution and allowed to stand to await results. In about 15 minutes the gold leaf underwent complete dissolution.

From this it would appear that the urea theory was probably correct; or was it the cyanate in conjunction with the urea together with the action of electric current producing cyanide, isopurpurate or some other gold-dissolving compound? Testing with silver nitrate showed an apparently considerable quantity of KCN, but nothing commensurate with its dissolving value on gold leaf.

Cyanamide.

Having made the discovery that urea in conjunction with the cyanate and electrolysis of the solution dissolved gold leaf, the next question was, would any similarly constituted organic compound do the same thing? Urea being an amide compound (carbamide), with the speed of lightning came the thought of cyanamide. I thereupon added a solution of cyanamide (using the calcium cyanamide of commerce, which is soluble to the extent of about 58 per cent to 60 per cent in water) to a cyanate solution, electrolyzed same, treated the solution as before with gold leaf, and dissolution of the gold took place in a few minutes.

With a view to studying this peculiar reaction, I tried the action of electrolysis on a solution of calcium cyanamide; without any admixture other than the addition of a little alkali to increase the conductivity, after electrolyzing for 10 minutes at a fairly high current density the solution dissolved gold leaf in about 15 minutes; this solution also gave a substantial titration with silver nitrate showing probably that cyanide had formed.

Again treating the same solution with the addition of potassium thiosulphate and boiling for the purpose of converting any cyanide present into sulphocyanide, on testing this solution with ferric chloride it gave the characteristic blood-red coloration of the iron sulphocyanide—surely ample verification. But to make it more confirmatory I added a solution of ferrous sulphate to convert any cyanide in the solution into ferrocyanide. On addition of ferric chloride it gave on boiling the characteristic prussian blue.

This satisfied me that some cyanide was formed by the electrolysis of calcum cyanamide, but as to the amount produced

from a given quantity of cyanamide I was in the dark, owing to the peculiar behavior of the titrating solution in presence of organic compounds. It seemed from different experiments made that the production of cyanide by electrolysis of cyanamide solution never rose above three-tenths to four-tenths of a pound KCN per ton of solution.

During all these tests I used solutions of analogous strength to working-scale practice. On referring to Ward's "Dictionary of Chemistry," page 314, line 65, Vol. 2, I found the statement in substance, that cyanamide combines with potassium cyanate to form mono-potassium amidodicyanate of the formula $(CN)_2 NH_2 OK$. Reflecting on this reaction I thought probably that, as the cyanide was formed by the electrolysis of the cyanamide, it was oxidized by the current to cyanate, the cyanate then acting upon the unaltered cyanamide to form potassium amidodicyanate, and potassium amidodicyanate when electrolyzed forms a powerful gold solvent, although it does not give any appreciable titration with silver nitrate.

It will be, I trust, appreciated that to go into the discussion of these reactions would not only require an enormous amount of space, but prove too severe a tax upon my time. Suffice it for the present to say that I have a number of notebooks filled with the most interesting data in regard to these reactions, showing how they branch off and become diversified. I, therefore, propose later on to publish the results of the different experiments and give the theory relating to the data of my notebooks.

To go into practical work is more to the point of this article. However, before leaving the theory, which is always the necessary scaffolding for building up an edifice, I may state that when a solution of calcium cyanamide is mixed with a solution of alkaline ferrocyanide and allowed to stand for a few hours, and even after weeks' standing, it becomes an active solvent for gold, even without the aid of electrolysis, and when applied to the treatment of ores amenable to straight cyanidation gives results equivalent to the simple cyanide treatment even in very dilute solutions.

Further this process can be used on ores which have already been treated by the cyanide process; that is to say, on the residues commonly known as dumps. There exists in these dumps a large proportion of prussian blue, which, when treated with alkali, becomes in substance potassium ferro and ferricyanide or soluble prussiates.

This opens up a cheap means of treating ores amenable to the ordinary cyanide process as well as ores—residues—which have already been treated by the cyanide solution. That is to say, ores in which the precious metals have not been wholly extracted, and would not pay for retreatment by the ordinary cyanide process on account of the cost of cyanide.

The following is an example of using this process upon ore amenable to cyanide treatment:

Two thousand pounds of water containing 1 lb. of calcium cyanamide, 1 lb. of alkaline ferrocyanide, 1 lb. of lime. The ore is subjected to this solution in the ordinary way, say, in proportion of two parts of solution to one part of ore, for a period of, say, 8 to 10 hours, or until extraction is complete.

The following is an example of using the process upon ore which has already been treated by the cyanide process and exposed to atmospheric oxygen which occasions the formation of prussian blue and other ferrocyanogen compounds.

This ore when treated with a solution containing 1 lb. of cyanamide, 1 lb. to 5 lb. of lime (the amount of lime, of course, depending upon the acidity of the ore) in 2000 lb. of water gives results equivalent to the use of straight cyanide solution when used in the proportion of two or three parts of solution to one part of ore.

Again, if this mixture is electrolyzed the solution of the gold is extremely rapid. Calcium cyanamide mixed with alkaline sulphocyanide, electrolyzed, immediately becomes cyanide and dissolves gold rapidly. To show the action of another amide compound, for example, guanidine carbonate mixed with potassium cyanate in conjunction with electrolysis of the solution dissolves gold leaf in less than 10 minutes. I might go on here and cite a whole list of amidogen compounds in conjunction with cyanate solution if space would permit, and I have no doubt this would be interesting to those following this subject.

In the midst of this work on cyanogen-bearing materials the thought occurred to try the combination of calcium cyanamide and iodine. I forthwith made a solution of calcium cyanamide, adding a solution of iodine to see if this combination would dissolve tellurium, and was awarded by finding it to be an excellent solvent. I next tried gold leaf on the same combination and had the satisfaction of seeing the gold undergo dissolution in a short time—a scum of calcium carbonate forms on the surface, preventing more rapid dissolution. From this I deduced that the addition of iodine formed cyanamidogen iodide.

I am unable to find any reference to this compound in chemical literature which I have read, and I, therefore, take the liberty of calling it by the above name and further presume the formula to be CN_2I_2. I next tried the action of cyanogen iodide upon a solution of calcium cyanamide and knowing that cyanogen iodide would be destroyed by an excess of alkali, anticipating that this combination woulld dissolve telluride of gold. Upon applying this solution to tellurium, telluride of gold, these substances at once dissolved, showing that the dissolution was probably due to cyanamidogen iodide, and I again presume the reaction to be in accordance with the formula

$$2\,ICN + CaCN_2 = Ca(CN)_2 + I_2CN_2.$$

Practice of Process.

I will now describe the general method of using the Clancy process on the working scale. The ore is crushed by stamps, rolls, ball mills or any other efficient form of preliminary grinding mills. The degree of fineness necessary for the process being about 100 mesh, this degree of fineness is probably most economically accomplished by the use of tube mills.

The ore is crushed in the cyanide solution containing calcium cyanamide, sulphocyanide and the halogen salts; it is, therefore, under the influence of straight cyanide treatment practically after leaving the rock breakers until it reaches the agitation tank. In the agitation tank it meets with electrolysis, which acts upon the cyanide containing the additional chemicals in solution, forming powerful solvents for the precious metals as already described.

Before describing the treatment solution the manner of dissolving the cyanamide should be known. Calcium cyanamide of commerce comes as a black powder and is about from 58 per cent to 65 per cent soluble in water. It is necessary, therefore, to dissolve the cyanamide in a separate tank and filter from its insoluble residue. A small air-agitating tank is admirably suited for this purpose, which at the same time will act as a cyanamide storage tank. The cyanamide solution may be made as highly saturated as desired and the calculated quantity drawn off when required and added to the cyanogen-bearing solution.

The treatment solution is made up to 2000 lb. of water containing 1 lb. of cyanide, 2 lb. of alkaline sulphocyanide, 2 lb. of calcium cyanamide, ¼ lb. of alkaline iodide, 20 lb. commercial sodium chloride.

I use, after the rock breakers, say, for example, crushing rolls. The product leaving the rolls about 12 mesh is fed into the tube mill and converted into a pulp by feeding the mill with the treatment solution and ore in the proportion of two or three parts of solution to one part of ore.

The discharged pulp after separation of the over size is transferred to the agitation tank to undergo electrical treatment. If it is thought necessary to remove the sulphides before, after or during the treatment the following method presents an ideal scheme:

Let solution containing finely divided ore in suspension be contained in the well-known cone-shaped tank and agitated in

the proportion of two of solution to one of ore, or in the proportion of three of solution to one of ore; if the agitation is then stopped for a few minutes the finely divided sulphide particles settle to the bottom of the cone and by simply opening the discharge valve at the cone apex the sulphide may be completely drawn off together with a small proportion of the non-sulphide pulp. The sulphide product may be run over blankets or similar contrivances and the finely divided concentrate collected. The excess of pulp solution is returned to the agitation tank for treatment.

Here is a means to eliminate the use of concentrating tables and obtain a product of very high value in a very small bulk. Again, the pulp being in a very finely divided state the pyrite or sulphide portion is not accompanied with quartz or gangue —therefore, a clean high-grade concentrate—which result is not capable of accomplishment by the use of concentration tables without the employment of a large quantity of solution with its attendant expense.

The pulp now in the agitating tank carries the correct alkalinity, this being previously established in the tube mill by adding lime so as to contain from 1/10 to 2/10 of a pound "protective" alkalinity per ton of solution.

The conductivity of the pulp is adjusted by adding common salt until the required voltage is obtained. Twenty pounds of salt per ton of solution, as already stated, will invariably decrease the resistance of the pulp so that the voltmeter will register about from 5 volts to 6 volts. In the majority of cases a current of about 50 amp per ton of ore is adequate. It will be easily seen that the cost for electrical energy is not by any means prohibitive.

With iron oxide electrodes it is possible to obtain a current density considerably over 50 amp per square foot of anode surface so that one electrode 3 ft. in length by 3 in. in diameter will be sufficient for the treatment of from 3 tons to 4 tons of ore In other words, approximately thirty of these iron oxide electrodes would be required for the treatment of 100 tons of ore per day.

If the treatment tank is constructed of iron the tank itself may be used as the cathode. This arrangement would, of course, decrease considerably the cost of installation. The electric generator is the chief item of cost, but a low-voltage generator, say, a 10-volt machine, capable of giving the necessary amperage can be obtained at any of the electrical warehouses.

It is obvious, therefore, that the process may be applied to any existing fine grinding plant provided with agitating tanks. All that is necessary is simply to introduce the electrodes into the circulating ore pulp containing the necessary chemicals and switch on the current. It is essential in every case to maintain the alkalinity, i. e., protective alkalinity at about 1/10 lb. alkali per ton of solution so as to allow of the formation of cyanogen iodide cyanamidogen iodide. About eight hours' treatment under electrolysis usually being sufficient to obtain the necessary extraction

Regeneration of the Cyanide.—After the eight hours' treatment with the current the pulp solution is brought up to about 1 lb. per ton of protective alkalinity by adding caustic soda and the cyanide contents regenerated up to about 5/10 lb. or 6/10 lb. cyanide per ton of solution. The regeneration of the cyanide is then accomplished simply by giving the pulp two hours more current It will be understood that the reason for adding the extra alkali is that cyanide regeneration cannot take place in the presence of a halogen compound unless the solution containing sulphides, sulphocyanides and cyanamide is made alkaline

It will be seen that the whole value of the process depends upon the recovery of the halogen compound. While I have described in the examples a proportion of two parts of solution to one part of ore a proportion of three parts of solution to one part of ore may be used with advantage—that is to say, by using three parts of solution to one part ore a much smaller amount of alkaline halide may be used per ton of solution, thus giving the same ratio of halide salt per ton of ore as in the two of solution to one of ore pulp and consequently a less proportion of soluble halide to be displaced by the water wash in the final slimes cake.

Costs.—In this incomplete description of the process the use of chemicals and current have been dealt with, but no mention of costs has been made. I will, therefore, take the following solution to represent the typical working solution, 2000 lb. of water containing 1 lb. of cyanide, 2 lb. of sulphocyanide, 2 lb of calcium cyanamide and ¼ lb. alkaline iodide. This appears a formidable mixture when looked at cursorily, but on analysis it does not work out beyond the limits of economic treatment

For example:

1 lb. of cyanide	18 cents.
2 lb. of cyanamide	6 "
2 lb. of alkaline sulphocyanide	12 "
¼ lb. of alkaline iodide	35 "
Total	71 cents.

This would not represent the total cost of one ton of solution, for, notwithstanding the effect of electrolysis, practically all the haloid salt or salts previously added, together with the sulphocyanide, will be found impaired at the end of the operation, the cyanide and cyanamide alone suffering the necessary decomposition. It is clear, therefore, that no matter what the proportion of solution to ore only the consumption of cyanide and cyanamide per ton of ore is to be taken into account. The amount of cyanide consumption on the ore in presence of cyanamide works out at about 1 lb. of cyanide per ton of ore treated. This consumption of cyanide is regenerated at the expense of 3 cents for cyanamide and at the very outside 3 cents for current (figuring current at 1 cent per kw-hour), making a total cost of 6 cents per ton of ore.

Added to the above cost is the cost of the electrical energy necessary for the electrolysis of the ore pulp. The cost of electrical energy for this purpose works out at about 10 cents per ton of ore treated. This added to the cost for cyanogen-bearing material and regeneration would make a total of 16 cents per ton of ore. These figures would represent the total cost provided that all the solutions were recovered without mechanical loss. From this it is evident that the recovery of the solutions for reuse is a matter of vital importance to this process. The mechanical recovery of the solutions is therefore entirely dependent upon the efficiency of the filter employed.

The Moore filter eminently fulfils the requirements of the process for the recovery of the solution inasmuch as it gives perfect uniformity of cake both in thickness and porosity, which means perfect resistance, and perfect resistance guarantees perfect displacement. In fact, the business combination of the Moore filter with the Clancy process was originally brought about not from any arbitrary or usual business considerations, but because I, after the most rigid examination of all existing filter processes, learned in my demonstrations that this type of filter was the only one that could be depended on to recover practically all of the solutions.

This filter process is too well known to describe it here minutely. Suffice it to say, when the final step in the cycle of treatment is reached—that is, when the cake has been washed with barren solutions—the moisture (containing soluble salts) saturating the cake at this juncture may be completely water washed by giving it the requisite amount of water for displacement. The necessary amount of water for displacement is readily seen without calculation by the diminution of the water level in the wash tank. Further, this may be adjusted to a nicety at the will of the operator, obtaining results not capable of accomplishment by any other known filter. It has been necessary for me to mention the unique points of this filter so as to show how the solutions can be recovered with a minimum of mechanical loss and with a minimum of water

wash. In a properly constructed Moore filter, type A, the loss of solution under proper manipulation should not exceed 10 per cent of the total solution.

It should be understood that the above figures are based on the treatment of rebellious or refractory ores and that they would in all probability be much reduced when dealing with non-rebellious ores.

New York City.

Metallurgy in British Columbia in 1910.

BY E. JACOBS.

The most prominent features in metallurgy in British Columbia are those in connection with the smelting of lead and copper ores. Of nearly a dozen smelteries—one zinc, three lead and the remainder copper—only four are being operated at the present time, namely that of the Consolidated Mining & Smelting Company of Canada, Ltd., at Trail, smelting both lead and copper ores, and the respective copper smelteries of the Granby Consolidated Mining, Smelting & Power Company, Ltd., at Grand Forks; the British Columbia Copper Company, Ltd., at Greenwood (both in the Boundary district); and the Tyee Copper Company, Ltd., on Vancouver Island. There does not appear to be any immediate prospect of other smelting works being operated in the province.

Stamp mills, mostly for gold-quartz ores, have been erected and equipped in various parts of the province, including one with 80 stamps and cyanide plant, at Ymir; one with 46 stamps and cyanide plant at Fairview, Osoyoos mining division; one with 40 stamps and cyanide plant at Hedley; two with 20 stamps each at the Queen mine, Sheep Creek, and the Granite-Poorman mine, respectively, both mines being in the Nelson mining division; one with 15 stamps at the Jewel mine Boundary district; several with 10 stamps each, of which only that at the Second Relief mine, Nelson division, has been worked lately, and some with fewer stamps, chief among these last being that with four stamps at the Nugget mine, Sheep Creek.

Of more than a score of concentrating mills, all but two of which produced nearly $80,000 worth of gold in 1910, which have been equipped for treating silver-lead ores, and several of these for saving zinc as well, less than one-half have been operated during the year, and only two or three the year through.

Stamp Milling and Cyaniding.

The Hedley Gold Mining Company's 40-stamp mill at Hedley, Similkameen, is the only one meriting more than pasing notice at this time. The Ymir and Fairview plants have long been idle; the Granite Poorman, Queen and several smaller mills crush the ore and save most of the gold on the amalgamating tables, and send concentrates to the smeltery to recover that not lost in the tailing. The Jewel mill is provided with a Nichols slimer, but has not yet been run expect experimentally since its recent completion. Passing over all others some attention may be given to the Hedley plant.

Hedley Gold Mining Company's 40-Stamp Mill.—The Hedley mill was erected and for several years operated by the Daly Reduction Company prior to its acquirement last year by the Hedley Gold Mining Company. To date its aggregate of production has been of a value of between $2,500,000 and $2,700,000 from 231,000 tons of ore. A recently published return showed an average assay value of $12.18 for 29,385 tons of ore, and an average recovery of $11.25 per ton, the period covered having been eight months, January-August, of 1910.

The following description has been supplied: The stamps in the four 10-stamp batteries are 1050 lb. each. Formerly they dropped 6 in. at 106 per minute. Last summer changes were made giving a 7¼ in. drop and 99 per minute, one result being a considerable increase in the crushing capacity without requiring the 20-mesh screens to be changed. During recent months new equipment has been put in, the completion of which alters the milling practice to the following: Stamps crush to only 16 mesh, the pulp is classified by eight Bunker Hill screens, the coarse product being concentrated on eight corrugated belt vanners, and the vanner tailing pulverized to 100 mesh in a 100-ton Gates tube mill having Montana-Tonopah lining. The tube mill product unites with the fine from the Bunker Hill screens (running less than 60 mesh), and after being reground passes to classifiers and hydraulic sizers, the coarse from the latter is concentrated on 16 smooth belt vanners, the fine from the sizing passing to 10 Deister tables. The middling from these tables is concentrated over two similar tables. The tailing from the smooth belt machines passes to the sand vats, and that from the Deister tables to the slime vats, this being the method of separating sand from slime. None of the sand reaches the leaching vats coarser than 60 mesh. This new method provides for the single treatment of the sand, the proportion of which is but 25 per cent of the whole. The slime is collected in four 200-ton settlers and pumped thence into six 200-ton agitating tanks in which mechanical agitators are used. The gold solution is decanted from the agitating tanks and treated by two Merrill precipitation presses, in which zinc dust is used. The slime residue in the agitating tanks, remaining after decantation, passes to two Oliver filter presses, the solution to the precipitation presses, and the caked slime to the dump. The first cyanide being introduced into the sand vats and agitating tanks, there is no crushing in cyanide solution. The consumption of cyanide per ton of sand and slime treated amounts to ½ lb. and costs 10 cents per ton. The concentration amounts to 200 tons per month; its base is arsenopyrite, and it runs high in gold. It has been sent to the smeltery at Everett, Wash. The foregoing information is supplied by the superintendent, Gomer P. Jones, and is practically the same as he has presented elsewhere.

Charles Camsell, of the Canadian Geological Survey, in an official report, says: "As the ore is an auriferous arsenopyrite, gold is virtually the only product, and an extraction of about 92 per cent of the total value of the ore is made from amalgamation, cyanidation and smelting. A small percentage of copper, silver and some platinum is known to occur in the ores, and assays show a variable proportion of cobalt, nickel, lead and bismuth. The arsenic is a waste product, so far as the mines are concerned, and no credit is given for it by the smeltery."

About 47,000 tons of ore were milled in 1910, and from this gold to the value of approximately $516,000 was recovered. Power plant and new mill equipment put in during the year cost $115,000.

Silver-Lead and Zinc Concentration.

The Consolidated Mining and Smelting Company's concentrating mill at its St. Eugene mine, Moyie, East Kootenay, is the largest concentrator operating on lead ores in British Columbia. During seven years, 1904-1910, it has treated approximately 1,000,000 tons of St. Eugene ore, from which about 185,000 tons of lead-silver concentrate have been produced, of a total value of $10,250,000. The St. Eugene ore is of a silicious nature, lending itself readily to concentration, which has been about six to one, the concentrate averaging 66 per cent lead. The mill flow sheets show the following: Farrel crusher, picking belt, Blake crushers, rolls, Vezin sampler, elevators, trommels, bull jigs, trommels, rolls, three and four compartment jigs, Huntington mills, Callow screens, sloughing-off tank, Callow tanks, Wilfley tables and Frue vanners. Final middlings from the Wilfleys, containing 20 per cent zinc and 4 per cent lead, are stored for further treatment for zinc. The lead concentrate is shipped to the company smeltery at Trail. The capacity of the mill is 400 to 600 tons daily.

During 1910 an ore-testing plant was practically all put in

It comprises 7 by 9 Blake crusher, set of 15 by 25 rolls, automatic Vezin sampler, three trommel lines, two elevators, four-compartment Richards jig, two Hodge jigs, one each No. 4 and No. 5 Wilfley tables, Wilfley slimer, Overstrom table, Frue vanner, Behrend dry separator, Dings electric magnetic separator, four screens (Callow style), flotation tank, settling tanks, classifiers, etc. A Wetherill separator is being installed, and the putting in of a unit of the Murex magnetic process is contemplated. In addition, the mill is equipped for making cyanide and amalgamation tests. With the exception of these last-mentioned equipments, the mill is designed to make tests on large tonnages of ore, practically all the machinery and plant in it being of the standard sizes used in mills.

A concentrator of 200 tons daily capacity was running at the Blue Bell Mine, Anisworth division, during 1909 and part of 1910. It was designed to treat a large body of ore—more than 300,000 tons—the sampling of which showed an average content of 10.6 per cent lead, 11.7 per cent zinc, and 3.27 oz. silver per ton. This ore was estimated to carry: Quartz and other gangue, 30 per cent; iron sulphides (chiefly pyrrhotite), 35 per cent; zinc blende, 22 per cent; galena, 13 per cent. The products were to be, respectively, lead, zinc and iron concentrates. After leaving crushers and rolls, the ore was elevated to a trommel line, passed thence to two and five compartment jigs and Huntington and Ferraris mills, and the lead pulp thence to Callow screens and tanks and to No. 5 Wilfley tables. The intended separation of the zinc from the iron was not generally successful, so a marketable zinc product did not result. Between 70,000 and 80,000 tons of ore have been milled and more than 6000 tons of concentrate produced, the silver and lead contents of which, however, fell short of the proportions indicated by the above-mentioned sampling.

Of the several mills at Whitewater, Sandon and near Silverton, respectively, that have for several years produced both silver-lead and silver-zinc concentrate, only the last will have notice. In an official report it is thus briefly described: "The Wakefield mill is a well arranged and well ordered plant driven by water power. Jigs operating on 16 mm., 12 mm., and 6 mm. material produce a lead concentrate only, the middlings being recrushed. Jigs operating on 4 mm. material produce a lead and zinc concentrate, the middlings going to a Huntington mill. All smaller sizes are classified, and go to seven Wilfley tables to produce both lead and zinc concentrates. The first Wilfley also makes middlings, which are recrushed. This mill has been operated under lease by the Van Roi Mining Company (Ltd.), of London, England, but its lease having expired, the company is now erecting and equipping its own mill in the same neighborhood.

Concentration of copper ores has been in progress at the Le Roi No. 2 Company's mill, Rossland, and that of the Britannia Mining and Smelting Company on the coast in New Westminster Division. The Le Roi No. 2 mill has a capacity of 80 to 100 tons a day, but usually only half that quantity is put through it. The mill plant includes three Blake crushers, two 6-ft. Chilian (Trent) mills, and four Wilfley tables operated by electric motors. The ore is hand picked; the first-class ore is shipped to the Trail smeltery, while the second-class, constituting about 35 per cent of the whole, is milled. The latter averages 0.4 per cent copper and 0.085 oz. gold per ton, and yields a concentrate containing about an ounce of gold per ton and 2 per cent copper, with an extraction of 75 per cent of the gold and 20 per cent of the copper.

The machinery and plant put in its mill several years ago by the Britannia Company included forty Hartz jigs, Hancock and Wall jigs, Blake crushers, Gates rolls, Huntington, Chilian and Monadnock mills, Overstrom, Wilfley, and Cammett tables (36 in all); 40 Johnston and Frue vanners, Sperry slimers, Richards classifiers, trommels, sand pumps, etc. Some time ago production was maintained at between 400 and 500 tons per day, but latterly the plant has been operated only intermittently, and then chiefly for the purpose of experimenting. In concentrating Britannia ores the result aimed at was the removal of the excess of silica, in order to make a desirable product for smelting with a minimum of copper loss, rather than to concentrate to a high grade of copper. The extremely hard matrix of much of the ore limits the application of concentration. Experience proved that the copper losses in the slimes, resulting from fine crushing, were too great, and that a large part of the ores contained mineral that could easily be separated from its gangue without fine crushing. The practice for some time was treatment which gave a concentrate about neutral as to its iron and silica contents, and from the picking belt a shipping ore carrying a large excess of silica. A modification adopted was to take from the picking belt both the high grade neutral shipping ore (to go to the smeltery crude) and the waste, and to leave the remainder to be crushed for concentration by jigs and on tables.

Smelting Ores Containing Copper.

Granby Company's Smeltery.—The Granby Company's smeltery at Grand Forks is stated to be the largest copper reduction works in the British Empire and among the biggest in the world, second only to the great Washoe works at Anaconda, Mont. It has eight rectangular water-jacketed blast furnaces, six being 44 by 266 in. and two 48 by 260 in. at tuyeres, depth 15 ft. 8 in.; capacity 4000 to 4500 tons of ore per day; 12 Connersville blowers, with a total air capacity of 239,000 cu. ft.; specially designed mechanical charging system; slag haulage by 14-ton steam locomotives and 44-cu. ft. slag cars; three converter stands, 84 by 126 in., operated by electricity, and one stand 72 by 100 in. for which hydraulic power is used, converter capacity being 36,000,000 pounds of copper per annum; three blowing engines, total capacity 19,700 cu. ft. of air per minute, and much accessory plant and machinery.

Conditions at the Granby Company's works are different to those obtaining at other copper smelteries outside the Boundary district of British Columbia in that the Granby Company has to treat an ore giving a very low percentage of sulphur on the charge—as low as 2 per cent. On the other hand, the ore contains silica, iron and lime in such favorable proportions that no barren flux, neither iron nor lime, is required. If it were necessary to use an iron flux in smelting Boundary district copper ores, it would not be practicable to mine and treat them at a profit unless the price of copper was much higher than it usually is, for the iron ore obtainable in the district in considerable quantity does not contain any marketable value. The slag from the Granby furnaces runs from 45 to 48 per cent silica all the time, while occasionally it has contained as high as 52 per cent, which has been the maximum reached. The ore is smelted as it comes from the mine. As a rule, it ranges from 40 to 48 per cent silica. Smelteries in other parts generally have ores running higher in sulphur than they require, but they have the advantage of a low silica content in the ore. It will be seen, therefore, that, as already stated, conditions under which the Granby Company's furnaces are ordinarily operated are quite different to those attending the reduction of copper ores elsewhere than in the Boundary district.

The low average of Granby ores makes it imperative that labor costs shall be kept down as much as possible, consequently wherever it is practicable to use them, mechanical devices are substituted for the labor of men. This is especially the case in feeding the furnaces, there being no mixing of the charge or other handling of materials between the ore and coke bins and the furnaces. Cars pushed by electric locomotives receive the ore and coke direct from the storage bins and dump them into the furnaces, the mechanical arrangements being such as to admit of this being done quickly and effectively. The company's engineers have devised many ap-

pliances for labor saving, so that economy in working methods is general throughout both mines and smeltery.

It is probable that in the near future changes will be made in the smeltering practice that will admit of a further saving being made in cost of treating Granby ores. The company has lately secured under option of purchase mines that it is believed will yield largely heavy sulphide ores. Heretofore it has been, and still is, the practice at the company's smeltery to produce a 35 to 40 per cent copper matte at first smelting, which involves a loss in the slag of about 5 lb. of copper to the ton of ore. With the sulphide ores above referred to as an aid to the proposed change, it is believed a saving will be effected, and as a consequence more profit be made out of the ores from the company's big mines at Phoenix, B. C. The ore will be smelted to a 10 per cent matte, which will be regraded to a 45 per cent matte for converting. It is thought that not only will the cost of converting to blister copper be less than at present, but by utilizing as fuel the heavy excess of iron and sulphur in the sulphide ores the percentage of coke required in the charge will be reduced. It is expected, too, that the copper loss in the slag will be less by 1½ to 2 lb. per ton of ore, which of itself would mean an appreciably large addition to the profit to be made out of ores from the company's mines at Phoenix. Mr. Wakely A. Williams, superintendent of the smeltery, has been with the Granby Company ten years, during which period he has effected many economies in connection with the handling and treatment of ores received at the works.

British Columbia Copper Company's Smelteries.—During 1910 the British Columbia Copper Company increased the total blast furnace capacity of its smeltery at Greenwood, Boundary district, by one-third, thereby bringing its maximum treatment capacity up to about 2600 tons per diem. This was done by increasing the hearth area of two of the three furnaces by 50 per cent. Previously all three furnaces were 48 by 240 in. at the tuyeres, and their approximate average total capacity had been about 1800 tons daily, with a maximum of nearly 2000 tons. Now they will average between 2400 and 2500 tons per diem.

This increase in smelting capacity has necessitated a corresponding increase in the plant for feeding the furnaces and taking away the molten slag; also an extension of the crane run at one end of the converter building. For slag hauling a larger locomotive, capable of handling two 25-ton slag cars at a time instead of one, as in the past, has been put in. Heretofore there have been in use two 15-ton Baldwin-Westinghouse electric locomotives for slag hauling and three 7½-ton locomotives for furnace charging. An additional charging locomotive has been put in to meet the increased capacity. The slag cars are 25-ton ladle cars, side dumping, and each is equipped with an electric motor having worm gear for tilting the car to dump its contents. The company was the pioneer in using this style of car for slag dumping. The design has since been adopted by the Canadian Copper Company at Copper Cliff, Ontario, and probably a dozen other smelting works, the latest having been the International and the Mason Valley companies in Utah and Nevada.

In the copper converting department of the works at Greenwood there are two converter stands with 84 by 126 2-in. horizontal shells, these taking matte of 45 to 55 per cent copper tenor and producing blister copper 99.3 per cent pure; also containing 20 to 30 oz. silver and 5 to 15 oz. gold per ton. This product is sent to New Jersey for refining. The stands are tilted by hydraulic accumulators, and the shells handled by a 40-ton, four-motor traveling crane. Two more converter shells have recently been added, making a total equipment of seven.

Electric power, transmitted from Bonnington Falls, distant more than eighty miles, is used in the works, in which there are installed motors having a total capacity of about 2000 hp. Among the motors are three of 300 hp, each belted to one of three large Connersville blowers, which supply the blast to the furnaces.

The company obtains the greater part of its ore supply from its own mines and those of the New Dominion Copper Company, which it controls, and it endeavors to maintain an average monthly output as follows: From the Mother Lode, 36,000 tons; Rawhide (New Dominion), 20,000 tons; Wellington Group, 3500 tons; Napoleon at (Boyd's Wash.), 2500 tons. Mother Lode and Rawhide are typical Boundary district copper ores—as they are quite low in marketable metals; Wellington group ore is chiefly oxidized, and having little value other than gold; of the Napoleon ore, the sulphide goes to the smeltery, where it serves as a pyrrhotite flex to reduce the tenor of the copper matte, while the oxidized ore is to be treated in a mill in course of erection near the mine. Provision is being made for increasing the ore supply from the company's mines, and thus render the smeltery independent of custom ores, if these be not obtainable to advantage. The Rawhide output is to be increased to 30,000 tons a month, while 3000 tons will be obtained monthly from the Lone Star, situated just south of the International Boundary line and eight miles from the works. The Lone Star mine is being connected with the Canadian Pacific Railway by an aerial tramway five miles in length; it has an available reserve of 300,000 tons of gold-copper ore of a higher average copper tenor than that from the company's other mines, besides being a good matte maker.

Coke is obtained from southwest Alberta, the railway distance between the colliery and smeltery being 370 miles. With three blast furnaces running, the consumption is 8000 to 9000 tons a month. A reserve of 10,000 to 12,000 tons is kept as an emergency supply in case of interruption in the regular receipt of coke from the colliery.

The approximate figures of production during 1910 are as follows: Material smelted, 456,000 tons. Contents: Gold, 25,640 oz.; silver, 85,900 oz.; copper, 7,351,000 lb. Included in the ore receipts were the following: From the Mother Lode Mine, 352,000 tons; Oro Denoro, 11,473 tons; Wellington Group, 19,200 tons, and Rawhide, 54,400 tons. Mr. J. E. McAllister is general manager, and the number of men employed is 500 to 540, of which total 118 is the normal force at the smeltery.

Tyee Copper Company.—This company's smeltery is situated at Ladysmith, Oyster Harbor, Vancouver Island. Custom ores, chiefly from various parts of the Coast district of British Columbia, southern Yukon, and southern Alaska, have, during recent years, constituted the chief part of the total supply obtained.

At a general meeting of shareholders held recently in London, England, the chairman said in part: "During the year the effects of the low price of copper were to seriously hamper the company's business, mine owners having no inducement to proceed with development, to ship ore freely or to open new mines. Nevertheless the company received 21 per cent more ore than in the previous year. The quantity received, however, had only sufficed to keep the furnaces running 188 days, but as the standing charges were the same whether the furnaces were fully or only partially employed the financial results afforded hope for improvement. Between May 1 and Aug. 1, 1910, the ore received was 19½ per cent in excess of the quantity for the corresponding period of 1909. The manager had reported from British Columbia that negotiations for the acquisition of an important group of properties had been almost closed." Since then it has transpired that the Cornell Mine, one of the Van Anda Group on Texada Island, B. C., has been leased by this company, and that the question of also taking over other property in the group is under consideration. The grade of the ore from the Cornell is shown in the following figures: Production in 1909, 10,178 tons; average assay value, copper (wet), 3.78 per cent; gold, 0.297

oz., and silver, 1.38 oz. per ton. During seven months of 1910 the tonnage was 4501 tons, and the average assay value: Copper, 4.47 per cent; gold, 0.407 oz., and silver, 1.73 oz. per ton. Transportation from the mine to the smeltery is by water, and the trip occupies but a few hours.

Lead Smelting and Refining.

As the Consolidated Mining and Smelting Company, of Canada (Ltd.), smelts both lead and copper ores, and also refines lead, brief notice of all its operations will be put under one subhead. Much of the information here given has been published before, but the present data have been specially revised and corrected by officers of the company.

The company's works are situated at Trail, on the Columbia River, distant from Rossland by railway between ten and eleven miles. Some 550 men are employed. Distinctive features here, in comparison with other reduction works in British Columbia, are the use of (1) the Huntington-Heberlein roasters and converters, and (2) the Betts electrolytic process for refining lead.

An idea of the extent of the company's operations may be obtained from the following figures, which are those for the calendar year 1910: Tons smelted, 445,210. Metal contents: Gold, 138,901 oz.; silver, 2,017,007 oz.; lead, 33,871,837 lbs.; copper, 5,282,139 lbs. The figures for the company's fiscal year ended June 30. 1910, were generally higher, especially in lead produced (42,368,816 lb,) while total value was nearly $400,000 greater.

Lead Smelting.—The lead ore supply is mainly heavy sulphide with comparatively little dry ore, the lead furnaces carrying a charge of from 35 to 45 per cent lead, as compared with the customary 10 to 15 per cent. More than 80 per cent of the ore received runs from 20 to 75 per cent lead, the remainder consisting of dry gold and silver ores, a small quantity of oxidized lead ore, and some pyritic mill concentrates. The lead beds are made up with heavy ores and concentrates only. They contain about 600 tons each, with 38 to 40 per cent lead. The sulphide, with lime rock, lead matte, etc., is treated in the Huntington-Heberlein roasters and converters. The resultant lead sinter is smelted in the blast furnace with dry ores, etc.

There are seven Huntington-Heberlein circular roasting furnaces 26 ft. in diameter, with hearths revolving once in three minutes. Stationary cast-iron rabbles work the ore through in about two hours. The making up of the roasting charge is important. An approximate analysis of the charge as now run is as follows: Lead, 40 to 44 per cent; iron, 10 to 13 per cent; silica, 8 to 11 per cent; lime, 7 to 10 per cent; zinc, kept under 10 per cent, if possible. Pyritic concentrates and matte form part of the charge. The converters number 24. They are 8 ft. 8½ in. in diameter, the diaphragm in four sections, bolted together. The time of converting varies widely with the charge, but averages three 10-ton charges in twenty-four hours, with a 10 to 12 oz. blast, which is decreased to 2 oz at the close. Properly made up, carefully fired, roasted, and converted, the product is fairly hard, yellowish or grayish yellow, showing some litharge, and is easily broken up with little fine material. In the furnace it runs with cool top and bright tuyeres.

Two lead blast furnaces are, respectively, 45 by 160 in. and 45 by 140 in. at the tuyeres. One of these has smelted as much as 260 tons in a day, but the average is about 190 tons of ore, beside by-products, matte, slag, etc., these constituting 20 to 30 per cent of the charge. The height of the charge is 17 ft. 6 in. above the tuyeres, and the blast used is 32 to 34 oz. The charge usually has about 85 per cent of Huntington-Heberlein roast, varying considerably with the ores, and some oxidized and silicious dry ore and lime rock. The coke is about 12½ per cent of the charge, not taking into account slag and easily melted by-products. An addition of 100 to 300 pounds of slag from the high-grade copper furnace is made to each charge to keep it open, make the furnace action uniform and reduce the percentage of zinc. The lead in the charge averages about 40 per cent, but it has been successfully run as high as 55 per cent. Sulphur is kept below 4 per cent, the lower the better, as the matte fall is less. The speed of running decreases markedly with a small increase of sulphur. From 30 to 40 per cent of the sulphur is eliminated in the blast furnace.

The slag is kept at about 31 to 33 per cent silica, 18 to 20 per cent lime, 24 to 30 per cent iron and manganese, 7 to 12 per cent zinc oxide, and 8 to 16 per cent alumina. It contains about 0.4 oz. silver and a little more than 1 per cent lead. With high zinc the lime is reduced to keep the slag fluid. It is not profitable to use silicious ores to bring the slag much above 33 per cent silica, as the furnace runs too slowly. Alumina gives no serious trouble. Iron and manganese may vary between the percentages above given without much difference in the furnace running. Scrap iron is added occasionally, although unnecessary when the charge is good.

The lead bullion flows from the side of the furnace into a kettle of fifty tons capacity, from which it is pumped into vertical anode molds. The matte and slag are tapped into a large receiver at the end of the furnace, the slag overflows into a granulating launder, while the matte is tapped from the receiver into pots. A single day's production of bullion has been as much as 130 tons, but about 75 tons is the present rate of production, with one furnace running. An average analysis of the bullion is: Copper, 0.22; manganese, nil; zinc, 0.098; antimony, 0.32; arsenic, 0.28; nickel, cobalt, and cadmium, nil; bismuth, 0.0133, and lead, 98.5 per cent; silver, 100 oz., and gold, 0.5 oz. per ton. A furnace is operated continuously for about seven months, then if accretions have accumulated sufficiently to make it profitable to blow it out and clean up the other furnace is blown in.

Lead Refining.—In the electrolytic refinery there are 240 tanks, each 3 by 8 by 3½ ft., arranged in cascades and pumps to keep the electrolyte circulating. Each tank has 20 anodes, which are spaced 4¼ in. apart in the tanks. Cathodes, consisting of sheets of pure lead, about 1-16 in. in thickness, are placed between and outside the anodes, 21 in each tank. The electrolyte used is lead fluosilicate with free fluosilicic acid, averaging 12 per cent acid and 5 to 6 per cent lead. The tanks are operated with a current density of 16 amp. per square foot of cathode area. The voltage averages 0.32 volt per tank, the contacts causing a loss of not more than 0.02 per tank. A tank is worked out in about eight days, and 15 per cent of the anodes, as scrap, goes back to the melting pot after having been washed. The cathodes are removed from the tanks, washed, melted, and cast into pigs of lead of 100 lb. each for the Chinese trade, and half that weight for other markets. An average analysis of 2,000 tons of lead ready for shipment was: Arsenic and bismuth, nil; zinc, 0.0005; silver, 0.0013; copper, 0.00075; lead, 99.9938; iron, 0.00075; tin, 0.0001; antimony, 0.0028.

The slime from the anodes is agitated in tanks with hot water, filtered and dried, and next melted in a water-jacketed reverberatory furnace lined with magnesite brick. The impurities are oxidized off, and doré metal running 960 to 975 parts gold and silver is obtained. The doré metal is parted with sulphuric acid. The silver sulphate is run into steam-heated tanks, where the silver is precipitated as a slime by bars of metallic copper. The silver and gold are collected, melted and cast into bars. The gold is 995 fine and the silver 999, the latter in bars of about 80 lb. each. Fine gold and some silver are sold to the branch of the Royal Mint, Ottawa, Ontario, and the United States Assay Office at Seattle, Wash. Silver finds its chief market in the Orient; refined lead is sold in eastern Canada, the Orient and other places.

By-products are copper sulphate and antimony. The copper sulphate solution, obtained in the course of treating the slimes, is crystallized into copper sulphate, making a commercial product used by farmers to kill smut in seed wheat.

Copper Smelting.—There are five copper blast furnaces, as follows: Two of 42 by 300 in. at the tuyeres; one of 42 by 263 in., and two of 42 by 240 in. The height of the charge is 9 ft. above the tuyeres, and the furnaces are operated with air pressure of 36 oz. Two forehearths are used with each furnace, but only from 0.25 to 0.50 per cent of the total matte is caught in the second settler. Most copper ores are smelted raw to a low grade first matte. The slag from the low-grade furnaces runs: Silica, 43 to 46 per cent; FeO, 14 to 15 per cent; lime, 20 to 22 per cent; copper, 0.1 per cent, and a very little gold and silver. The first matte, containing about 15 per cent copper, 27 per cent sulphur, and 56 per cent iron, is granulated, roasted in O'Hara furnaces to 10 to 12 per cent sulphur, blown in Huntington-Heberlein converters to a product which runs 3 to 4 per cent sulphur, and then smelted in a high-grade furnace, with suitable ores, to a matte running about 40 per cent copper, which last product is shipped to Tacoma, Wash., for bessermerizing. The slag from the high-grade furnace analyzes: Silica, 40 to 42 per cent; FeO, 26 to 28 per cent; lime, 16 to 18 per cent; alumina, 10 to 12 per cent, and copper, 0.35 per cent, with gold 0.02 oz. per ton. Some of this slag is used in the lead furnaces and part of its gold content recovered there.

General.—The ore supply comes chiefly from East Kootenay, Slocan, Nelson, Rossland and Boundary districts, and some from the neighboring State of Washington. Coke, of which about 225 tons is used daily, comes from Hosmer, Crow's Nest Pass, 249 miles distant. Limestone, chiefly for flux, is obtained from the company's quarry at Fife, 68 miles away. Requirements are about 200 tons a day, and cost is $1 per ton in the charge bins. Electric power is transmitted from Bonnington Falls, 33 miles distant, at 20,000 volts. At the smeltery it is stepped down to 550 volts. The total power consumption is about 3000 hp. The works are advantageously situated, with railway connections to all important mining camps on the Kootenay and Boundary districts, and with American roads to the southward. Mr. W. H. Aldridge is managing director of the Consolidated Mining and Smelting Company, of Canada (Ltd.), and has his headquarters at Trail.

New York Section American Electrochemical Society.

A meeting of the New York Section of the American Electrochemical Society was held at the Chemists' Club in New York City on Dec. 16, 1910. The chairman of the section, Mr. E. A. Sperry, presided. The meeting was very well attended.

The first part of the meeting was devoted to a general discussion of the fixation of atmospheric nitrogen, introduced by a brief paper by Dr. E. F. Roeber, who reviewed in succession the principles first of those processes in which air is treated, as it is, by means of electric spark or arc discharges (Bradley & Lovejoy, Birkeland & Eyde. Pauling, Badische Company) and then those processes in which the nitrogen is first separated from the oxygen in the air and afterwards caused to enter into combination with something else (manufacture of calcium cyanamide and synthesis of ammonia). As to the comparative power consumption in the manufacture of nitrates or nitric acid by arc discharges and in the manufacture of calcium cyanamide the following figures were given as representing best commercial practice: One kw-hour binds 17 grams of nitrogen in form of HNO_3 and 70 grams of nitrogen in form of calcium cyanamide. (In the latter case the power consumption in making calcium carbide is included.)

In the quite extended discussion which followed Messrs. Baekeland, Beckman, Loeb, Richards, Sperry, Tucker, Weedon, Whitney and others participated.

Mr. J. C. Clancy then presented a paper on his new process on the substitution of calcium cyanamide and electrolysis for cyanide in gold metallurgy. The paper is printed in full elsewhere in this issue.

Separation of Oxygen by Cold.*

By James Swinburne.

This industry has arisen, almost by accident, out of that of the liquefaction of gases. It had long been the ambition of scientific men to liquefy first one gas and then another; the liquefaction of such gases as oxygen, hydrogen and nitrogen being a sort of North Pole to be reached. It is doubtful whether reaching the North Pole is of any value to anybody but newspaper men, whereas the production of very low temperatures was bound to increase our knowledge very materially. Like most researches that appear at first to have purely increase of knowledge value, low-temperature work has proved to be, in addition, of direct commercial importance.

Though the separation of gases by cold has arisen out of the liquefaction of air, the two problems are not quite the same. Leaving out of account the details of the machine employed for the moment, we may consider the apparatus as a closed vessel, insulated to prevent heat getting in through the walls and having three pipes. At one pipe we may assume that air at the temperature of the room, but at a higher pressure, enters the machine; at a second and third, oxygen and nitrogen come out at the pressure and temperature of the room.

Air under high pressure has less internal energy than the same mass of air at the same temperature and lower pressure, so that if air is continually going into the apparatus at the temperature of the room and at high pressure, and its constituents are coming out at the same rate at the temperature and pressure of the room, more energy is coming out of the apparatus than is going in.

This is impossible as a permanent regime. It cannot be put right by less air coming out than goes in, because that would mean an accumulation of air. A balance can be obtained by taking out most of the air at the pressure and temperature of the room, with a high internal energy per kilogram, while a little of the air comes out at a low temperature, with less internal energy per kilogram. This portion may be quite small and may be liquid.

But the air is supplied under pressure and comes out at the pressure of the atmosphere, so it must be expanded inside the apparatus, and as there is no way of getting the energy of expansion out of the apparatus as described, it must be converted into heat, provided it is expanded irreversibly. This means loss of power, which is a small point, and production of heat just where it is not wanted, which is a serious matter.

If, therefore, in addition to the three pipes, there is communication by a shaft, and the apparatus may be arranged so that the expansion of the air works some sort of engine which turns the shaft so that work is given out, the yield of the liquid air must be larger. The ideal liquid-air machine would, therefore, take in compressed air and give out air at the same temperature, but at the pressure of the atmosphere; and a stream of liquid air, and as much power as possible, would be given mechanically through a shaft or other mechanical transmitter. It will be explained presently that the Linde separation process is nearly reversible, capable of being but little improved by giving out energy.

It is generally assumed that the problem of the separation of gases is substantially the same. This is not the case, however, and the overlooking of the difference has led inventors and designers of such apparatus into a series of absurd mistakes.

Take the ideal apparatus for separating gases; we have a pipe leading in gas under pressure at the temperature of the room, and two pipes coming out; one carrying oxygen more or less pure, and the other nitrogen more or less pure, at the temperature and pressure of the room. Now the internal energy of the gases coming out is greater than that of the compressed gas going in, and if the energy were alone to be

*A Faraday Society paper, presented at the December meeting in London.

considered, once the apparatus was started and got into running order as regards temperatures, air might go in at the pressure and temperature of the atmosphere, and come out as oxygen and nitrogen at the temperature and pressure of the atmosphere, the process being continuous and costing nothing appreciable. But as the apparatus cannot be made quite heat-tight in practice, the air might have to go in under slight compression, so as to have slightly less internal energy than the gases coming out, the difference being just enough to compensate for the leakage of heat through the casing of the apparatus. This is an ideal apparatus which several inventors have designed; but it is not possible to make such an arrangement work.

The energy is not the only quantity to be considered. Let us return to the simple case of an apparatus which is perfectly heat-insulated, fed with air at the pressure and temperature of the room, giving out oxygen and nitrogen separately at the same temperature and pressure. The entropy of the separated gaess is less than the entropy of the air going in, so if the process is continuous there must be a continual disappearance of entropy inside the apparatus, which is not thinkable. Air at a higher pressure, but at the temperature of the room, has less entropy, so it may be supposed we can get things right by compressing air until it has the same entropy as a compressed mixture, as the issuing gases have as they come out. We would then have an entropy balance.

This will not suffice in the ordinary course, however, because the compressed air must be expanded, and if that is done irreversibly inside the apparatus, more entropy is produced there, and to make a proper entropy balance either the expansion must be done reversibly or air must be so compressed that its entropy is reduced below the sum of that developed by any expansion and that of the separate gases. Imagine this done; have we a thinkable machine? No, because now our energy balance is wrong; we have air going in with less internal energy than the gases coming out.

It will be found on further consideration that the energy balance is got right by using imperfect heat insulation, heat leaking in to make up the extra energy of the issuing gases over that of the entering air; the entropy of the compressed air plus the entropy due to the heat leaking in, plus any entropy due to irreversible expansion or conduction of heat in the interchanger, being equal to the entropy of the gases coming out. If desirable the case may insulate perfectly, and some of the air or gases may be delivered continuously in the liquid state. The point to notice is that as the oxygen and nitrogen coming out have less entropy than if they were mixed at the same temperature and pressure, the apparatus has to be fed with compressed air, and energy is therefore needed to separate the gases. This point has been so often overlooked that it is necessary to dwell upon it. As far as I know, I was the first to publish it.

It has sometimes been supposed that because the gases are separated at a very low temperature the data determining the energy necessary to separate them do not depend on the difference of entropy of the separate gases and the mixture at the temperature of the room, but at the temperature of separation. This is erroneous and may give rise to further mistakes. That it is erroneous is clear from the illustration. Suppose the difference of entropy of cold gases separate and cold gases mixed is less than the corresponding difference at the temperature of the air; even if the entropy of the entering air is reduced enough to compensate for this only, and not for the entropy of the issuing gases, the entropy balance of the machine as a whole is wrong.

Heat leaking in is difficult to value, as there is an irreversible change as the heat is conducted to the cold gases, which means increase of entropy; and there is increase on entropy, due to irreversible expansion. For simplicity take a heat-tight case, and the energy balance kept right by drawing out a small stream of liquid air. Then, if there is no irreversible process inside the apparatus, there is no growth of entropy in it; and the entropy of the gases coming out, plus the entropy of the liquid coming out, must be equal to the entropy of the air going in. In other words, the entropy balance depends on the entropy of the issuing and entering streams, and not on the entropy of the gases at some inside point where they are at very low temperatures, though the separation actually takes place there.

To make the matter clearer it may be better to put the entropy question more fully. The change of entropy of a gas may be split up into entropy of change of volume, which varies as the logarithm of the volume or of the free path of the particles, and entropy of change of temperature, which varies as the logarithm of the speed of the particles. We may call these two the volume and the speed entropies ϕ_v and ϕ_s. When two gases mix, the volume of each increases in a way—that is to say, it pervades a larger space, and the molecular free path of each particle is increased if you take into account the collisions with other particles of the same gas only. The mixture of two mutually inert gases thus involves increase of entropy, which I will call entropy of mixture ϕ_m. As this depends solely on the logarithms of the free paths, it depends, in the case of two gases at the same temperature and pressure, only on the relative volumes of the gases before they are mixed.

There is still another point to be considered. If a perfect gas is, say, expanded at constant temperature, its internal energy remains the same; but such gases as oxygen and nitrogen have a slightly larger internal energy at the larger volume, as discovered by Thomson and Joule, and this involves an increase of entropy in excess of that due to the increase of mean path; this may be called the Kelvin entropy or the Joule entropy; preferably the Kelvin entropy, as the term joule, being used to denote the unit of heat, might give rise to confusion. We may call this ϕ_x, if a symbol is ever wanted.

We may imagine a closed apparatus with entry pipe for air and exits for nitrogen and oxygen. We have then air going in at one pipe and oxygen and nitrogen coming out at two others, and heat leaking in from the outside. The conditions for permanent regime are that the internal energy of the O and N coming out must be equal to the sum of the internal energy of the air entering and the heat leaking in. If this were all, of course, we would arrange no heat leakage, and let the air come in and the oxygen and nitrogen come out at the pressure and temperature of the air. But we must get the entropy balance right, too—that is to say, that the entropy of the oxygen and nitrogen coming out must be at least equal to the entropy of the air going in, and if heat is leaking in, or if there is any other irreversible process such as free expansion going on inside, or irreversible conduction in the interchanger, the entropy of the gases coming out must be greater.

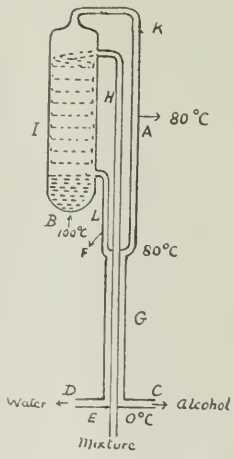

FIG. 1.—IDEAL DIAGRAM OF ALCOHOL STILL.

So many people are unfamiliar with this way of discussing such questions, that it may be better to deal with the matter also along other lines.

As the separation of the oxygen and nitrogen is carried out in a rectifier, we may discuss the action of the rectifier, taking an alcohol rectifier first as being more familiar, and not involving out-of-the-way temperatures; though, as a matter of fact, it

is in a way very much more troublesome to discuss fully.

Fig. 1 shows an ideal, or paper, diagram of an alcohol still reduced to a simple form. A mixture of 50 per cent alcohol and water at 0° C. comes in at *E* and flows through a heat interchanger *G* until it reaches 80°, the boiling-point of alcohol. It then passes up the pipe *H* to the top of the rectifier *I*. The pipe zigzags, or winds, down to a point where the temperature is 82°, the boiling-point of the mixture. The liquid is then discharged into the rectifier, which may be full of trays, and trickles down to the bottom and out by the pipe *F*, and thus to the interchanger *G*. The alcohol vapor comes off by the pipe *K*, and is nearly all condensed at *L* and passed to the interchanger. The water and alcohol come out at 0° C. at *D* and *C*.

The rectifier must have heat supplied at *B* to boil off enough steam to rise against the trickling fluid, and, assuming all the parts of the apparatus heat tight except where arrows are put, we have coming out of the system heat at *A*, where the alcohol vapor is condensed, and at *F*, where the water coming out at 100° is cooled to 80° before entering the interchanger. As alcohol and water heat on mixing, the output from *D* and *C* have higher internal energy than the ingo at *E*, but except for that, all the heat that goes into the interchanger in the pure water and pure alcohol goes into the mixture and goes back into *I* by the pipe *H*.

Suppose the still deals with 1 kg of mixture per minute, 500 grams of water come down the pipe *L*, so the heat given out in cooling the water to 80° is obtainable. The heat of mixture of water and alcohol gives the extra internal energy of the output. If the heat capacity of the mixture were the same as that of the alcohol and water separately, the water and liquid alcohol entering the interchanger would just heat the mixture to 80° as they cooled to 0°, but, as might be expected from their mixture evolving heat, the heat capacity, in which is really involved the disgregation energy, is higher, so either the water must be admitted above 80°, or the alcohol must not be all condensed before entering.

As we know the heat of vaporization of alcohol, we get the heat we could get out at *A*. Knowing the quantity of heating coming out, we know the heat put in at *B*. It might be thought that this would be anything the workman liked; this is not so, for if we put in too much, water would come out with the alcohol, and if too little, alcohol would come out with the water.

The broad result is that heat has to go in at 100° and come out almost wholly at 80°, the heat being roughly the heat of vaporization of the most volatile liquid.

To see what happens when we do the separation by mechanical power, we can imagine an ideal heat engine worked so as to deliver heat at 100° and to take in heat at 80°, or other temperature. If it does not get in enough at 80° it must make up the difference by direct degradation of work into heat or better by taking in heat at 0°: if it gets in more than is wanted at 80° it can deliver at 0°. If the process merely involved taking in the heat of evaporation of 500 grams of alcohol at 100°, and giving it out at 80°, calling the heat *H*, the engine would have to take in $H\,353/373$ at 80°, or $353^\circ A$, and by doing $20\,H/373$ of mechanical work on the medium, *H* would be put into the rectifier at 100° C. The $20\,H/373$ left over could be let down to 0, doing $(353-273)/353 \times 20\,H/373 = 0.0122\,H$.

Thus, if the heat of vaporization were 100, the first engine would do 5.36, and the other engine would recover 1.22, so that the total mechanical work to be done would be 4.15 of the heat of vaporization of the alcohol, or 18,000 joules per minute, or 300 watts. Instead of adopting the rough statement that we have to supply the heat of vaporization of the alcohol, we may work the example out approximately.

The alcohol vapor at *K* carries over 436,000 joules, as compared with its liquid state at the same temperature. This is not all given out at *A*, because a little is wanted for the interchanger *G*. This has mixture coming in, being heated from 0° C. to 80° C., and 500 grams of water and 500 of liquid alcohol, both at 80° C., will not heat it, as the specific heat of the mixture is higher than the mean of components. This is to be expected, because equal parts of alcohol and water give out 29,000 joules per kilogram on being mixed, so that the disgregation energy of the mixture must be greater, and the heat taken in between 0° and 80° is largely disgregation energy. The extra heat needed is about 34,800 joules. This is returned to the rectifier by the hot mixture, so we get out $436{,}000 - 34{,}800 = 401{,}200$ joules at *A*. We also get 42,000 by the water coming out at *L* at 100° C. and being cooled to 80° C. before entering the exchanger *G*. We thus have to supply 443,200 joules at 100° C. at *B*.

The heat of the water in *L* may be utilized. The difference of entropy of 500 grams between 100° C. and 80° C. is 116.5 in practical units, so the heat put in at 100° C. is $373 \times 116.5 = 43{,}500$ joules; of which 42,000 are from the heat of the water and 1500 from work done. This leaves 399,700 to be supplied by an ideal engine working between *A* at 80° C. and *B* at 100° C. Such an engine will take in 378,280 as heat and put in 21,420 as work. But there are 401,200 joules coming out at *A*, so there is a balance of 22,920. An engine working between *A* at 80° C. and 0° C. will utilize 7820 joules out of this, rejecting 15,100 at 0° C. We thus have $1500 \times 21{,}420 - 7820 = 15{,}100$ joules to put in mechanically, or 252 watts.

One of the chief difficulties in this sort of calculation is the uncertainty as to the numerical data. This case is also complicated by the internal energy of the alcohol and water separate being greater than that of the mixture.

To get rid of the effect of the internal energy of the separated liquids in the last case the arrangement shown in Fig. 2 may be discussed. Here mixed alcohol and water as vapor comes in at *O*. It goes to an interchanger *G*, where it is cooled down to 80° C., being thus condensed, and it flows by the pipe *H* to the rectifier, which is as before, except that it gives out steam at *L* instead of water. The alcohol coming out at *K* is taken away as vapor. The mixed vapors condensing in *G* will give off much more heat than is necessary to raise the alcohol vapor from 80° C. to 100° C., so heat will be given off along *G*.

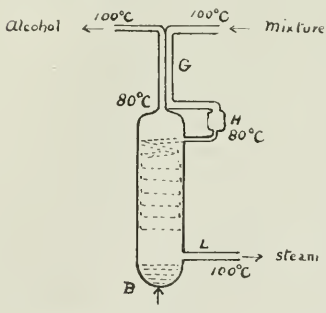

FIG. 2.—IDEAL DIAGRAM OF ALCOHOL STILL.

For want of data as to heat of vaporization of mixtures of alcohol and water I have taken the heat that would be given out by 500 grams of each and deducted the extra internal energy, 29,000. This gives 1,541,550 joules given out along *G*, after deducting 950 for heating the issuing alcohol vapor. We must therefore supply this at 100° C. at *B*. As this large amount of heat is given off all along *G* it is difficult to know its value. To get it we would need a table giving the boiling-points of the range of mixtures of alcohol and water, and the composition of the vapor coming off, and to be quite accurate we should have a column of heats of vaporization.

For simplicity we shall not be far wrong if we take the heat as being given out at 94° C. This needs 25,200 joules given mechanically and 1,541,550 taken in as heat from *G*. Another engine will have to put in 1120 joules as work, taking in heat at 0°, and giving out 3805 at 100° C. We have thus 26,320 joules per minute, or 438 watts. This figure is most likely rather inaccurate, as the data are doubtful. The example is complicated by the heat of combination of alcohol and water

too. Broadly speaking, what has to be done is to put in heat at the boiling point of the less volatile liquid, taking out heat at a temperature between it and the boiling point of the more volatile liquid. This involves either supplying the system with heat at one temperature and recovering some of it at a lower temperature, or supplying energy mechanically.

The case of separating oxygen and nitrogen is simpler, as there is no development of heat on mixing the gases. It is exceedingly unlikely that there is any liberation of heat on mixing the liquids, but in the ideal case discussed next, if there were such heat it would not affect the reasoning, as its influence would cancel out.

Suppose for simplicity we are operating, say in Dis, at a temperature of 90° A, and we have a system as shown with a pipe leading in air at 90° A, and two pipes delivering oxygen and nitrogen at 90° A.

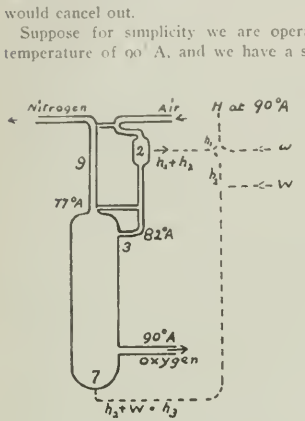

FIG. 3.—DIAGRAM OF SEPARATION OF OXYGEN AND NITROGEN.

Air comes in at *A* and the pipe branches, most of the air going to the cooler 2, where it is reduced to say 82° A, a good deal being liquefied. The partly condensed air then goes into the rectifier at 3. Nitrogen comes out as gas at 4, at 77° A, and is raised to 90° A by a portion of the entering air going down the interchanger 9. Heat has to be supplied to the rectifier at 7 to boil off the oxygen, and heat is given out at 2. Thermodynamic engines take in heat from 2, and deliver some of it at 7 and the rest to the outside. We have to find the power necessary to drive the engine to separate a kilogram of air per minute, or the energy to separate a kilogram of air. A kilogram of separated air in an ideal engine with semipermeable pistons could do 13,180 joules in allowing the gases to mix at 90° A; 13,180 joules must therefore be done in separating them. As the internal energy of the separated gases is the same as that of the mixture, this work must be wholly degraded into heat. The system that takes in air and gives out oxygen and nitrogen at the same temperature and pressure must therefore give out heat, taking in work; and the heat for each kilogram of air is thus 13,180 joules.

We do not really need to trouble ourselves as to what goes on inside our closed system. We do not need to know anything about the heat of vaporization, boiling-points, or heat of mixture; all we have to know is the work that can be imagined as obtainable by a suitable arrangement for mixing the gases, and we then know the mechanical energy ideally necessary to separate them, assuming the system to have no irreversible process going on inside.

But it is worth while to look inside.

Call the heat given out at 90° A, H, and 90° A, h_1 and the temperature of 2, Θ_2. Let the heat H be given out by a thermodynamic engine taking in work w_1 and heat h_1 from 2 at Θ_2. Then H is made up of $h_1 \times w_1$ and $h_1 = H\Theta_2/\Theta_1$, and $w = H(\Theta_1 - \Theta_2)/\Theta_1$. The rest of the heat given out at 2 is taken in by a second thermodynamic engine at Θ_3 and given to the rectifier at Θ_3. Call this heat h_2; then the heat given to the rectifier is $h_2\Theta_3/\Theta_2$, and the work done by the second engine, W', is $h_2(\Theta_1 - \Theta_2)/\Theta_2$. But $H = W' + w$ and $h_1 + h_2 = h_3$, as the rectifier takes in and gives out the same amount of heat. From these equations we get $w = H(\Theta_1 - \Theta_2)/\Theta_1$, and $W' = H\Theta_2/\Theta_1$.

Suppose that instead of working at 90° we lead the inlet and two outlet pipes through an interchanger, and work at 273° A, or say 270° A, to make a round number. The work W is not altered, as the conditions of that engine are not interfered with; but the first engine that takes in heat h_1 at Θ_1 now has to give it out at 270° A instead of 90° A. It is simpler arithmetically to imagine a third engine taking in the heat H at 90° A and giving it out at 270° A. This engine does 2 H.

It is hardly necessary to repeat that the mathematical power needs vary as the absolute temperature, as the work ideally obtainable on mixing the oxygen and nitrogen at atmospheric pressure varies as the absolute temperature.

Take now the case of dry air to be separated completely into oxygen and nitrogen. The work ideally obtainable from a kilogram of the gases mixing is 40,000 joules, so to separate a kilogram a minute needs 666 watts in a perfect machine where all the processes are reversible. This would come out fairly expensive on a very large scale.

In this case it has been assumed that the air is to be separated completely into pure oxygen and pure nitrogen. It does not follow that it pays to separate in this way unless both gases are wanted pure in the proportion of four to one. If oxygen alone is wanted, it does not pay to make pure nitrogen for that means abstracting oxygen from a mixture with very little oxygen, while air is obtainable at no cost. To separate a kilogram of air at 0° C. into pure oxygen and nitrogen needs 40,000 joules to be converted into heat*; to take out half the oxygen needs only 17,000 joules, but as this gives only 115 grams of oxygen, 34,000 joules are needed to give the same weight of oxygen. If the oxygen is wanted for enriching air for combustion purposes, it is cheaper to extract it impure. But for pure oxygen the most economical proportion to extract—on paper—is about 50 per cent. In practice it may be better to take out a little more, as the less gas you pass through the apparatus the smaller it may be.

It might be said that given a kilogram of air cooled down, a machine that extracts all the oxygen does its work better and is cheaper; this is not the case, however, as air does not cost anything appreciable, even cooled.

We have therefore the problem of taking in heat at a temperature of about 82° A and giving it out at 90° A, and the temperature of the room, say, 290 A.

There are broadly two methods in use. It is not necessary to trace the history of the liquefaction of gases, and to follow out the ingenious intensification methods. They are all well described in the literature of the subject.†

The first method is Linde's. This takes advantage of the discovery by Kelvin and Joule that such gases as air have less internal energy when compressed. If you feed a heat insulated system with air under compression and let it expand before it comes out, the air coming out must at first take out more energy than the air going in carries with it; so the inside of the apparatus must get colder and colder. If the outcoming air comes through an interchanger, it will come out at the same temperature as the ingoing air, so the other end of the apparatus must get colder and colder until some of the air is liquefied. The balance will come when the internal energy of the gas and liquid air taken out is equal to that of the compressed air going in.

In practice the same machine does the liquefaction and separation; but it is simpler for our purpose to treat a machine in which air is compressed and allowed to expand as a thermodynamic engine, using the liquid air as a means of extracting heat at a low temperature. It was this invention of Linde's that made the production of liquid air, and his subsequent invention of combining the rectifier with it that made the commercial production of oxygen thermodynamically from air possible.

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*For simplicity, atmospheric pressure is taken as a megadyne per square centimeter; and heat is taken in joules, not calories; entropy, when necessary, can be taken in entrops, which, multiplied by the lowest available temperature, give the incurred waste in joules.

†The engineer will find G. Claude, "Air Liquid," and Ewing, "The Mechanical Production of Cold," interesting.

The other invention is due to Claude. It was, of course, obvious that cold could be produced by compressing the air at a high temperature and then expanding it so as to do external work at a low temperature, and the merit does not here lie in the idea. But to work out such a system is a very different matter. The practical difficulties in making an engine that would give out any reasonable power at a temperature of 80° A are very serious indeed.

So far I have discussed the subject on what are sometimes incorrectly called theoretical lines, that is to say, ideal lines, in which all sorts of assumptions are made so that the discussion should be easier. Thus I have discussed perfect heat insulation and completely reversible changes and perfect thermodynamic engines. The principles are the same on a large scale; but there is a leakage of heat, and all sorts of irreversible changes of heat, and a compression pump does not come up to the thermodynamic ideal in efficiency. In the industry as now carried on the question of efficiency is of very little importance. Oxygen is sold in small quantities in bottles; and the costs of compressing, handling, selling, delivering and so on, and the collection of the empties, are enormously large in proportion to the mere cost of making. Thus we have discussed oxygen, taking 34,000 joules for 230 grains, or about 2 kw-hours per cubic foot.

The great future for the separation of the gases is in the direction of making nitrogen for the manufacture of calcium cyanamide, and oxygen for use in blast furnaces, and other work where a high temperature may be worth the extra cost. It is, therefore, important to see how far the gases can be produced on a large scale at prices within a reasonable margin.

We may, therefore, sum up the rectifying part of the plant as an apparatus which gives up heat at a low temperature, say 82° A, and absorbs heat at, say, 90° A, the heat taken in at 90° A being equal to that given out at 82° A. A refrigerator must, therefore, be a thermodynamic engine capable of giving heat at 90° A, absorbing heat at 82° A and giving out heat at the temperature of the works, say, 290° A.

It is generally supposed that Linde depended on an irreversible expansion of a gas. This is not the case, and I would put the matter in another way.

If gas is compressed, cooled to, say, 290° A, and sent into an interchanger, out of which it comes by a return pipe at the pressure of the air, there must be cooling going on inside, as the air delivered has higher internal energy than the air going in. You have therefore an apparatus for producing cold—that is to say, for absorbing heat at a temperature below the room. The question is, what is the lowest temperature at which this apparatus will take in heat? The lowest temperature is that of the boiling-point of the liquefied gas at the pressure of the atmosphere. If the apparatus is working in such a way that it can take in heat H at temperature O_2 the increase of entropy is H/θ_2. But if instead of working in this way the heat is taken in at Θ_1, which is a higher temperature, the increase of entropy is H/θ_1, which is less, so the apparatus makes up the deficit of entropy by irreversible expansion.

On starting up a Linde machine it begins by abstracting heat from its own air and metalwork at the temperature of the air, so the entropy is made up by irreversible expansion. As the temperature at which the heat is absorbed falls, the irreversible expansion gets less and less, until finally the whole of the gas is turned into liquid. Then there is some production of heat by the liquid passing from a high to a low pressure through an obstruction, and also some slight evolution of gas to cool the released liquid from the boiling-point under the high to the boiling-point under the low pressure. The irreversible loss due to liquid is $pv/10$ in joules, where v is the volume of the liquid. If the compression is small, this is small, but if it is high, pv may be considerable. It would be quite easy to get this energy out by means of a little engine, as there is no expansion and no difficulty in lubrication, and it would be worth while on a large scale The liquid may also be cooled to a lower boiling-point while still under pressure, so that there is no irreversible evolution of gas on its being exhausted from the engine.

The refrigerator has to do the work of two thermodynamic engines, for it has to abstract a lot of heat at one temperature, give a large proportion back at a higher temperature, and give out heat at the temperature of the works. The ideal apparatus consists of a compressor with water-cooling delivering gas at the temperature of the works and a high pressure to an interchanger. The pressure is chosen so that the gas liquefies under it at 90° A, giving up its heat to the rectifier. The liquid at 90° A is then cooled in an interchanger to 82°, and it is then passed through a hydraulic engine, coming out at atmospheric pressure and temperature 82°. It is now vaporized, taking in heat at 82°, and goes out by the delivery side of the interchanger.

The process described is not quite reversible, because the specific heat under constant pressure of the gas coming down the interchanger is greater than that of the gas coming up under atmospheric pressure; there has therefore to be an abstraction of heat all along the interchanger to make up for this difference.

The idea of this apparatus is something like that of an ordinary carbonic acid machine worked under different conditions. If a carbonic acid machine were worked in the tropics and the cooling water were above the critical point of CO_2, the compressor would supply compressed gas only. Some sort of refrigerator would be necessary to liquefy the CO_2, and the liquid CO_2, would then be evaporated at a low temperature. We thus have three temperatures to consider—the temperature of the works at which heat due to compression is given out, the temperature at which the CO_2 is liquefied under pressure and the temperature at which the CO_2 is finally evaporated. In a refrigerating plant three temperatures would be a nuisance, so a fluid is chosen with a critical point above the temperature of the works. But for rectifying three temperatures are necessary—the temperature of the works, the temperature of the boiling-point of the least volatile liquid and the temperature of the liquefaction of the mixture. We can therefore use a fluid which is compressed at a temperature which may be much above the critical point of the liquid.

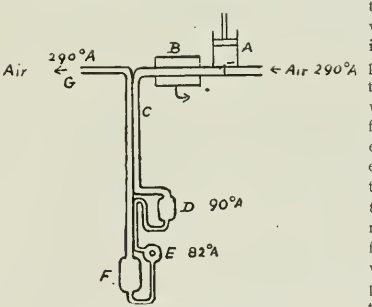


FIG. 4.—IDEAL LINDE REFRIGERATOR.

Fig. 4 shows an ideal Linde refrigerator. The compressor A delivers a fluid, say air under pressure, to the cooler B. Heat comes out there at the temperature of the works. The air goes on down through the interchanger C until it is cooled to 90° A. Then it is cooled in D, which is really a coil of pipe immersed in the oxygen to be evaporated at the bottom of the rectifier. From D the liquid, still under pressure, goes down the interchanger until it is cooled to 82° A, and it then works a little engine, E, and comes out into F. F is really a coil of pipe in the liquefier of the rectifier. The air is evaporated in F, and passes up the interchanger and escapes at G.

There are some points to be noticed. As already pointed out, the interchanger must take in heat by leakage, which is an irreversible process; and the heat taken in at F will be greater than that given out at D, as it takes more heat to vaporize a liquid at a low pressure and temperature.

The processes in this refrigerator are, however, substantially reversible.

If schemes for dealing with liquefied gases are examined, it will often be found that a liquid is being condensed or evaporated by a gas in the interchanger. That is always a piece of bad design involving irreversible change. It may be inevitable, of course, but it should be recognized as an evil which must be carefully watched.*

Linde points out that the mechanical work necessary may be reduced by working between two high pressures. But there is a limit to this, as if both pressures are very high the lower temperature limit of the plant is reduced; so the limit is reached by the lower pressure, which must be low enough for the fluid to evaporate at 82° A, or whatever temperature the heat has to be extracted from the rectifier. If we choose a fluid, say nitrogen, this gives the lowest pressure that can be used. Then the other pressure must be high enough for the liquid to condense at 90° A. Some latitude can be got by control of the pressures in the rectifying plant, but it is simpler for the present to take the rectifier as working at atmospheric pressure for the sake of ease of calculation.

The refrigerator, then, has to supply for each kilogram of air split into pure gases 135,000 joules at 90° A, absorbing 135,000 at 82° A. The liquefying of this fluid is in contact with a fluid that is being evaporated, so that that part of the process is ideally reversible. The re evaporation is in contact with the air which is being liquefied, so that part of the process is ideally reversible. The passage of the liquid from the high to the low pressure can very easily be made to give out work. It is not the same problem as dealing with expanding air, as the volume is small, the pressure constant and the liquid acts as lubricant. The process, so far, is essentially reversible. The specific heat of compressed gas is higher than that of the same gas at a lower pressure, so a little allowance must be made for that. It is thus possible to use a process which has no external work done by expansion of cooled gas in such a way as to get little loss by irreversible changes.

To work out an example, nitrogen can be taken as the fluid, as it is easier to discuss than a mixture. To get 135,000 joules, taking heat of vaporization of nitrogen 200 joules per gram, we need 675 grams of nitrogen, and we can liquefy this at 90° A under about four atmospheres, evaporating it again at 82° A at two atmospheres. Taking the density of liquid nitrogen as 0.8, the hydraulic engine gives out 170 joules, a mere trifle in this case, and the compression of the nitrogen from two to four atmospheres needs 37,000 joules, so we need under 37,000 joules. As the separation needs 40,000, this is clearly an erroneous figure, but the data are very uncertain. We must need more than 40,000.

To work on a large scale with atmospheric pressure in the separator and four and two in the refrigerator would be quite out of the question. The volumes of air would be enormous, and the interchangers would be very inefficient. Those who have had to work with gases that have to be heated or cooled will realize the extent of this trouble at once. To make the plant practical it will be necessary to work the rectifier under pressure. This raises the temperature of the rectifier; but that does not mean that we can separate the gases more economically; the cost of separation depends, as already stated, on the difference of entropy of the gases entering and leaving the plant, not on what goes on in the plant, provided there are no irreversible changes. Using pressure allows the refrigerator to be worked at higher pressures, so that the interchanger may become practical.

Assuming the ideal energy needed to separate a kilogram of air into its gases to be 40,000, the question is: What will it cost in practice? On a large scale it may be safe to say 100,000 joules. We would then get 230 grams for 100,000 joules; or a ton would need 120,000 kw-hours. Taking the cost of a kilowatt hour as low as 0.2 cent, this means $250 a ton.

It does not look as if oxygen at this order of figure could be used for blast furnace work. Taking a blast as using four and a half tons of air per ton of iron, to increase the oxygen content of 1 per cent of the oxygen content the cost comes out at $2.50 per ton of pig. This does not look as if it could be of use.

The principle of separating gases by cold can be applied to other gases. Thus hydrogen may be purified, so that water-gas can be used as source of balloon gas. Methane can be separated out by solution in water under pressure.

Chlorine can be separated from the impure gas from the anode chambers in electrolytic works. Such chambers are generally worked under a slight exhaust to avoid troubles due to leaks. Air thus leaks in, and the chlorine coming off is moist and impure. Chlorine can be condensed into water, and it might appear that condensing such gases as chlorine and methane into water would allow of their being separated with less expenditure of energy than the entropy consideration demands. This is not so. It takes greater pressure to get the gas into the liquid out of a mixture than the solution exerts in an atmosphere of the pure gas.

Chlorine, by the way, is easier to handle than is generally supposed. It can be pumped in iron pumps flooded with sulphuric acid. It can also be dried and then compressed in iron pumps lubricated with sperm oil saturated with chlorine. It is not necessary to dehydrate it absolutely for compression up to, say 50 lb.

There may be other gases that can be separated or purified by liquefaction and rectification; it is difficult to think of any case where the principle cannot be applied.

Progress in Ore Dressing in United States and Mexico During 1910.

By H. A. Guess.

Water Concentration.

The record of progress during 1910 compares favorably with that of any recent year. If it were possible to make a general observation outlining the direction the various betterments have taken it could be said that progress and improvement had been made chiefly in the way of detailed steps looking toward the recovery of the mineral, without unnecessary sliming—either by stage grinding with mills, with recovery of the released mineral before the second grinding, or by carrying roll recrushing to such a degree of fineness that in certain cases it largely replaced the work of grinding mills. Coupled with this has been the attention devoted to the better classification of table and vanner sizes and especially to the more complete removal of all slime from feed going to stratifying tables. Equally important, too, have been the little changes and betterments—bulking large in the aggregate—which have been going on at the different plants over the country, all tending to economy in operation, the handling of a larger tonnage, the making of a cleaner product or a higher recovery.

Following the ore in water concentrating processes by steps it may be said that nothing noteworthy in the way of new machinery or arrangement has developed during the year in coarse or dry crushing preparatory to concentration.

In jigging—where jigging is used—the Hancock jig, with its large capacity, has continued to find favor, particularly in the lead district of Southeast Missouri, and all mills built there within the past two years use that type of jig, some 9000 tons per day being at present handled there by that system.

In screening other than for sizes finer than 1 mm, trommels continue to be the more usual equipment, but of late the impact screen, which after its earlier trials of some ten years ago has fallen rather into disrepute, has been improved mechanically so that its use is greatly increased, some of the largest of the

* There is another factor very common with regard to gas issuing from a nozzle. Suppose it runs out into the atmosphere, it is thought that as it has to expand against the pressure of the atmosphere, doing work in joules equal to a tenth of its increase of volume, it must be cooled to a corresponding degree. This is a mistake nearly everyone makes until he thinks of it under a second time. As such an error would give rise to many misapplications in connection with the liquefaction of gas, it is not impertinent to call attention to it.

concentrators on porphyry copper ores now adopting impact screens entirely.

For fine crushing the tendency has been to carry the work as far as possible with rolls, and where grinding mills are used it has been in many cases found profitable to conduct the operation in two stages with classification and table treatment in between, the coarser sizes being returned from the roughing tables to the second mill. Certain of the porphyry concentrating plants have adopted and within the past year extended this system with improvement in recovery, as well as increase in tonnage per mill section and per horse-power consumed. During the year extended working tests were carried out under the above-mentioned system in comparison with direct crushing with Nissen stamps and the advantages of the stage-grinding system, in such comparison, were clearly apparent in tonnage output per square foot of mill floor space, in power consumed and in percentage recovery. Mills of the Chilean type appear to find most favor, especially among the larger plants, although altogether there are doubtless more Huntington than Chilean type mills in use throughout the country. Occasionally tube mills are seen in concentration plants for fine grinding of certain middling products, and their use within the past year has been on the increase, particularly the more recent Hardinge conical type. The tube mill has heretofore been regarded chiefly as a slimer, due to its extended use for that purpose in cyaniding, but by suitable adjustments in rate of feed and ratio of water it is showing encouraging results as a fine granulator in comparison with the more usual grinding mills, the percentage of slime formed being not excessive nor the horse-power and supplies unduly high.

The classification of table and vanner sizes has received much attention during the past year, especially in regard to the more complete removal of all slime from feed going to stratifying tables. The removal of slime is not difficult in the coarser sizes of table feed, but in the finest sizes it has been impossible by any continuous discharge type of hydraulic classifier to make the separation from slime clean cut. The usual procedure in such cases has, therefore, been to recover as much of this slime as possible by collecting the table back-water for subsequent vanner treatment and to let the rest go to waste with the table tailing. In cyanide practice mechanical classifiers are largely used for the separation of fine sand from slime and the same idea is now beginning to be applied at some of the larger water concentrating plants. At the largest plant in Southeast Missouri mechanical classifiers similar to the Esperanza drag classifier of cyanide practice, only a little longer and arranged with spray pipes for washing, have been installed—one in each 600-ton mill section—taking the undersize of 1-mm trommels and separating therefrom all slime as overflow product and all sand as a washed drag product, the slime after thickening going to vanners and the sand going to hydraulic classifiers for division into various sizes of table feed. Another system lately adopted on a large scale by certain of the porphyry copper concentrators, and looking to the same elimination of slime from all table sizes, is the use of intermittent discharge hindered settling classifiers of the Janney type, which deliver the classified products in thickened form. The Richards vortex and hindered settling classifier and also the more recent pulsating classifier continue to have a wide use.

In table concentration each year sees new models—all practically of the original Wilfley type—put upon the market. Joplin was about the last of the older established districts to awaken to the losses being made through inattention to the treatment of sizes too fine to jig and in this and its adjoining districts many tables have of late been installed as adjuncts to the older jigging plants. A large proportion of these are of local manufacture, some half-dozen different styles being on the local market there, all of them of very good construction. In the other and better known districts of the United States and Mexico the Wilfley, Overstrom, Card, Deister and James continue to be the types more frequently seen and in about the order mentioned

The Deister No. 3 table competes fairly successfully with the vanner on slime, its merits being more or less confirmed in that one of the large porphyry copper concentrators now nearing completion is adopting the Deister No. 3 for its slime concentration. In general, however, for slime concentration vanners continue to be the standard machine, although there is recently some revival in the use of buddles on slime, the larger installations being of steel frame with concrete top; also some interest has been aroused in traveling canvas-belt machines, such as the Craven slimer, which in the matter of recovery does good work. It is usually considered that in vanners the Frue type does a little better where there is considerable mineral, the Johnson suspended type being more in favor for second treatment or where the mineral is in small amount. The Isbell vanner supported from a steel girder foundation, instead of the more usual four-post construction, has been introduced this year and may in practice present some features of improvement. Retreatment of vanner tailings over a second series of vanners is now becoming accepted practice unless the ore is very lean, the first tailings being usually thickened before going to retreatment.

The disposal of tailings is, with the advent of larger mills, becoming something of a problem, especially in a flat country where surplus water is not available. In Missouri, where such conditions exist, the more recent plants use dewatering devices feeding the comparatively dry tailings to an inclined conveyor belt which acts as a stacker, the motor drive being at the upper end. Extensions to the belt are made as occasion demands, the upward slope being maintained—usually about 16 ft. to the 100 ft. It may be noted that for such work the less troughing the belt receives the less tendency to crack and the longer its life, and this in less degree is also proving the experience in the handling of dry material with conveyor belts.

Cyaniding.

In cyaniding the practice of all-sliming treatment is on the increase and the tube mill remains the approved machine for such work, although at some of the larger plants comparative tests are still being run of tube mill versus pan, for fine grinding and sliming, and at the Homestake a number of Wheeler pans are in regular service. In connection with the tube mill as a slimer a comprehensive series of grinding tests at Pachuca—details of which were given in a paper by Mr. V. B. Therrod before El Instituto Mexicano de Minas y Metalurgia—led to the conclusion that the grinding efficiency increases with the percentage of solids in the feed up to 55 per cent to 60 per cent; that it also increases with the quantity of ore fed per minute to a point which varies with the dimensions of the mill, and that the horse-power efficiency appeared to be lower in the mills of larger diameter—5 ft. x 20 ft.—than in the smaller mills—4 ft. x 18 ft.

In the separation of sand from slime for purposes of return for regrinding or for leaching, the Dorr classifier continues to have a wide use, and more recently, to some extent, drag classifiers of the Esperanza type. For the same purpose the Caldecott diaphragm cone is also being adopted, and its use seems on the increase.

For the agitation of slime preparatory to filtration the Pachuca tank or its equivalent continues to be the accepted system for new installations, and at several plants these tanks are of late being connected up in series, the first overflowing to the second, and so on, thus giving continuous agitation in transit without loss in head; with simplicity in operation and equal percentage extraction. Considerable interest has been aroused by the Just process, which uses agitation by air through porous silica brick placed as a floor in flat-bottomed tanks, the air being introduced beneath and coming up evenly through the brick. The same material is also being used to some extent as a filter in leaching tanks.

Much attention during the year has been devoted to the development and perfecting of filtering processes for slime. Various patented filters, both vacuum and pressure, have been

added to the already large number of such devices on the market, so that, with intelligent selection of the machine best adapted for each particular type of slime, steady progress seems assured in the matter of filtration. In this connection, too, the use of slime thickeners of the Dorr type is relieving the work of filtration to some extent. Of the various filtering devices now on the market the best known are, perhaps, the Butters, Merrill, Burt, Grothe-Carter, Ridgeway, Kelly, Moore and Blaisdell.

In precipitation the economy of the zinc dust over the shavings method in labor and supplies is such that its use is being more widely adopted each year, especially at the larger plants and new installations.

Altogether, it may be said that each year sees great increase in the variety of ores successfully treated by cyaniding, and especially the silver and silver-gold ores of Mexico.

Miscellaneous Processes.

The more usual ore-dressing processes are the foregoing, namely, water concentration and cyaniding, either of which may sometimes include amalgamation.

Of the remaining processes it may be said that chlorination continues to have a limited application in the treatment of certain concentrates and also continues to compete with cyaniding in the treatment of certain crude ores, notably in Colorado; that leaching of certain copper ores is still occasionally practised, using either dilute acid or acid ferric salts; that a few dry concentrators of the S. & S., the Behrend and other types are being installed now and then, where no water is available, and that recently the Quinner dry-placer machine is being used to some extent, as, for example, in the Altar desert district of Mexico.

In general, however, it may be said that the special processes, for the most part, center around the recovery of zinc blende from complex materials, usually middling products from wet concentrating mills, containing in addition iron and perhaps copper sulphides, carrying usually some gold and silver, frequently also with small percentages of lead remaining from the wet concentration, some of the gangue material being often of heavy specific gravity—garnet, siderite, barite, fluorite and the like.

Considerable progress in the separation of such materials has been made during the past year, the processes grouping themselves, in the main, under three heads—magnetic, electrostatic and flotation.

In point of tonnage handled the magnetic processes of separation continue to lead, either with or without preliminary roasting; in the former case the iron and copper being the magnetic products, and in the latter usually the zinc. The Wetherill, both belt and rotary, the Dings and the International are the more usual types in use.

The electrostatic process is now represented by the Huff separator, which after being perfected in the separation of zinc and iron sulphides in Wisconsin, has shown good results similarly in Utah.

In flotation the Elmore oil-vacuum process is now giving very fair results in one or two small installations in Mexico, the grade of zinc recovered being particularly good. Its application in this country, however, has so far proved limited.

Several of the various acid and oil-acid flotation processes as used at Broken Hill, including also the Horwood process, have been tried out in America, but so far nothing has developed beyond the experimental stage.

Without doubt the most interesting of all the flotation processes so far is that of the MacQuisten tube, in which the sulphides are mechanically floated upon water by passing through tubes some 8 in. in diameter by 6 ft. long, riffled internally by a rib of triple or quadruple pitch. In point of size 20-mesh material is about the maximum that can be readily floated, elimination of slime and a certain amount of classification being also required. A disadvantage is the small capacity per tube, about 3 tons per 24 hours, and the necessity of passing the material through a series of tubes—perhaps a maximum of six in all—in order to obtain a satisfactory extraction. The chief field would seem to be where water concentration fails, as in the separation of sulphides from gangue of about equal specific gravity—such as garnet, siderite, etc.—and very good results are now being obtained upon such material either as original ores or middling products.

Flat River, Mo.

A New Electric Furnace for Steel Melting and Refining.

By Jon. Härdén, E.E.

The electric steel furnace appears to be slowly but steadily gaining ground in this country (England).

A 1½-ton Kjellin induction furnace has recently been erected in Sheffield for crucible steel work, and has successfully completed its guarantee run. This furnace is especially adapted for producing the highest grade of tool steel and is regarded as a most effective adjunct to a crucible steel plant. It is, however, well known that a furnace of this type is not adaptable for any extensive refining of steel, owing to the refining surface of the bath being small.

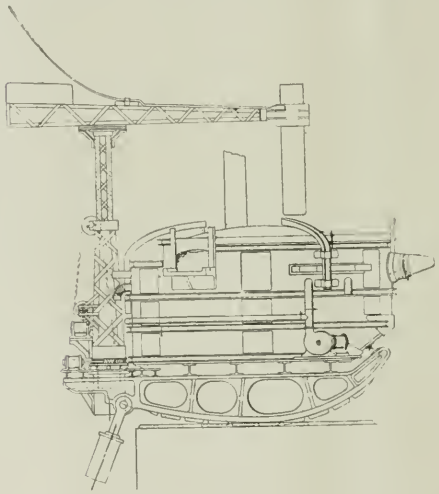


FIG. 1.—VERTICAL SECTION OF COMBINED ARC AND RESISTANCE FURNACE.

It has long been recognized by steel makers generally that although the induction furnace is an excellent tool in the hands of the crucible steel maker a furnace which would permit more extensive refining, and also the construction of large units, at the same time keeping the advantages embodied in the induction furnace system, is much to be desired.

It is because of the comparatively low power factor of larger furnaces of the plain induction type that this class of furnace has not been constructed in large units to any great extent.

In order to overcome this the Roechling-Rodenhauser furnace, which is partly a resistance furnace and partly an induction furnace was designed. A considerable number of these furnaces have been erected and are in operation abroad, giving very good results.

Many steel makers, however, are of the opinion that it would be an advantage if a furnace could allow superheating the slag for refining purposes, but at the same time enable the metallurgist to so regulate the heat of the charge that gases could be expelled without difficulty. At the same time the

perfect circulation of the bath should be maintained, so as to secure a perfectly uniform material. A new furnace designed on these lines, which the inventors call the "paragon" furnace, appears to possess many points in its favor.

As is shown in illustrations, Fig 1, 2 and 3, this furnace is a combination of an arc and a resistance furnace, which makes it very much like an open hearth furnace, and it may indeed be worked exactly in the same manner.

The slag blanket is kept at the desired temperature by means of arcs playing over the surface of the slag while the remainder of the heating power is transmitted to the bath by means of terminal plates, such as are used in the Roechling-Rodenhauser furnace.

The result of this combination is that the slag may be heated to any desired temperature while dephosphorization and desulphurization take place, but the heating effect of the arcs need not be increased to such an extent as to keep the whole charge superheated, which, of course, would tend to overheat the steel immediately beneath the arcs, but a carefully regulated amount of current can be at the same time sent through the resistance plates, thus also heating the charge from underneath.

When the dephosphorization and desulphurization are finished, it may be desirable to alter the gradient of heat in the furnace so that the lower portion of the bath is hotter than its surface, in order to expel the gases more readily. This result is easily obtained by diminishing the power transmitted through the arcs and increasing the power transmitted to the side plates. In fact, it is even possible to extinguish the arcs for a time and keep the bath liquid by means of the side plates, which for this purpose are provided with a cooling device of enforced air.

It is obvious that the metallurgical process is thus considerably facilitated, inasmuch as the metallurgist has the

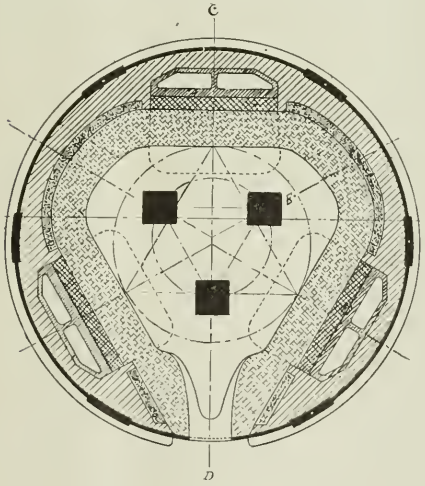


FIG. 2.—HORIZONTAL SECTION.

power of controlling the gradient of heat in the bath at will.

This combination has another advantage, viz, that the electrodes do not require to be of such large section for a given capacity of furnace as would be the case if the total amount of power were transmitted by means of the arcs only, owing to the fact that normally about 50 per cent of the total power is transmitted through the arcs, while the other 50 per cent is transmitted through the side plates. The advantages, therefore, are threefold.

Firstly, the size of electrodes for a given furnace capacity is only about half that of an arc furnace, and consequently with a given electrode section available the melting capacity of the furnace can be nearly doubled.

Secondly, owing to the destructive effect of large arcs immediately under the roof of the furnace, the roof very quickly deteriorates, but in the case of this new furnace the roof is not subjected to such severe strain owing to the reduced size of the electrodes and to the smaller amount of power transmitted through the arcs, therefore the maintenance of the furnace costs less and the roof lasts longer.

Thirdly, the control of the load factor of the furnace is

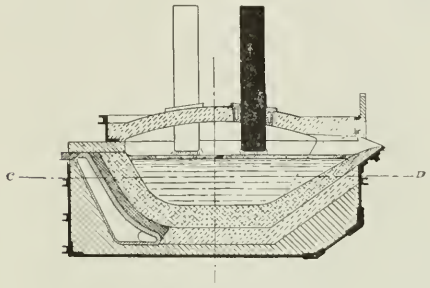


FIG. 3.—VERTICAL SECTION.

better because the current flowing through the resistance plates is a perfectly steady one, whereby the heavy current fluctuation otherwise caused by large arcs is compensated.

The result of this is that no special machinery is required either for compensating for the lower power factor of a true induction furnace or designed with a large flywheel effect, as in the case of a simple arc furnace, on account of its fluctuating load factor. Standard machinery and standard transformers only appear to be necessary, which, of course, decreases the first outlay for the plant, and under the circumstances may prove to be almost a deciding influence.

The power consumption per ton of steel should also prove more economical by means of this combination, as the high temperature of the steel is only required during desulphurization, it being well known that dephosphorization does not require such a hot slag as desulphurization, and it follows as a logical sequence that the higher the temperature of the steel for any length of time the greater also the heat losses during the process; in view of this, the total melting and refining process should be more enconomical with this combination.

It, therefore, appears that the type of furnace includes several of the advantages of the induction furnace, viz, a good circulation of the charge, inasmuch as it is heated from below as well as from above, and a high load factor.

No electrical apparatus, in the ordinary sense of the word, seems to be an element of the design of this furnace, because the power supply transformers may be situated in an adjoining building protected from dust and injury.

The low voltage current only requires to be transmitted to the furnace by means of insulated bus bars and cables.

The regulation appears to be simple, as if the furnace is operated direct from a generator the field rheostat controlling the current to the generator is all that is required; or, if transformers are used in connection with an existing power supply, so-called regulating transformers with interchangeable terminal steps are employed. In fact, comparatively little regulation in this respect should be found necessary, because the current in the arcs ought to be easily regulated by altering the length of the arcs, while the current in the resistance plates is more or less automatically regulated by the condition of the charge.

The author is informed by the owners of the patents, Messrs. The Grondal Kjellin Company (Ltd.), that furnaces of this type will shortly be tried.

Review of the Metallurgy of Copper in 1910.

By L. S. Austin.

Conveying and Transmission.

The belt conveying system at Cananea, which at first presented difficulties in operation, is now in continuous use. It is ideal in theory and it costs only half of any other effective system. R. R. Moore states that the conveying belt, as used at Garfield, has stood the wear and needed but little repair, and the system is now in satisfactory operation. It appears that when installed by engineers experienced in this special line and afterward reasonably handled the conveying belt is durable and smooth in operation. The management of the International Smelter at Tooele, where this system has been installed, is also apparently of this opinion.

In general, in feeding the blast furnaces, where the charge and fuel are weighed, the charge-car is commonly used. Of them all the Hodges car used at Granby works most rapidly and effectively. The load of the car can be adjusted to the condition of the furnace both as to quantity and position and can be dropped where desired. In those systems where the car delivers at the long sides of the furnace it is shot in more at random, and this is apt to show itself in irregularities on the surface of the charge, in more hanging up and in more overfire.

In Mexico, at the plant of the Tezuitlan Copper Company, Mr. Cyrus Robinson, the constructing engineer, convinced of the importance of a uniform charge both in the size of the ore as well as of its composition, takes every precaution to assure this. There are two lines of wire-rope tramway, one of 15 tons hourly capacity carrying limestone to the mouth of the mine, the other of 45 tons capacity taking both ore and limestone to the smelter. Storage is effected in steel bins 40 ft. high and 25 ft. diameter. There are five bins at the mine and twelve at the smeltery having a combined capacity of 3000 tons. The bin bottoms are of rocks dumped in to form the slope which is then covered by rails set at an angle of 45°. These cost but half what a rectangular wooden bin would come to. The ore is delivered to each of the bins in turn to secure even distribution. Finally it is drawn off into a large charge-car and the coke is added. There is an overhead feed-hopper of 65 tons capacity into which the car drops its load. This hopper is divided into partitions to half its depth. A row of gates handled by hydraulic cylinders enable any gate to be opened where desired, so that the ore is dropped where the furnace most needs it, thus securing more elastic feeding conditions. Ordinarily this is done by dumping as seems best into the side doors of the furnace and to some extent (though seldom done) by trimming the charge.

Roasting.

This is now largely done in one or another type of the McDougall roasting furnace. These furnaces have been improved in mechanical details. Some attempts have been made to do away with the water cooling, or at least to minimize its abstraction of heat from the charge. At one plant the rabble blades are slipped over the rabble arm and can be readily slipped off when they fail. But besides this the bladeholders have been shaped so that the flue dust lodges as a layer on them, so preventing the abstraction of heat. As a result less cooling water is used, the arms are preserved and the output of the furnace has been increased. At one of the larger plants the concentrates, carrying 25 per cent copper, really need but little roasting for reverberatory treatment. They are accordingly thrown upon the lower hearth of the roasting furnace, where sulphur is eliminated and where they become hot, and so in suitable condition for the reverberatory smelting to follow.

The dropholes of the McDougalls by which the ore falls from floor to floor are set opposite the sidedoors. Here the ore sticks to the brick work and is hard to remove. This brick surface has been replaced by iron plates and from them the accumulation can be easily barred off. R. R. Moore in his article on reverberatory practice speaks of brackish water as causing scaling of the water surfaces and pipes. The remedy for this would seem to be to use a good quality of water over and over by putting in a satisfactory water-cooling tower.

Reverberatory roasting produces more sulphuric anhydride than blast roasting, and we find the plants using the McDougall furnace successively putting in brick offtakes because of the short life of the sheet-iron outlet flue.

Reverberatory Smelting.

To get rapid melting of the charge in the reverberatory furnace we must burn the coal rapidly. But the gases escaping at the throat should be at least 1100° C. or at the melting point of the slag. Otherwise we encounter the difficulty experienced at first at the Garfield Works when the draft was poor, viz., the setting or crusting of the slag at the front end of the furnace. But with hot gases the question arose how to utilize the heat of these gases and this has been solved generally by using it to produce steam for power purposes.

In open-hearth steel practice the problem has been differently solved, that is by recuperation, by which the hot gases go to a brick checker work and upon reversal of the current air is drawn through the checker work, which adds thus to the intensity of combustion. In the open hearth the temperature is built up by a series of reversals of draft, so that care has rather to be taken to avoid so high a heat as to melt down the roof of the furnaces. It has been stated that the rate of transmission of heat to the charge or the rate of melting will vary with the square of the difference of temperature between its melting point and that of the burning gases. Thus, assuming the melting point of the charge to be 1100° C. and that of the gases 1500° C., melting will proceed at a certain rate due to the difference of 400° C. If the heat of the gases is increased by 100° C. only, or to 500° C. in all, the rate of melting will be increased in the ratio of 16 to 25, or 56 per cent. Is it not better, then, to adopt the open-hearth idea of utilizing the heat to the best advantage for the object in view rather than to employ it for an ulterior purpose? It reminds me of a similar problem in blast-furnace work. Is it better to effectively briquette flue dust or to so run the furnaces as not to make any?

An advantage possessed by the reverberatory furnace over the blast furnace is that the charge can be greatly varied in composition and yet good work be done. The work may be slower, but so long as the slag flows it will be clean. On the other hand, in the blast furnace a viscid slag eventually refuses to issue from the taphole and the furnace "freezes up."

Another advantage is that flue dust is made only at the time the charge is dropped. So little is the loss in this way that no damper is closed at the time of dropping. The passages to the main flue in ordinary practice are large and open and there is little accumulation of dust to interfere with the draft. At Great Falls, where the regenerative open-hearth furnace has been tried, it is claimed that the checker work has been partly closed by accumulating dust. This difficulty should be overcome by a design such that the dust can be removed as it accumulates, and so that the regenerators can be repaired without serious dismantling.

The tendency to-day in copper mining, exemplified in the porphyry coppers, is to handle lower-grade ores that necessarily require concentration. It is these concentrates, too fine for treatment in the blast furnace, that seek an outlet for treatment by the reverberatory. To obtain tonnage in the latter furnace its length has been increased step by step until now a furnace 100 ft. long appears to give the greatest satisfaction. To retain the heat the walls of the furnace have been increased to 30 in. and the roof to 15 in. or 18 in. thickness. So heavy a roof, however, limits the width of the furnace to about 20 ft. and smelting area must be got by increase of length.

The practice to-day seems to confine itself to dropping the charges near the firebox end and to depend upon the charge spreading out gradually owing to its tendency to flow while hot and fine. It accordingly floats slowly toward the front of the

furnace, melting as it goes and dropping out its burden of matte as fast as the matte forms. The result is a cleaner slag. It appears probable that this practice has come about because the heat was so rapidly abstracted from the burning gases by the charge that the fire end was the only effective place for the melting. What might be the effect of regeneration and at the same time of an increased melting area by dropping at other points, as was formerly done, remains to be seen.

The Coal Bed.—In Wales the importance of a deep firebed was long recognized. Clinkers were allowed to accumulate on the grate bars to furnish a bed for the live coals, and by pushing aside the grate bars those clinkers were removed and the fire kept clear. In firing the firebox was filled to the top of the bridge wall and coal was used to close the firehole, thus dispensing with a firedoor. It is along these lines that firing is yet done, generally using the old style of grate bar, which can be so well manipulated. Still, to burn coal rapidly frequent grating should be practised, and this means a loss of perhaps 10 per cent of the fuel as coked droppings. This coked material can be washed for the removal of the ash, or otherwise burned with other fuel for steam making. The depth of the bed may vary from 2 ft. to 4 ft. as experience indicates, but even the deeper beds occasionally burn with holes. The coal is dropped from hoppers and consequently does not get even distribution on the firebed. While doors have been provided for dressing the surface of the fire, it is hard to see its condition in the highly heated firebox and so air may get through to the hearth and thus cool the melting zone. Experiment with the larger furnaces shows that, especially before a new firing, 50 per cent more than the theoretical quantity of air may easily get in, with consequent loss in the intensity of melting heat. It makes one disposed to recommend the down-draft method of burning coal such as Hawley has instituted for boiler firing. Here the upper surface of the coal bed is easily seen and the chance of irregular burning is avoided. Again, the fine, partly burned coal, dropping down through the water-cooled firegrate, is burned on a subsidiary grate below.

At Cananea, where the first reverberatory was fired by powdered coal, the practice was given up in favor of oil, and so satisfactory has been the performance that a second furnace has been installed. One notes that when powdered coal was used the ashes from it accumulated in the outlet flues to the waste-heat boilers and there slagged, at the same time choking the flues. Using oil and allowing for the heat utilized for steam making, the net charge for smelting was 0.43 bbl. to 0.57 bbl. of oil per ton of charge smelted, and this too while melting a mixture of 37 per cent flue dust with 51.5 per cent calcined concentrates.

The Blast Furnace.

While the reverberatory has received a large measure of attention from metallurgists through the year, blast-furnace operation must not be forgotten.

Years ago W. L. Austin laid down the proposition that for real pyritic smelting 65 per cent of pyrite at least should be in the ore. Whether this figure is final or not, at least most of the sulphide ores treated need fluxing, and when this is true then coke must be employed to supplement the heat yielded by the sulphides themselves. A furnace in full operation accumulates hangings or accretions on the walls, but with jackets extending to the feed floor, or to the top surface of the smelting charge, such accretions are easily burned off. Formerly, when the walls were of brick, Austin proposed to prevent such hangings by "column charging": that is, the charge close to the walls was to be of silicious or at least non-agglomerating ore, while within this and constituting the core of the charge were to be fed the sulphides so liable to sinter. I have tried it experimentally, but never have seen it adopted elsewhere.

There is less bosh in copper practice (7 in.) than in lead smelting, as the duty of the furnace in the first case is to do its most active work at the focus near the tuyeres and to complete the melting at the hearth. That this is by no means generally well done is evinced by the slagged tuyeres which must be continually poked to keep them open, and by the very considerable overfire as compared with silver-lead practice. Copper furnaces vary greatly in respect to overfire, and I have noted that when a self-fluxing ore is smelted there is less overfire than where much limestone flux is used. In spite of the endothermic effect of the limestone at the top of the charge, which effect continues up to a temperature of 800° C., the hot top prevails. Slag formation appears to be slow and the fire creeps up and keeps on top. With a fresh charge all looks well again, but before the next one is put in the gases have again risen to 700° C. to 800° C. and the furnace on top is a mass of seething flame, difficult even to look at. Meanwhile the blast is driven through narrow openings produced by the rodding of the tuyeres at high pressure and the hearth, except with excessive fuel, is too cold. We are not to think that this condition is universal. Lower blast pressures, good feeding and suitable slag composition assist in improving these adverse conditions.

Blast furnace jackets are now well built. The inside plate next the fire may be 5/8 in. for stiffness, the outside 3/8 in. Again there are no joints that can be attacked by the molten materials of the furnace, and jackets are made with welded joints especially at the tuyere opening.

The interior width of the blast furnace varies from 48 in. to 56 in., seldom more, and capacity is attained by adding to the length. Even at the Granby Works an addition of 3 ft. to the length is admittedly an advantage. Accretions are more easily removed and the active zone within the end jackets has been proportionately increased by the addition.

Increase in the height of the smelting column has resulted in the saving of fuel, but at the expense of the grade of matte, since the lower column with its greater overfire has caused the elimination of more sulphur.

At Cananea the crude ore charge carries 6.4 per cent Cu and hence, to get a good grade of matte, sulphur elimination is not so important as in the lower-grade ores. The matte fall is then 18 per cent and the matte carries 37.4 per cent Cu. I may add that they claim there is no trouble to keep the tuyeres bright and that the slag crust at the tuyere level is not hard. The tuyeres are barred or cleared out three times daily and are punched every three hours.

Converter Practice.

The great event of the year has been the development of basic copper converting. The process, using a Peirce & Smith basic-lined converter, was introduced at the Garfield plant of the American Smelters Securities Company, and of late these converters have been added to until there are six, five in continuous operation, the sixth as a reserve.

The converters are of the barrel type, 12 ft. diameter by 26 ft. long. The Peirce & Smith patents provide for the expansion of the basic brick-lined interior and for tuyeres that will not be broken by such expansion. The basic process itself is not patentable nor even the lining mixture used. In the United States basic-steel converting has been but little used, but its methods have been thoroughly developed in Europe and are a great aid to those who are now undertaking the conversion of copper matte.

A converter of the size above described will produce 40 tons of blister daily and will output 1800 tons to 4200 tons of blister copper on one lining with minor repairs at the tuyere region. As will be readily understood a low-grade copper matte presents no difficulty in treatment, so different from the case with the acid-lined converter. Within the year the Yampa smelter found it more profitable to sell matte to the Garfield works than to treat it locally. This matte was of too low grade to treat in the acid converter. Later on the Yampa plant was abandoned altogether and the mine output of 600 tons daily turned over to the Garfield works. When the latter plant lost the treatment of the Utah Consolidated ore they were glad to replace it with Yampa ore even at a low treatment rate.

To go back to basic converting: The Anaconda company has,

since last spring, been trying it out in ordinary converter shells with a high degree of success. They use as before the Repath independent tuyere and take up expansion by the use of wooden liners inserted in an occasional longitudinal joint. Around the tuyeres in place of the brick lining comes a packing of basic material with a suitable binding. In this way rapid work is done and the lining is comparatively stable and tight as against air leakage.

The apparent advantages of basic converting are many, especially for a new plant. Since the converter shells are not lifted by traveling cranes, and only such cranes are used as are necessary for handling the ladles, the superstructure can be made lighter, or much handling effected without the use of cranes. We thus do away with the detrimental effects of the sulphur fumes on the crane man.

While time is lost in operating a basic converter, such loss is less than with the acid-lined one.

Silicious ore can be freely used without needing clay or clayey slime as a binder, but at some plants such use is desirable because there is plenty of clayey slime made at the nearby concentrating plant.

The lining department and its machinery and upkeep, with the force needed for lining, is saved; there is but little drying out of linings required.

There are certain disadvantages in operating, but they are being overcome with greater knowledge.

In connection with the above, Arthur M. Day has developed a process, the outlines of which are not yet forthcoming, by which it has been possible at McGill, Nev., to produce slags running as low as 10.2 per cent SiO_2 and as high as 77 per cent FeO, or indeed any grade of slag desired. It can be readily seen that the quantity of slag is thus largely diminished, and in fact the whole process is hastened. On the other hand, in the acid-lined converter, the life of the silicious lining is correspondingly lengthened.

The Smelter Smoke Problem.

Copper smelting works have received severe lessons by disregarding the emission of smoke resulting from roasting and smelting the ores. Offensive as are sulphurous-acid fumes, the damage inflicted has been largely due to the presence of small quantities of sulphuric acid which has formed sulphates, to arsenic and particularly to solid particles of flue dust.

A variety of methods have been tried at great expense to overcome the difficulties, and the most successful has been the bag house proposed and tried 25 years ago for the saving of lead furnace fumes. When made of cotton the bags last but a short time if there is any SO_3 present in the gases, and even woolen bags have too short a life. The problem has been how to dispose of the comparatively small quantity of SO_3 to render it innocuous. This has been accomplished at the United States Smelting Company works in Utah by neutralizing the sulphuric acid by means of zinc oxide admitted to the flue where the escaping gases travel.

The Cottrell process of electrostatic precipitation of the dust is also making headway. In a small unit it has given satisfaction and now we learn that it has been worked up to a successful issue at the Coram plant of the Balaklala Copper Company. A single unit consists of a chamber 10 ft. x 10 ft. with a hopper bottom for the collection of the fallen dust. Within the chamber is a series of positively charged bars set between iron plates. These bars are made of two copper wires covered with strands of asbestos. The bars carry a current of high potential and give a positive charge to the dust particles which precipitates them on the low potential bars of iron. The plates are then jarred to shake the dust thus collected to the hopper bottom.

Salt Lake City, Utah

The Camp Bird, Ltd., Ouray, Col., treated during the month of October, 1910, 6827 tons of ore, producing 7383 oz of bullion and approximately 716 tons of concentrate. The statement for that month shows $109,000 applicable to profit.

A New Form of Electrode.

By A. Mahlke, Ph.D.

In this journal a number of articles has been published about the heat losses through the electrodes in electric furnaces. In the discussion of this object a point has been overlooked which might be of some importance in practice. The electrodes which were used in the investigations for determining the heat losses had a much smaller cross-section than the furnace. Consequently the current density in the furnace charge near the end of the electrodes (see Fig. 1) was much greater than in the interior of the furnace further away from the electrodes. The reason is, of course, that most of the lines of current start from the end of the electrodes, because the electrode resistance is much smaller than the resistance of the furnace charge and the current passes, therefore, through the electrodes as far as possible.

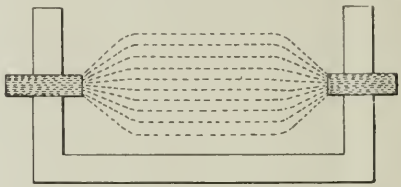


FIG. 1.—LINES OF CURRENT IN FURNACE.

On account of this greater current density in the charge near the end of the electrodes, the charge is at a much higher temperature near the ends of the electrodes than in any other part of the furnace, so that there is always a great difference in temperature within the furnace.

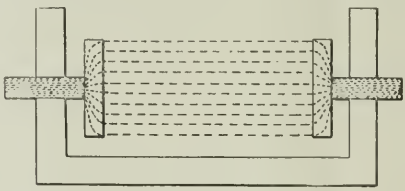
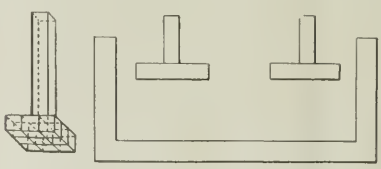


FIG. 2.—MODIFIED ARRANGEMENT.

This difference in temperature will exist during the whole process (though decreasing at the end of it), as long as the current in the furnace generates near the inner terminals of the electrodes much more heat than in the interior of the furnace, where the current density is less.

This difference in temperature also causes other inconveniences. The electrodes are heated to a much higher temperature than is necessary for the process in the center of the furnace and are burnt off to a greater degree and cause a greater heat loss than if the temperature of the whole interior of the furnace was uniform. To avoid these inconveniences one should fix at the end of the electrodes big plates of well-conducting carbon material, as shown in Fig. 2, which have the same cross-section as the furnace itself. These plates could be fixed to the electrodes by screws, wedges or any other contrivance made out of carbon, such as are used when short



FIGS. 3 AND 4.—ELECTRODE CONSTRUCTION AND USE IN ARC FURNACES.

electrodes are to be jointed in order to have electrodes of greater length.

By using such electrodes one would have the same current density throughout the whole furnace (see Fig. 2), and so the same amount of heat would be generated in each part if the resistance is the same everywhere in the furnace.

It would be practical to make the carbon plates thick enough to give them a considerable amount of heat capacity so that they can take a good deal of that heat generated in the part of the electrodes between the plates and the walls of the furnace, this part being heated to a higher temperature than the plates on account of the greater current density. But since some portion of this heat is conducted through the plates into the interior of the furnace, the heat loss is diminished as well as the burning off of the electrodes.

For resistance furnaces one could make such electrodes by packing carbon blocks around the end of the electrodes, as shown in Fig. 3. The resistance between the electrode itself and the blocks could be lowered by fixing the blocks to the electrodes with a graphite paste.

While these electrodes would be especially suitable for resistance furnaces, they might also be used for arc furnaces. The big mass of the plates forming that part of the electrode from which the arc arises (Fig. 4) would not get so hot as the thin end of the electrodes now generally used. So the amount of electrode material necessary for the process in the furnace might be decreased, which would be of importance for many purposes. The big plates only would be renewed, while the other parts of the electrodes carrying the plates could be used for many furnace runs.

Progress in Cyaniding During 1910.

BY R. W. GORDON.

So many specially trained engineers and chemists are now engaged on original work in connection with cyaniding that improvements are yearly increasing in number, if not in importance. The employment of educated men on mill work, too, has led to a more liberal policy in the publication of special information, both in the technical press and through constant interchange of visits and experience between cyanide operators. Progress, therefore, is proportionately more rapid as each man can commence his original investigations where his neighbor left off.

Nothing revolutionary has occurred in the cyanide field in 1910, improvements being chiefly confined to details, but the first mill using a continuous process throughout was placed in operation during the year, and the same mill adopted, with reported success, chemicals for aiding cyanide in the dissolution of values from a sulpho-telluride ore.

For fine crushing and grinding, especially on hard ore, stamps followed by tube mills have been specified in practically all the new mills, and much has been done in the past year, both in this country and South Africa, to properly apportion the work between them. Coarser and coarser material is being fed to the tube mill, and the point at which the cost of power and pebbles outweighs the initial expense and maintenance of stamps is a most important question, and one which must be solved for each installation.

If it were not for the nuisance of elevators and screens, rolls, crushing either wet or dry, would often replace stamps ahead of tube mills, and several plants using rolls and tube mills will be placed in operation early this coming year.

Chilian mills have held their own for crushing moderately soft material, and are attractive as occupying less space and costing less per ton for installation when foundations, bins, and buildings are considered. At the Goldfield Consolidated Mill a notable installation of these mills was made in 1908-9, where six Chilian mills were introduced between 100 stamps and six tube mills for intermediate grinding. At a comparatively low expense and without any addition whatever to the bins or building the capacity of the mill was materially increased. No data have been published to show whether any great economy has been effected by stage crushing and the elimination of the final undersize between each stage, but in a mill of this size stage crushing would certainly appear to be advantageous, for conditions could never be more ideal.

Considerable discussion is still taking place in the technical press between advocates of tubular and conical mills. So far no reliable data has been produced to show that conical mills are more efficient than the standard tubular type. The former seems to have given entire satisfaction in some places, and in others, again, particularly where a comparison has been possible, it has been discarded and replaced by the tubular. The use of steel rods instead of pebbles as a grinding medium in tube mills has been suggested and tried out on a small scale, but no millman has yet been induced to make a working test. Rather extravagant claims are also made that by the use of these rods material can be taken direct from the crusher and reduced to the necessary fineness at one operation, which is certainly a desideratum.

The use of shorter tube mills has been urged with the idea of removing as quickly as possible material which is already fine enough and returning the oversize to the feed end for retreatment. On the other hand, at El Oro a special attachment was recently added to the discharge end to prevent the oversize from leaving the mill. It would seem as if the accumulation of oversize at one end might have the effect of causing unequal liner wear.

It is now thoroughly understood that the efficiency of a tube mill depends almost entirely upon the consistency of the feed, excessive moisture being detrimental in every way. A dewaterer, therefore, as much as a classifier of coarse and fine material has become an essential part of tube-mill equipment. In individual cases, the Bayliss and Akin classifiers, which operate on the screw or worm principle, first used by George Moore at Mercur in 1903, have been installed for this work, and the Caldecott diaphragm cone is also used, but the Dorr classifier in its improved form was generally specified in 1910 installations, and appears to have given entire satisfaction.

In the year 1909 the continuous movement of pulp through a series of agitators was first adopted at the San Rafael mill in Mexico, after experiments on a working scale, in which both charge and continuous agitation were thoroughly tried out. At first thought it would appear possible for a particle to pass through three or four tanks without the necessary period of contact, but practically this chance is proved to be almost negligible and continuous agitation is now being used in many important plants, such as the Liberty Bell, in Colorado, and the Esperanza, in Mexico. By its means objection to the use of very deep air agitators has been overcome, as so little headroom is needed between the points of intake and discharge. Great care must be taken, however, to draw from one tank into the next a pulp having exactly the same consistency as that fed in, otherwise there will be an accumulation either of fine or coarse material. Such an adjustment can easily be made, however, and once made the agitators require practically no attention. This method has been applied successfully to so-called Pachuca tanks, to Hendryx agitators and to other forms of mechanical agitators.

With the objection to great depth removed, the Pachuca tank has become more than ever the favorite form of agitator, largely on account of the removal of all wearing parts from direct contact with the pulp. Among other agitators now in the market might be mentioned the "A Z," lately introduced from Australia, which employs practically the reverse of the Hendryx principle, and forces material from the center along the flat bottom of an ordinary shallow tank and up its sides. Although it consumes considerable power, it presents advantages over other forms, where heavy materials, such as sulphides, require agitation.

Continuous settling having been furnished by the Dorr thickener, and continuous agitation being also an accomplished fact, the need for a simple continuous method of separating valuable solution from valueless solids after agitation, or rather displacing the former by water, became more apparent.

Years ago, and before the invention of modern mechanical filters, an ingenious method of continuous dilution or decantation was evolved by John Randall in a small wet crushing mill near Deadwood. Intermittent decantation with its volumes of solution was then the order of the day, when Randall conceived the idea of reducing the amount of solution in circulation by using one solution only, and that a mixture of his final barren solution and water washes. This solution he continuously decanted off the final tank, used it for diluting the preceding one without return to the stock tank, again decanted it, and again used it for dilution until the first tank was reached. It thus gradually became enriched sufficiently to be worth precipitating, while the pulp moving in the opposite direction became impoverished. Randall's scheme worked perfectly on paper, but, unfortunately, the mechanical means then available for settling and pumping were unsatisfactory. It remained for Dorr to adapt the idea successfully to a series of his thickeners, first at the El Palmerito mine in Mexico and later in this country.

A simple continuous filter which will give perfect displacement is still badly needed, however, as continuous decantation yields tailings high in moisture and also, unfortunately, high in cyanide, especially if strong solution is in general circulation in the mill.

The Moore filter suit still drags along, and it is an open question whether Moore will eventually control the whole vacuum-filter field, as he undoubtedly deserves to, or whether it will be thrown wide open.

Merrill's pressure filter is being installed at the new Gertrudis mill, but it requires special conditions, whereas movable or stationary vacuum filters are applicable everywhere and give uniformly good results. Various forms of continuous revoluble filters have been placed in operation during the year, but so far they are too complicated to be widely adopted, and none of them compare with the intermittent vacuum filters in the displacement of valuable solution. South Africa has at last followed this country in adopting vacuum filtration instead of intermittent decantation, and we presume will shortly be trying out continuous agitation and decantation also.

Separate treatment of sand and slime is seldom considered in a modern mill, but mention must be made of a novel method in use at the new Portland mill, in Colorado. There separate treatment is used, but the commercial values in the sand are in solution when the classifier is reached, and all that is needed is a wash.. The thickened sand from the classifier is, therefore, led through a second classifier—or, more properly, a dewaterer—containing barren solution, and from there, through a third containing water, to the dump.

The possibilities of the Clancy chemical process have become more and more apparent, and the reported success of the Crow-Tippet method at the Portland mill has but whetted the appetites of producers of sulpho-telluride ore for a mill operated under straight Clancy principles. Such a mill will shortly be built in the Cripple Creek district, and a satisfactory economical outcome would appear highly probable since the Clancy claims have been published. The field for such a process is large and profitable, as quantities of ore are known to exist containing values in sulphide form, but are too low grade to afford roasting. The operation of the first plant will, therefore, be watched with much interest.

For precipitation in the larger mills the use of zinc shavings seems to have given place almost entirely to zinc dust, the press generally installed being the Merrill triangular press, which was perfected at the Homestake. One great argument in favor of the use of a zinc dust press is safety of the precipitate, either in case of fire or of light-fingered employees.

On the whole, the year 1910 was well spent by the cyanide fraternity. Costs were reduced, tonnage increased and more bullion produced by the cyanide process than ever before.

A Vertical Magnifier

By H. M. Boylston.

The little instrument[1] illustrated in Fig. 1 was devised in this laboratory as the result of a long search for a simple, convenient and inexpensive means of measuring the impressions made in metal specimens by the Brinell ball test for hardness.

It consists of a doublet lens made up of two plano-convex lenses having a combined focus of 38 mm. and adjusted in a cylindrical mount about 3 in. high and open at the bottom, as shown in the figure. Provision is made for vertical illumination and a small scale is added for measuring small distances accurately. The focusing glass used by metallographists makes a satisfactory basis for constructing the instrument, the lenses in this being specially corrected for flatness of field and sharpness of focus.

A brilliant illumination is obtained by inserting a plain glass reflector *G* in a slot cut in the cylindrical mount at an angle of 45°, one side of the cylinder being left open to admit light from any convenient source (daylight, incandescent electric lamp, Welsbach gas burner, etc.). The path of the light from its source to the specimen to be measured or examined is indicated by arrows in the illustration.

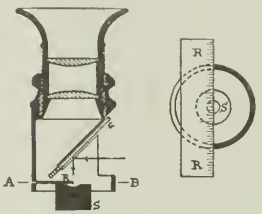


FIG. 1.—VERTICAL MAGNIFIER.

A thin steel scale *R* with proper graduations on one edge is inserted near the bottom of the cylinder in a slot cut for that purpose and is adjusted so that the graduated edge corresponds to a diameter of the cylinder.

The instrument is held in one hand with the opening directed toward the source of light, while the specimen *S*, with impression uppermost, is held against the steel scale with the other hand in such a position that the scale corresponds with a diameter of the circular edge of the impression, one of the graduations of the scale being set tangent to an edge of the impression. The diameter of the impression is then easily read off on the scale which with the impression itself has been magnified considerably by the lenses of the instrument. A scale graduated to 0.2 mm and hence easily read by interpolation to 0.1 mm has been found very satisfactory, since no greater accuracy than this is warranted; for the impressions obtained in this test often are not perfectly circular and the lip of the impression is often rounded instead of being sharp. Steel scales graduated to 0.1 mm can be obtained, however.

Although designed primarily for the purpose just described this instrument has many other uses. It can be used to great advantage in examining fractured surfaces, or for examining the polished surfaces of metallographic specimens for flaws in the polishing, or for minute blowholes. It will also lend itself readily to any case where a low power magnifier is needed or where small distances are to be measured rapidly with the degree of accuracy already indicated. It is hoped that the magnifier will be found useful in many cases, in the machine shop, for instance.

The focus of the instrument may be adjusted within limits for large specimens, in which case the scale is removed from the slot and laid directly on the object to be measured, the magnifier being placed above and against the scale.

Metallographic Laboratory of Harvard University.

[1] Another somewhat less convenient, but fairly satisfactory device of this sort is described on p. 53, "Laboratory Experiments in Metallurgy" by Sauveur and Boylston.

Metallurgical Progress in Utah and Nevada During 1910

BY ROBERT H. BRADFORD

Salt Lake City is conceded to be the most important smelting center in the United States. The tremendous ore supply of Bingham, Park City and Tintic gives a combination of easily smelted mixtures and has justified the building of exceedingly large smelting plants near Salt Lake City. The favorable position of this region as a railroad center enables the smelters to draw large supplies of ore from all parts of Utah, from Nevada, Idaho and even from California. The ore supply is more diversified than anywhere else in the Union.

Large custom smelters with their keen competition for custom work have given smelting rates more favorable to the seller than those obtaining elsewhere.

The large companies maintaining custom plants in this region include the American Smelting & Refining Company, the United States Smelting, Refining & Mining Company, the International Smelting & Refining Company.

The Murray plant of the American Smelting & Refining Company has been operating during the year at somewhat reduced capacity as has been common to most of the lead plants of this company. The improvements in the pot roasting process have reduced the cost of roasting the sulphide ores and have given very desirable materials for blast furnace smelting. Record runs for lead blast furnaces of dimensions 46 in. x 168 in. have been made during 1910. The greatest tonnage for a single day was 318 tons of ore and flux, exclusive of fuel and returned slag. Average daily tonnage for one furnace for a whole month was 280 tons. These figures show the highest tonnage yet reported from any lead smelting plant.

The Garfield plant, a copper smelter, has operated on increasing ore supplies during the year. The concentrates from the gigantic concentrating mills of the Utah Copper Company, from the new mill of the Ohio Copper Company, along with ores and concentrates from numerous mines of the surrounding country have taxed the reverberatory and blast furnaces to their limits.

The basic lined converters of the Peirce & Smith type have done excellent service. By means of the process using basic lined converters the cost of converting at Garfield has been notably lowered. The converter shell is 23 ft. long by 10 ft. in diameter. Next to the shell, along the bottom, is a lining of ordinary fire brick varying in thickness from 4 in. to 13½ in. This lining is thickest along the bottom and becomes thinner as it rises to the sides. Inside this lining is the basic lining of magnesite bricks. The circular throat is at one end. Matte and silicious ore are charged through the throat. Thirty two cast-iron tuyeres are provided, each with special valve and flexible hose connection to the air main. An opening called the breast opposite the tuyeres provides for the removal of the slag and metal.

The initial charge of matte consisting of four or five ladles of matte (40 tons) is poured in through the throat and about three tons of dry silicious ore are added. The blast is introduced and the converter rotated until the tuyeres are well below the surface of the charge. When the blow is well begun the blast pressure reaches 10 lb. to 12 lb. per square inch. When the silicious ore has been fluxed the slag is poured and additional amounts of matte and ore introduced. The operation is repeated and continued until enough "white metal" is formed, when the addition of matte and ore ceases and the white metal is blown to blister copper.

Care is taken to thoroughly dry the silicious ore before charging it into the molten matte to avoid explosions.

At the United States lead smelter at Midvale the arsenic plant is of interest. The blast furnace flue dust collected at the bag house is returned to two Brinton roasters having rotating hearth and stationary rabbles. Each furnace has capacity of 10 tons per day. The dust heated to redness in this furnace is in large part volatilized. The residue from the hearth goes back to blast furnace. The fumes from the Brinton furnace consist principally of arsenious oxide (As_2O_3). The sublimed arsenic product is collected in zigzag flues built of brick masonry. The product from these flues is taken to a reverberatory furnace and revolatilized or resublimed, the fumes again collected in zigzag flues. The product from the second set of flues is practically pure As_2O_3. It is taken from the flues in cars and transported by a monorail tram to the packing room, where it is packed in barrels with capacity of 400 lb.

Many improvements in converter roasting are being perfected at the United States plant. Much flue dust is being sintered into a form suitable for blast furnace treatment. None of the ores fed into the rectangular converters receives any previous roast. Thin layers of ore and short time for roasting a charge have increased the capacity of the roasting plant. Careful mixing and moistening of the ingredients have assisted materially in the converter roasting.

The International Smelter at Tooele, Utah, was put into commission on July 25, 1910. This plant is entirely new and thoroughly up to date. Construction work was commenced Dec. 9, 1908, just 19½ months before the first furnace was blown in. Considering its extent and the massive construction this plant holds the record for speed of construction.

The smelter was built primarily to treat the ores from the Utah Consolidated Mining Company's mines at Bingham and the South Utah mines at Newhouse, Utah, but is provided with capacity to treat also much custom ore. The Highland Bay smelter at Murray originally treated the Utah Consolidated ores, but this plant was closed down as the result of injunction proceedings brought by the farmers of Salt Lake County.

The ore is sent from Bingham to the International by a Bleichert aerial tramway four miles long built over a very rugged route. The tramway has a capacity of 125 tons per hour. The receiving bins at the smelter are of steel with a capacity of 10,000 tons. The ore from these bins is conveyed to the sampling mill by belt conveyors, where it is crushed and sampled. From the sampler the ore is conveyed by belt conveyors to the receiving bins at the McDougall roasters. From these bins the ore is again conveyed by a belt conveyor to the charge hoppers of the McDougall roasters. In transit from the bins to the roaster plant the ore is weighed automatically as it passes over a Blake-Denison continuous weighing machine.

The roaster plant contains 16 McDougall furnaces of 45 tons daily capacity. From these furnaces the ore drops into hoppers arranged directly over the tracks of the electric tramming system for transportation to the hoppers of the reverberatory furnaces.

Five coal-fired reverberatory furnaces 19 ft. x 102 ft., with a grate area of 7 ft. x 16 ft., are provided, each with a capacity of 300 tons in 24 hours on natural draft. The flame from each furnace passes through a 746-hp Stirling boiler and develops 600 hp of energy. The slag is skimmed into slag cars of 225 cu. ft. capacity and hauled over the electric tramming system to the slag dump. The matte tapped from the side of the furnace runs through launders directly into the converters.

The converters are of the barrel type 96 in. x 150 in. and are operated electrically. A 60-ton electric traveling crane is provided to serve the converters. The casting room has a 30-ton crane and there are two small 12½-ton cranes used to transfer the slag from the converters directly to the reverberatory furnaces in the molten condition.

The metallic copper is poured into the casting ladle and cast into bars ready for shipment to the refinery.

The following notes were made and published* by the writer on the occasion of the visit of the Utah Society of Engineers to the International smelter:

*Salt Lake Min. Rev., Nov. 15, 1910.

A striking feature of the new International smelter as observed on the trip was the massive steel construction everywhere apparent giving large roomy buildings with high roofs affording excellent ventilation. Surely the idea of supplying the workmen with good air is worked out successfully.

The means of transporting materials about the plant was a most interesting feature. The standard broad gage tracks for the electric locomotives allowing for the movement of large railroad cars anywhere about the works, the free use of the belt conveyor with its automatic tripping and weighing devices, the unusual adaptation of the electric traveling cranes for ladles and converters provide an ease and economy of movement unsurpassed in modern smelting practice.

The careful sampling along with the exact weighing of materials makes it possible to check one department against another and thus place the responsibility for any losses due to careless operating at any stage of the smelting process.

The plant has an excellent location, being near the mines supplying most of its ore, the mountain range intervening being of little consequence, thanks to the aerial tramway; also near two through railroad lines and yet away from the agricultural districts. The gently sloping site, allowing for extensive building construction on any one level, and yet with enough relief for the materials of the smelting operations to be moved horizontally with but one lift, combines the good features of the terraced and the level site. The extent of the grounds laterally will allow of the addition of extra units with ease.

The simplicity of the process by which they pass from the mine ore to copper bullion illustrates the tendency of most recent commercial practice, but three operations being required, namely, roasting, smelting and converting. Contrasting this with the Welsh method in operation in Wales when that country was looked to as the leader in copper smelting, we see the operations cut to one-third. This Welsh method involved nine different operations, consisting of partial roasting, smelting, another roasting, a second smelting, etc., each process consuming fuel, material and energy far out of proportion to the advance of the resulting product toward the copper bullion desired. The "conservation of resources" is everywhere apparent at the International. The sulphur of the ore in the McDougalls furnishes all the heat for roasting and part of that needed in smelting. The roasted ore is not cooled, but charged hot into the reverberatories where it receives the additional heat necessary for fusion. The fuel of the reverberatories gives heat for the fusion and for the generation of steam more than sufficient for all power about the works. The matte in the converters is kept sufficiently heated by the burning of its iron and sulphur by the air forced into it so that the metallic copper is taken out well above the fusion point, allowing it to be poured slowly into the molds ready for market.

Cyaniding.

Utah enjoys the distinction of having the oldest cyanide plant in the United States. The Mercur mill was the first to try out the McArthur-Forrest process in this country. The Mercur mines are still producing handsomely and the ore, while not treated now in the original mill, is still responding to the cyanide process in the improved De La Mar mill now owned by the Consolidated Mercur Gold Mining Company.

Until the beginning of the present year the roasted ore was not classified and treated in the filter plant, but was kept separate from the oxidized ore. Now the roasted ore is mixed with the oxidized before the cyanide solution is added and all ore is classified and either leached or filtered according to its fineness.

An increased extraction is claimed from passing air up through the leaching pipes into the sands at intervals during the leaching. Experiments are being carried out with a view to using zinc shavings instead of zinc dust for the precipitation of the gold.

The company is now retreating old tailings to the extent of 100 tons or so per day. The supply of base ore from the mine is amply sufficient to keep the mill supplied with the 700 tons per day required.

The old tailings dump at Manning, four miles from Mercur, is being treated by the Manning Leasing Company. These tailings from the old Mercur mill carry $2 to $2.50 gold per ton. About $1 of this value remains in the tailings after present treatment. The company is using the Holderman filter process very successfully. It treats 100 tons of tailings per day at a cost of 75 cents per ton. The tailings from the old dump are conveyed to bins and sluiced with cyanide solution from bins into Chilian mills. All material passes through 18-mesh screen and is run by gravity to Holderman tanks fitted with filter leaves. The solution runs by gravity through canvas leaves and then to the gold tanks. Wash water or weak solution is run into tanks for 16 hours and the pulp is then sluiced over the dump. All clear solution from filter leaves goes to gold tanks and again through Holderman leaves to zinc boxes. The gravity filter leaf in the gold tank does excellent service, as is evidenced by the crystal clear solution as it enters the zinc box.

The Daisy Combination mill at West Dip, five miles west of Mercur, is milling 140 tons of ore from the company's mine. The process employed here is much the same as that used by the Consolidated Mercur mill.

The Utah Leasing Company, operating the Old Telegraph Mine at Bingham, has built a 50-ton cyanide mill to treat the silicious ore that the former company had left in the mine owing to high penalties for silica imposed by lead smelters. The ore is first crushed by a Ferraris-Blake crusher to ½ in. in diameter, then fed by an automatic plunger feeder into a tube mill. Cyanide solution is added as ore enters the tube mill. From this mill the pulp, after addition of more solution, is classified. Sands return to the tube mill. All slimes are agitated in Pachuca tanks for 36 hours. Pulp is then transferred to 100-ton settler. Thickened pulp goes to Kelly press and all clear solution to zinc boxes. The cost is $1.50 per ton for an extraction of 82 per cent.

Nevada.

During the years 1906 and 1907 the Western mining States enjoyed a period of prosperity previously unheard of. No one camp or no two or three camps were responsible. The entire mining West shared in the prosperity. If any State received more attention than another Nevada was that State. Her mountains were hurriedly prospected, with most flattering results. Many rich gold camps sprang up and the gold and silver ore taken from the surface outcrops created such a stir that it became common understanding that a gold prospect in Nevada was more attractive than such a prospect anywhere else. In fact it was not uncommon for the prospector to succeed in selling his claims simply because he could show they lay in Nevada. Purchasers seemed willing to buy geography.

It is not strange that many "wild cat" schemes were floated and that many investors lost their money. Nevertheless the fact remains that the gold and silver properties of real value opened up at that time and later show a most wonderful richness, and Nevada is now and will continue to be one of the foremost States in the production of the precious metals.

During the past year the State has furnished the mints with gold bullion in advance of any previous year. The mines in all parts of the State have added their quota, making a total that is enviable. While the large metallurgical plants like those of the Goldfield Consolidated, the Tonopah Mining, the Montana-Tonopah, etc., have operated regularly with large monthly shipments of gold and silver bars, the metallurgical feature of striking importance is the erection of the many smaller mills throughout the entire State. In many cases transportation facilities have not warranted the shipping of ore to custom works at any considerable distance and individual mines have built their own mills. Where the production of the

small mine would not justify the building of a mill, a number of small producers have contracted a certain small tonnage each to a custom mill to be installed near at hand. Thus 10-stamp mills, 20-stamp mills, 30-stamp mills have been put into commission or contracted for very widely.

The metallurgical progress of recent years has made this possible. The recent development in cyaniding silver minerals, the improved methods of getting the gold ore finely crushed and cheaply filtered have assisted in making possible the treatment on a small scale of the gold and silver ores of the State.

It will be impossible to give even a brief description of the mills put into commission and those being erected during 1910, and I shall attempt only to give some data regarding a few of them. In many instances the gold is coarse enough to be readily caught by the amalgam of the plates. Where the ore is in condition for a good extraction to be obtained by amalgamation alone, stamp milling with apron plates is all that is employed. In other cases the coarse gold only is amalgamated and the fine cyanided. Where the gold is partly in combination with sulphides present, concentrating machinery is also employed and two marketable products are obtained, bullion and concentrates. The concentrates are often shipped to the smelters, but may be profitably treated by special process at the mill. Cyaniding after fine crushing with long time and strong solution is effective in extracting the gold from the concentrates, and this process is resorted to at some of the mills. Then chlorination after roasting is employed with the concentrates at the Goldfield chlorination mill with excellent extraction.

The 60-stamp mill of the Tonopah-Belmont Company at Millers has operated upon 220 to 250 tons of ore per day.

The Tonopah-Liberty Mining Company has put into commission its mill at Liberty near Tonopah. The power for this plant is obtained from the Nevada-California Power Company. The all sliming process is employed and the mill is a model of modern construction. The ore passes through a Blake crusher, Chilian mill and tube to be reduced to slime, then a Trent mechanical agitator, combining power and air agitation, and finally to the Oliver filter. The mill has been in commission since Oct. 2 and is treating 130 tons per day.

The 30-stamp mill of the Tonopah Extension Company is treating 135 tons per day, obtaining concentrates to the amount of 40 tons and bullion valued at $45,000 each month.

The Bonnie Clare mill in the Gold Mountain district, Nye County, started up in September prepared to treat the company's ore as well as custom ore, stamp milling, amalgamation, concentration and cyaniding being employed.

The 10-stamp mill of the Nevada Reduction Company at Manhattan is a good type of the small mills recently erected and now building. After the stamps come the silver-coated copper amalgam plates, then the classifiers (in this mill the Dorr is used) giving sands for tube mill and slimes to join the product from the tube mill going to the filter (the Oliver is used here). The gold is precipitated by zinc shavings in the zinc box,

Small 10-stamp mills or mills of this capacity are to be found in operation or building at Rhyolite by the Tramp Consolidated; at Dayton, Lyon County; at Blackhorse by the Amalgamated Nevada Company; near Manhattan by the Manhattan Milling & Leasing Company; near Caliente by the Gold Chief Mining Company; in Esmeralda County by the Gold Bar Company, and at various other localities.

Larger mills ranging from 50 tons to 150 tons capacity are operating or building by a number of other companies, including the Nevada Hills Company, at Fairview, Churchill County; the Rich Hill Company, at Eldorado; the Rawhide Queen and Rawhide Coalition—the Queen mill is now running; the Austin-Manhattan Company, with its dry concentration process with the Huff electrostatic machine; the Rainbow-Amazon Company, at Jarbridge, Elks County, etc.

The Butters cyanide plant in Six-Mile Canyon, Storey County, is closed for the winter. During this time it will be put into first-class condition.

The Bamberger-Delmar 500-ton cyanide plant has been sold to the Great Western Machinery Company, of Denver, who are dismantling and disposing of the plant. The above are some of the mills that have been installed or undergone changes during the year, but it by no means embraces all that might be mentioned.

The Mason Valley Mines Company is erecting a smelter at Wabuska to treat the ores from the Nevada-Douglas and many other Nevada mines.

The mammoth Steptoe smelter at McGill, Nev., has been in full operation and has reached an output of 6,000,000 lb. of copper per month. The cost of smelting is kept down below that of most modern plants. The future of this plant along with the concentration mill of the company is most promising. The shipment of an entire trainload of copper bullion—25 cars of 50 tons each—from the Steptoe smelter to the Eastern refinery will impress one with the magnitude of the company's operations.

Gold and Silver.

Bullion Refining.—The details of electrolytic refining of cyanide precipitates at the Waihi Gold Mining Company, Waihi, New Zealand, are given in the *Aust. Min. Stand.* for Oct. 26, 1910, by J. G. GOODSMAN, who has been identified with the work in every department.

The zinc boxes are cleaned weekly at each mill, where the gold-silver alloy is prepared for subsequent treatment at the electrolytic refinery. The zinc slime is treated with 10 per cent sulphuric acid in wooden vats 9 ft. in diameter and 6 ft. deep. Agitation for one hour usually dissolves the zinc, after which the slime is washed with hot water, the solution decanted and the slime filtered by vacuum pump. The dry slime is then mixed with flux, consisting of 30 per cent of its weight of borax, tamped into iron trays and dried in an oven.

The custom of melting this material in plumbago pots, followed by cupellation of the metal, was abandoned some time ago, but will be adopted again on account of the inherent difficulties in the present system. At present the melting and cupelling are both carried on in a "test," but a great deal of trouble is experienced in getting mixtures which will withstand the action of the borate slag, and that of the litharge and copper oxide. The "tests" now used have a foundation of cemented firebrick and sides of cement 3 in. thick. The absorbent body is well tamped in and allowed to stand several weeks before using.

For use the "tests" are heated in a furnace under good blast and the charge is added with scoops. As soon as molten the "test" is tipped to allow the slag to run off. When all the slag has been run off the first lead is added, which amounts to 10 per cent of the weight of the dry slime. The cupellation is hastened by a blast of air through a tuyère, which drives the litharge over the breast of the "test." When the first litharge is almost removed, the retorted gold from amalgam plates is added, and a second charge of lead, 5 per cent by weight of the slime, making in all 15 per cent of lead added in cupellation. In the other system of melting in pots followed by cupellation only one charge of lead is used, amounting to 10 per cent of the weight of the bar bullion.

The slag from the melting operation and the litharge from the cupellation are reduced in a small blast furnace, and the resulting lead is either refined if rich in bullion or liquated and used again in cupellation. The gold-silver alloy is ladled from the "test" into molds 10 in. x 8 in. x ⅜ in., each bar weighing about 200 oz.

The electroylic plant has 30 cells, each having a capacity of 10 gal. of parting solution, 0.6 per cent nitric acid. In each cell are filter frames holding four of the bullion bars, with the anode terminal on the adjoining edges of the bars. The 4 sq. ft. of cathode surface is a ¼-in. silver slab fitting the bot-

tom of the bath and having a 4 in silver ribbon connection with the cathode terminal. The current used is 100 amp, or a current density of .25 amp per square foot of cathode surface. The voltage averages 0.5 volts, or 3.16 per cell. This figure varies with the temperature of the bath and with the decreasing distance between anode and cathode. The precipitated silver is scraped from under the filter cloths with wooden scrapers, washed to remove acid, melted in pots and cast in bars of 1000 oz., .998 fine. The gold and bullion scraps from the filters are washed through a 10-mesh sieve, the fine gold washed free from nitric acid and boiled for three hours with concentrated sulphuric acid. After another washing the gold is melted in pots and cast into bars of 1000 oz., .997 fine. The silver from the acid solutions is precipitated with salt and reduced with zinc, sulphuric acid being added to hasten the action.

Copper.

The Converter Flame.—The process of copper converting is indicated and controlled largely by the appearance of the flame issuing from the nose of the converter during the blow. The changes occurring are not as easily described as observed. An interesting article dealing with the successive stages of converting as indicated by the converter flame has been contributed to the Institution of Mining and Metallurgy by Mr. Donald M. Levy. The article is published in *Bulletin* No. 74 and is accompanied with reproductions of color photographs taken on Lumière plates, the exposures being made at night in the converter-room at the Washoe Smelter, Anaconda, Mont. The illustrations are excellent reproductions of the various flames, of which four are shown in the original paper.

The first shows the condition at the beginning of the blow, and is a rich red-brown flame due to the coal added, to the burning of iron and sulphur, and partly to the zinc, lead and arsenic in the fume. The flame continues red to red-purple for from two to eight minutes, after which the green flame appears and the slagging stage has properly begun. The green becomes more and more prominent, and after about 15 minutes is a vivid apple-green. This is shown in the second color plate, and is typical of the slagging stage.

The white-metal stage is indicated in the third color plate, where the green has disappeared and flashes of blue appear, resulting in a white-blue flame. At this stage the slag is poured off, and if the converter is too hot it is "doped" or charged with scrap copper.

The second part of the blow, from white metal to blistered copper, commences with a red-brown flame and smoke, but finally thins to a red-purple flame and continues thus to the end of the operation. This is shown in the fourth color plate. The end of the blow is difficult to judge, but is determined by the emission of small shots of hot copper, which no longer adhere to the hood over the converter, but rebound from it.

The green effect during the slagging stage is due to the formation of iron silicate, while the blue flame at the end of the slagging stage indicates that the iron has been slagged and that copper silicate is being formed. These colors are not produced unless the respective oxides are being slagged.

Following are observations made during a complete blow:

Converter A.—Matte charge.

Charged 2:25 p. m.—Flame red.
2:29 p. m.—Green appears.
3:00 p. m.—Green becomes brighter
3:17 p. m.—Vivid green.
3:18 p. m.—Skimmed; no metal reached.
3:21 p. m.—Blue appears in flame.
3:27 p. m.—Flame chiefly blue; converter turned down to skim. Doped

Converter B.—White-metal charge

Charged 4:13 p. m.—Doped.
4:14 p. m.—Converter turned up; flame red
4:16 p. m.—Flame red-purple.
4:18-5:20 p. m.—Flame peach to bronze-purple color

Ore Dressing.

Classification.—Metallurgists generally recognize the great service which Prof. Robert H. Richards has rendered to the industry of ore dressing. They will be glad to study his latest contribution on the development of hindered settling apparatus, which has been published in the November and December, 1910, issues of the *Western Chemist and Metallurgist.*

The causes of loss in concentration are first considered, and then methods of voiding the losses. The author find four ways in which mineral loss may occur: as including grains, as long or flat grains, as fine free mineral and as absolute slime. The treatment of flat, long or included grains will involve recrushing to a size which liberates the maximum quantity of mineral while making the least slime. Fine, free mineral in jig, table and vanner tailings offers the ripest field for improvement in milling, and the object of the present paper is to consider methods for saving this product. In regard to absolute slime, the author is still in doubt whether they can be saved by any means or not.

Free Settling.—The author then takes up four methods of making free settling tests to determine the ratios of quartz and galena which are equal settling. The methods are: (1) by letting clusters of grains settle together; (2) by letting individual grains settle one at a time; (3) by spitzlutte and microscope to get the ratio, and (4) by spitzlutte and photograph to get the ratio. The four methods give nearly the same results, viz., 3 to 3.5.

In the free-settling classifier the following things are essential: (1) means of feeding the pocket with the mixed sand to be treated; (2) rough sorting in the pocket, raising the finished light material by the rising water from the sorting column, and settling the heavy unfinished product to the sorting column; (3) finishing the sorting in the sorting column, allowing the heavy finished grains to settle down and out while the entangled grains are raised to the pocket again; (4) finally, admission of wash water into a pressure box, where the stream is divided, part rising and doing the sorting, the remainder passing out of the spigot with the discharged sand. The third point is most difficult to obtain, as there must be no downward currents to carry slime into the spigot, and the sorting column must have the same rising velocity in all parts of its horizontal cross-section. Classifiers are good or bad according as they do this one thing well or poorly.

What happens in a free-settling classifier is that, with the exception of the first spigot and last overflow, all the products are truly classified, i.e., they contain larger grains of quartz and smaller grains of mineral. The first spigot has an added increment of coarse grains of heavy minerals, and the last overflow has an added increment of fine material in addition to the truly classified grains. Hence, these products require special consideration in concentration.

Hindered Settling.—Under this head the author presents some new figures for the hindered settling ratios as determined in a new series of experiments. The earlier ratio of quartz and galena under hindered settling conditions was 5.85, but in the new series of figures it is 6.92. This figure is true for either continuous or pulsating current, but the latter has an advantage in avoiding the harmful downward currents. Hindered settling thus gives a wider range between the particles of quartz and galena, which are equal settling, than is possible with free settling, the ratios being about 3.5 for free settling and 6.9 for hindered.

The essential feature of getting hindered settling is a hindered settling chamber above with a constriction below, and the whole current of water rising through the two. The cause of the difference between hindered and free settling seems to be that with the latter both quartz and galena are settling in water with a specific gravity of 1, while with the former the grains are settling in a denser medium, probably specific gravity 1.5, due to the crowding of the mineral grains. The effect of hindered settling is twofold: (1) The slime is more per-

fectly kept out of the spigot product and (2) the advanced ratio helps the concentrating machines more than free settling.

The requirement for hindered settling is a proper ratio of area between the settling chamber above and the constriction below. The author has determined this to be 1 to 4, and believes that this will effect a condition of "full teeter" of the bed in the teeter or settling chamber and a discharge through the constriction below.

The author then takes up a consideration of jigs and shows that in the ordinary jig the act of pulsion is identical with that of hindered settling current. The return stroke, or suction, causes an intermission in this action, so that the grains of galena cannot continuously settle through the quartz under hindered settling conditions. In the pulsator jig, however, the conditions of hindered settling are maintained continuously and to this is ascribed the remarkably high capacity of this machine.

Five types of hindered settling classifiers are described. Just as the free settling classifier has an added increment in the first spigot and last overflow, so has the hindered settling classifier, with this distinction: In the latter the larger ratio between the diameters of the grains in juxtaposition is greater, and as a consequence the coarsest product has a larger proportion of all the values in it, and the water in the spigots and overflow are reduced. Fine free mineral grains likely to contaminate the tailings of the succeeding concentrating machines are more easily kept out of those tailings.

Finally, the author takes up various concentrating machines, such as the jig and table, explaining their faults and showing the benefits to be derived from classification or screening.

The original article is long, but will well repay study. Some of the data presented have been published, and numerous references are made to former papers by the author. In addition there is a great deal of hitherto unpublished work, illustrated with numerous cuts.

Ore Sampling.

The Philosophy of Sampling.—The correct sampling of ore is a necessary preliminary to good metallurgical work. In the *Canadian Mining Journal* for Dec. 1, 1910, JOSEPH T. MANDY presents some observations on this subject. Mechanical and hand sampling have had their supporters, and each side undoubtedly has some strong arguments to present, but the weight of argument favors the former method.

The author inclines to the belief that the whole matter of sampling resolves itself into a question of averages, and that in the long run the chance of error favoring either buyer or seller is rather remote. But since the accuracy obtainable varies inversely as the chances for error, he believes that such chances should be eliminated to the utmost. With each chance for error eliminated, the sampling becomes more accurate.

Mechanical Sampling.—The initial feed is of the greatest importance. It should be regular, and hence the author prefers hopper feed to hand-shovel feed. With the latter method the feed is like to be intermittent, and consequently the first cut of the sample may be irregular and incorrect. If hand-feed is adopted it would be necessary to have some intermittent hopper between the crushing apparatus and sampling machine. The same objection to intermittent feed applies to the subsequent treatment of the first cut, for no matter how continuous the original feed may be, it becomes an intermittent one after the first cut unless some device is installed to prevent it. The installation of rolls partly removes this objection, but not wholly, and it becomes apparent that some regulating device should be used after each cut of the sample. Another objection to mechanical sampling is that in the swing of the cutting device coarse particles of ore strike the edges and are likely to be thrown into the wrong compartment. In the ordinary Vezin machine the particles will always be thrown into the reject compartment, but with the Brunton stream diverter it is less likely to occur, while with the Brunton oscillating cutter the possibility of error has an opportunity to average. The greatest point in favor of the mechanical sampler is the elimination of the personal element, which is ever present in hand sampling.

Hand Sampling.—Here the personal equation is the greatest drawback, and no matter how conscientious the operator, his precautions against unfair or incorrect work are insufficient. This becomes true even with unskilled labor, for such operators soon learn the points of the process and eventually work incorrectly. Fractional shoveling, or taking every fifth or tenth shovelful of the ore, is a very poor method save for the poorest ores. Coning and quartering is an operation that is open to the strongest criticism on account of the uneven distribution of the coarse and fine ore in the cone and cake. This method of finally reducing the sample to small size is condemned by the author regardless of whether it is used in connection with preliminary mechanical sampling or hand sampling. A tabulation of figures is given showing the segregation of sizes of ore in a cone and in the resulting cake from which the quarters are rejected or saved.

The author concludes that in all fairness it should be acknowledged that mechanical sampling is superior to hand sampling, but that the latter is open to improvement in the points mentioned. In addition, it is urged that the cutting of the sample should be made mechanical clear through the operation, substituting riffles for the coning and quartering system.

New York Meeting of American Institute of Chemical Engineers.

The third annual meeting of the American Institute of Chemical Engineers was held at New York City from Dec. 7 to 10. The sessions were held alternately at the Hotel Astor, Columbia University and the Chemists' Club. The excursions, a full program of which was given in out last issue, had been carefully arranged so as to be of as manifold and actual interest as possible and were greatly enjoyed; the chain of excursions, distributed over several days, has become one of the most prominent features of the Institute meetings.

Dr. Charles F. McKenna's presidential address dealt in a very interesting way with the Evolution of Portland Cement Processes. We hope to publish the address in full in one of our next issues. An abstract of it and of the other papers presented will be found in the synopsis in this issue.

An especially interesting feature of the proceedings was the discussion on chemical engineering education. Dr. F. W. Frerichs presented the report of the committee on chemical engineering education. Dr. Fred W. Atkinson, president of the Polytechnic Institute of Brooklyn, presented a paper on the development of the chemist as an engineer. Prof. M. C. Whitaker, of Columbia University, presented a paper on the training of chemical engineers which meets the requirements of manufacturers. The general discussion of the subject extended over several sessions. Prof. Whitaker's paper is printed in full elsewhere in this issue.

The election of new officers of the Institute for the coming year gave the following results.

President—Dr. F. W. Frerichs, Herf & Frerichs Chemical Company, St. Louis, Mo.

Vice-Presidents—Mr. George P. Adamson, vice-president and general manager, Baker & Adamson Chemical Company, Easton, Pa.; Dr. Eugene Haanel, director of mines, Ottawa, Ont., Canada; Dr. Leo. H. Baekeland, Yonkers, N. Y.

Secretary—Prof. J. C. Olsen, Polytechnic Institute, Brooklyn, N. Y. Treasurer—Henry S Renaud, New York. Auditor—Herbert Hollick, Camden, N. J.

Dr. Samuel P. Sadtler and Dr. C. F. McKenna are the past presidents. The directors of the Institute are: H. F. Brown, Wilmington, Del.; Ludwig Reuter, Berkeley, Cal.; Thorn Smith, Detroit, Mich.; J. T. Baker, Phillipsburg, N. J.; Dr. W. M. Grosvenor, New York City; Richard K. Meade, Allentown, Pa.; William M. Booth, Syracuse, N. Y.; Dr. Edward G. Acheson, Niagara Falls, N. Y.; Prof. Edward Hart, Easton.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our London Correspondent.)

The Aëration of Portland Cement.

In a paper read before the Institution of Civil Engineers on Nov. 22 Mr. H. K. G. Bamber reviewed past and present methods of manufacture and advocated the abandonment of the practice of "weathering." He said that many engineers—especially those whose experience extends over a long period—have become so accustomed to the use of some aerating process that, perhaps by force of habit, they still continue to require it of their contractors, forgetting that cement which has already passed the stringent tests of the British standard specification has necessarily shown itself to be entirely innocent of any expansive tendencies. If this specification is adhered to for all requirements, the old aeration methods have become as obsolete as the ancient machinery of the cement industry, which has found its way to the scrap-heap. Contrasting the properties of cement made by the old processes with those of the best modern cements, it should not be difficult to convince engineers of the desirability of abandoning an obsolete method of treatment.

Mr. Bamber might have gone a step further and condemned the most undesirable practice of some cement testers in aerating "hot" samples before making the usual physical tests until the rise of temperature has been reduced very considerably. The writer recalls a case which came under his notice rather more than a year ago, in which an analytical chemist who adhered strictly to the British Engineering Standard Specification condemned a sample of cement as thoroughly unsound; another sample from the same source was sent to a certain professional cement tester, who repeatedly reported that the cement was "still too hot to test," and after aërating it for *more than three weeks*, certified that it was sound. The sequel was that the whole of the work executed with that cement had to be taken out and replaced. If a cement contain such quantity of free calcium oxide as will cause an abnormal rise of temperature on ganging, it cannot be satisfactory; and no amount of weathering or aeration will enable such stuff to compare with properly burnt cement free from excess of lime, because the resulting carbonate and hydrate—sometimes referred to as "hydrated carbonate"—must detract from the cohesive power of the cement.

The Detection of Fire-Damp.

At the Society of Arts, last week, Sir H. H. Cunynghame, in a paper on methods of detecting fire-damp in mines, described a simple addition to the ordinary safety lamp by means of which the "cap" which forms over the flame in the presence of methane or other combustible gas is rendered much more conspicuous. With an ordinary flame the cap is usually so pale that it can only be discerned when the lamp is turned down very low, when it was possible to see the cap in the presence of one and-a-half or two per cent of methane, and easy to observe it if the air contained three or four per cent

Professor Cadman and himself devised an improvement which made the cap more visible without unduly diminishing the size of the flame. A small piece of asbestos steeped in sodium carbonate solution and dried was so mounted that it could be brought into the flame of the miner's lamp; and the cap over the resulting bright yellow flame became quite distinct, and rendered the test much more sensitive He had found that sixty hours' continuous burning was required to exhaust the sodium carbonate in the asbestos; and by actual observation in mines had established the fact that one-half per cent of methane could be seen without any difficulty.

Detection of Inflammable Vapor or Gas.

In a paper recently read before the Institution of Engineers and Shipbuilders in Scotland Mr J H Heck described a method of testing air for petroleum vapors and combustible gases. The apparatus consists of a glass syringe, having a capacity of about four fluid ounces, and a small spirit lamp, giving a flame rather more than one inch in height. The syringe is filled with the air to be tested, and the nozzle is brought to a dull red heat in the spirit flame; then, holding the syringe horizontally, with the orifice close to and slightly below the tip of the flame, the air is expelled by a series of slight jerks of the piston.

An inflammable or explosive mixture ignites and burns at each puff with a bluish color for poor mixtures and a yellowish-blue for those richer. Weaker mixtures of air and vapor give a bright incandescent streak or cap; and air free from combustible gas or vapor produces no incandescent streaks, but only a slight flattening or shrinkage of the top of the flame.

This method does not appear to the writer to be wholly free from risk to the operator. Given a mixture having decided explosive properties, the preliminary heating of the nozzle might, in a bright light, be carried too far and cause ignition with possible disastrous results.

Non-Ferrous Electrometallurgy.

Professor Arnold, in his recent presidential address to the Electrometallurgical Society of Sheffield, said that the Sheffield University had arranged to devote £10,000 to the establishment of a department for research in non-ferrous metals. The University was already in a position to investigate the electrical smelting of brasses, bronzes and German silver, having acquired a Kjellin non-ferrous crucible, with which it was also intended to carry out tests to compare the relative efficiencies of electric melting and coke melting of base metal.

The Relation of Chemistry to Metallurgy.

Mr. A. C. Claudet read a paper on this subject at a meeting of the Society of Chemical Industry, held early in November. He said that the development of the composite science of metallurgy might have been much more rapid during the last thirty years—particularly in respect of some of the non-ferrous metals—if metallurgists had kept closer in touch with the advances in industrial chemistry. Both sciences were allied with mechanical engineering; and frequently the greatest difficulties were those encountered in mechanically working out a process.

As demonstrating the relation between chemical industry and metallurgy he instanced the concentration process employed at mines in the South of Spain on cupriferous pyrites; the treatment of Spanish cupreous iron pyrites, imported by England and other European countries, for extraction of sulphur and copper; and the treatment of lead-zinc-silver sulphide ores from Broken Hill. The manufacture of copper sulphate might at first sight seem to be purely a chemical process; but it involved various important metallurgical operations. The Pearce process for auriferous copper was another example.

The arsenical cobalt and nickel silver ores are now worked in Ontario, affording a promising field for chemical and metallurgical research. Their silver content was very large—often thousands of ounces to the ton; but their satisfactory treatment presented great difficulties, for the arsenic sometimes amounted to thirty per cent, and the cobalt and nickel to ten or fifteen per cent each, and a method of recovering all these as well as the silver was wanted.

Market Prices in November.

Copper rose to £57.16.3 on the 2d, remained steady until mid-month, and gradually declined to £57.5.0 on the 30th.

Tin opened at £167.10.0, dropped to £166 on the 9th; then recovered the same figure by the 23d, and rose to £169 at the month's end.

Lead, except for a slight decline on the 9th, kept at about £13.12.6 until the last week, when it declined to £13.11.3.

Hæmatite fell from 65/3 on the 2d to 64/- on the 16th, and improved to 64/6 on the 30th.

Scotch Pig opened at 55/10½, then fluctuated with a minimum of 55/4½, and closed at 55/9.

Cleveland Warrants from 49/10 fell to 49/5½ on the 9th, with but slight improvement until the end of the month, when the quotation was 49/9.

Alum, lamp, loose, per ton	£5.12. 6.
Antimony, black sulphide, powder, per ton	22. 0. 0.
Borax, British refined, crystal, per ton	16. 0. 0.
Sulphate of ammonia, f.o.b. Liverpool, per ton	12.16. 3.
Sal ammoniac, lamp, firsts, delivered U. K., per ton.	42. 0. 0.
Copper sulphate, per ton	19.10. 0.
Caustic soda, ash 48 per cent ordinary per ton	5.10. 0.
Litharge, flake, per ton	15.15. 0
Bleaching powder, 35 per cent, per ton	4. 5. 0.
Sulphur, recovered, per ton	5. 5. 0.
Camphor, 1 oz. tablets	1. 7.
Shellac, standard T. N. Orange spots, per cwt	4. 3. 0.
Hydrochloric acid, per cwt	5. 0.
Copper ore, 10 to 25 per cent, unit	9/4½ to 10.4½.
Tin ore, 70 per cent, per ton	£104 to 106. 0. 0.
Carbolic acid, liquid, 97/99 per cent, per gal	1. 1. 1.
Creosote, ordinary good liquid, per gal	2.
Naphtha, solvent, 90 per cent at 160 deg. C., per gal.	10.
Mica, small slab, original cases, per lb	6d to 1. 0.

India rubber: Paraffine, at the middle of the month prices had fully recovered from the depression at the end of October; and during the last week there has been an advance of 2d. to 3d. per lb.; prices being now steady at 6/3.

Platinum has risen to £8 per oz.

Synopsis of Chemical and Metallurgical Literature

Chemical Engineering.

Evolution of Portland Cement Processes.—In his presidential address to the American Institute of Chemical Engineers Dr. CHAS. F. MCKENNA gave a rapid review of the history of Portland cement from the day of Aspdin's invention, portraying particularly the different stages in the process from the crude method of Aspdin to the modern process of burning in rotary kilns and the fine pulverizing. The economy of the present day is such that for 1 cent 12 lb. of raw material can be taken from the earth, dried, ground, calcined and vitrified and the product ground again. Considerable attention was paid to the old forms of wet processes, and the statement was made that present-day European practice in grinding to semi-wet slurry hard materials, not marls, is the most important advance in modern times and should be taken up more vigorously in America. The basis for this is the claim that the dryer expense is saved, finer incorporation is obtained, better clinkering and easier grinding of the clinker. The critical feature of the modern semi-wet process is to secure the maximum of fineness of slurry with the minimum water. It was said that in good practice some materials can make with less than 35 per cent of water a slurry which can be pumped, and as it dries in the upper end of the long kiln the extra coal consumption is not marked. Modern rotary kiln practice, particularly the long kiln, was touched upon, and the use of preheated air from clinker kilns and diminished indraft of cold air.

Hydrated Lime.—The manufacture of hydrated lime was discussed in a paper of Mr. RICHARD K. MEADE before the American Institute of Chemical Engineers. Hydrated lime is mechanically slaked lime. Just the correct portion of water is added so that the resulting product is a dry fluffy powder. Most of the hydrated limes on the market up to a few years ago were very crude products, usually containing considerable carbonate. Now, however, processes have been perfected which produce a lime consisting almost entirely of the hydrate. The process for the manufacture of hydrated lime usually consists of three stages, (1) crushing the lime, (2) mixing the lime and water, (3) separating the hydrate from the unhydrated portion. In some processes instead of separating the hydrate from the unhydrated portion the two are ground together in some form of pulverizing mill.

The hydrating plant itself is usually located adjacent to the limekilns. The lime is drawn from the kilns and is immediately crushed in some form of rotary or pot crusher. From this it is conveyed mechanically to the hydrator, in which the slaking is done. There are several forms of this latter in use, but all come under two heads, continuous hydrators and batch hydrators. The continuous hydrator most used is the Kritzer hydrator, which consists of a number of cylinders mounted one above the other on a steel framework. Through the middle of each cylinder runs a shaft to which are fastened paddles. The lime is fed in at the upper cylinder and mixed with the water. The paddles stir it up and also force it to travel through the cylinder. It then drops into the next cylinder and here it is mixed up and passed on by the paddles in this and so on until it has passed all of the cylinders. The proportions of lime and water are determined by means of an automatic feed for the lime and a valve for the water.

The Clyde hydrator is a batch hydrator in which the lime and water are weighed in separate charges consisting of about 1500 lb. of lime and the proper amount of water for this much lime. The pan rests upon a ball bearing and is revolved by means of a gear and pinion on the under side of the pan, which is surmounted by a hood and a stack. Plows running across the surface of the pan are fastened to a stationary framework above the latter. The lime and water are fed into the hydrator through an opening in the side, and the plows stir it up and mix the two. When hydration is completed the lime is discharged through an opening in the center of the hydrator, which during the mixing is closed.

After passing through either hydrator that portion of the lime which has been slaked is in the form of a very fine fluffy powder all of which will pass through a very fine sieve. Mixed in with this, however, are always some unslaked lime, unburned limestone and silicious cores. For the better grades of hydrate it is usual to separate out this foreign material by means of screens, although where the amount of these constituents is small they are sometimes ground in with the hydrate. Fuller-Lehigh and Raymond mills are principally used for the grinding, while the separators most used are the Newaygo and the Columbian.

The packing of the hydrate is best done in cloth (100 lb.) or paper (40 lb.) bags automatically by means of the Bates system, which consists of a valve bag and machine for filling the same. The bags are provided with a valve and are tied before being filled. This valve is made by ripping, folding and sewing one corner of the bag. The cement is inserted through the valve by means of a tube on the bagging machine, which automatically fills and weighs the bag. The machinery for hydrating magnesian lime having a capacity of two tons or three tons per hour costs about $5,000 and for high calcium limes $12,000. The cost of hydrating lime is about 25 cents to 30 cents per ton. One ton of lime, however, will often make more than one ton of hydrate.

The advantages of hydrated lime are: It is a fixed product and does not deteriorate with age; it can be handled without risk of fire; it may be shipped by water and packed in cloth or paper bags; it can be used for all purposes for which lime is used and is more convenient than the latter. In addition to this it can be used for mixing with cement, for which purpose quicklime cannot be used.

Bleaching of Oil with Fuller's Earth.—This subject was discussed at considerable length in a paper before the American Institute of Chemical Engineers on Dec. 8 by Mr. DAVID WESSON. The literature of Fuller's earth and its application is very meager. It appears to be first mentioned about 1889. From the best information obtainable Fuller's earth was first used for bleaching fats and oils in Chicago on a large scale

about 1879, or 1880, although it is said to have been used for a long time previous in northern Africa for clarification of olive oil. The first application was very cumbrous, the earth being stirred into the hot oil and allowed to settle, after which the oil was decanted and the residue boiled with water and pressed. The filter press was first used between 1880 and 1882. The efficiency of a given sample of Fuller's earth cannot be readily determined by chemical analysis. The best method is by making a bleach test on a small scale. In comparing the bleaching properties of two earths they should always be passed through sieves of the same size in order to obtain an accurate comparison. Fuller's earth doubtless owes some of its bleaching properties to the presence of a certain amount of water of constitution. This, if driven off by either drying of the earth or by adding it to oil of too high a temperature, will prevent bleaching. The reason why Fuller's earth seems to bleach is not positively known. Fuller's earths differ greatly in their absorptive properties and show great differences in their liability to spontaneous combustion. The arranging of filter plants is a matter of considerable importance. The best place for filters is on the top of a refinery under suitable shelter. By having them at an elevation above the bleach tanks oil can be allowed to run back to the same, or can be fed to any part of the building, or run easily to storage by gravity; besides, the fire risk is much less, and when the filters are steamed out the nuisance of the steaming is entirely done away with. The mixing kettles for oil and Fuller's earth are practically cylinders provided with mechanical agitation and suitable heating coils. Filter pumps should be slow moving, capable of working up to a pressure of 150 lb. to 200 lb. The operation of the filter plant is quite simple. The earth is mixed with the oil at the proper temperature and then pumped through the filter press, the first portions of the oil being allowed to run back to the mixing kettle, or bleacher, as it is often called, until it comes up to the required standard, after which it runs directly to storage.

Fitzgibbon Boiler.—In a paper read before the American Institute of Chemical Engineers Mr. Jerome Alexander gave a brief description of the construction and operating principles of the Fitzgibbon boiler, a type combining vertical with horizontal tubular features. The boiler is internally fired, and is entirely without external or internal brickwork, thus avoiding the annoyance and expense of repairs at inopportune times. A heavy asbestos or magnesia covering reduces radiation below that found in ordinary bricked-in boilers. The boiler is a quick steamer, economical in coal consumption, and can be easily moved about.

Sewage Disposal.

The problem of sewage disposal was the subject of a symposium of papers presented at a joint meeting of the New York Section of the American Chemical Society and of the American Institute of Chemical Engineers at the Chemists' Club in New York City on Dec. 9.

Basic Principles.—Mr. Geo. C. Whipple pointed out in his paper that the basic principles underlying all methods of sewage disposal are to get rid of sewage without danger to the public health, with the least possible nuisance to the smallest number of people, with the least damage to property, and at the smallest cost. The process of purification is usually carried on in several stages: First, a partial removal of the suspended matter by screening or sedimentation, or both; second, a process of oxidation using sprinkling filters, contact beds or sand filtration, and, third, a final removal of bacteria by sand filtration or disinfection. The term "sewage purification" is popularly applied to any or all of these processes. This has contributed not a little to confusion of ideas. Laymen innocently suppose that when sewage is "purified" it becomes pure, whereas the sanitary engineer may mean only that it is purer than it was before. How much purer depends upon the method used. It would be of decided benefit to the question of sanitation if more definite terms were used, such as sewage clarification, oxidation, deodorization, disinfection, filtration, etc.

The chief danger from sewage is the contamination of public water supplies. For the protection of public water supplies it is not safe to depend upon sewage purification works alone unless more than ordinary precautions are taken, for the reason that most of the processes cannot be depended upon to remove bacteria from the sewage to a sufficient extent, while with the combined system in use complete purification is impossible. Sewage purification is of advantage, as it offers a second line of defense, an increased factor of safety. It also serves to prevent nuisances to sight and smell, which are of real damage to property and affect personal comfort. Sewage works themselves, however, may be the cause of similar nuisance.

What appears to be needed in this country at the present time is some method of co-operation by which needed sanitary reforms can be brought about at least expense. It is unbusinesslike to compel the purification of the sewage of a large upstream city in order to protect the water supply of a small city lower down, provided pure water can be furnished the latter in some better and cheaper way. Legislation that clothes the State authorities with power to prevent the pollution of sewage, but does not give them power to compel the purification of water or to control pollution by trade wastes, is unfortunate. It naturally leads to litigation rather than co-operation and may retard rather than hasten necessary sanitary reforms. If our State authorities cannot be trusted in this matter it may be that a proper solution of the difficulty will be found in the establishment of district boards similar to those in England and Germany, such boards having jurisdiction over the limits of particular watersheds. In some respects these natural hydrographic boundaries have advantages over artificial State boundaries. In the near future also our national government will take a hand in the matter. In whatever form the authority may be constituted the idea of co-operation should prevail, and ironclad rules against stream pollution should give way to a rational distribution of the burden of purification of both water and sewage, and an equitable adjustment of cost made between the parties interested, thus decreasing the total expense of sanitary measures required and utilizing natural resources for the purification of sewage in water as far as this is safe.

If the system of water carriage of sewage continues in use the time will some day come when the sewage of all of our cities will be purified, partially or completely, and all surface water supplies filtered. It is proper to anticipate this consummation as far as our means permit, but meantime it is good business and sound common sense to spend our money first where it will go furthest and do the most good, building water filters and sewage purification works, sometimes one, sometimes both, as they may be needed. Adequate remedies against stream pollution from the standpoint of nuisance have been usually obtained by an appeal to the principles of common law. Cases involving bacterial pollution by sewage have been thus far too few to establish definite precedents. It will be interesting to see whether, in view of our increasing population, and especially the increasing growth of our cities, the courts will ultimately decide that the use of unfiltered river water as a source of water supply by riparian owners is a reasonable use of the water. The speaker was optimistic in regard to the improvements that are being made in methods of sewage disposal. He believes that the near future is to see extensive developments in the art. Important as are the recent improvements in the technical details, still more important and necessary is a rational assignment of the means employed to the work at hand. The use that is to be made of the water into which sewage is discharged and the conditions tending to natural purification are to be studied with greater minuteness than ever before; the method of dilution is to be used more

rationally, and the designs of the sewage disposal plants made to fit more closely the particular conditions. The general result will be the construction of more and simpler plants for the treatment of sewage and, where occasion makes necessary, the installation of yet more efficient devices for bacterial purification than those now employed.

Sewage Disposal in Europe.—Dr. RUDOLPH HERING gave in his paper a review of European practice. Disposal of sewage by dilution was first practised. Of the 16 largest cities in the world having a population of over 1,000,000 inhabitants every one without exception now employs the dilution method where flowing water is available and a sufficient quantity of oxygen is dissolved in it to represent a flow of fresh water equivalent to at least 3 cu. ft. per second for every 1000 persons discharging sewage. Where a water supply is taken from a river it must be purified even after any sewage entering above has been treated. Where a river is not used as a water supply sewage may contaminate the food fish, oysters, etc. The commercial value of such food is insignificant compared with the cost of removing all the conditions that would impair their hygienic qualities. It is therefore much cheaper to take them at a greater distance from the large cities where no dangers exist.

The nuisance to sight produced by sewage is almost everywhere in Europe prevented by keeping all plainly visible floating matter out of a river by screening. Screens with a mesh as small as 1 mm are used. The nuisance to smell is prevented by constructing sewers with such smooth interior surface and good grade as will deliver the sewage practically fresh to the outfall and insure an immediate submergence of the sewage as far below the surface of the water as possible. This method is entirely successful in many Continental cities. The amount of sewage discharged into the Thames is so regulated that the dissolved oxygen in the water does not generally fall below 30 per cent of saturation. No nuisance results under these conditions. European experience justifies the conclusion that purification of sewage by the oxygen in flowing water is entirely satisfactory.

Sand purification of sewage requires the delivery of the sewage in fresh condition and a preliminary separation of the coarser suspended particles, after which the finely divided matter may be oxidized in intermittent sand filters. The more recent developments involve improvements on the septic tank method of purification. Dr. Imhoff has developed a tank having two compartments. In the upper the suspended matter settles out and is drawn off through a slot. In the lower compartment the sewage is decomposed with the evolution of methane, carbon monoxide, and dioxide. No sulphuretted hydrogen is produced. The decomposition is complete in from three to six months. The sludge which is removed dries quickly to a friable material similar to garden mold. The annoying question of the disposal of sewage sludge may therefore be considered satisfactorily settled.

Sewage Disposal in New York.—This was discussed in a paper by Dr. GEO. A. SOPER. For the most part the sewage of New York is conveyed in crude condition by the shortest route to tide water and discharged at or near the surface of the water. There are over 500 sewer outlets discharging 600,000,000 gal. of sewage per day. In all parts of the harbor above the Narrows the water is seriously polluted and rendered objectionable both to sight and smell. Along 175 miles of shore the water is so polluted as to be dangerous to bathe in. These conditions will rapidly become worse as the population increases. A standard of purity for the water of New York harbor has been proposed by Colonel Black and Professor Phelps as a result of studies, estimate and apportionment. They consider that the dissolved oxygen in the water should not fall below 70 per cent of what would be present if the water did not receive any sewage. This limit has already been exceeded. A definite plan and program of main drainage, including outfall works and purification plants designed to improve the present unsanitary conditions, have been proposed by the Metropolitan Sewage Commission. The duty of carrying out this plan so far as New York is concerned has been intrusted to this commission. This work is now progressing satisfactorily.

Unsolved Problems.—Prof. C. E. A. WINSLOW discussed in his paper unsolved problems of sewage disposal. The three main ends of sewage treatment—the elimination of suspended solids, the oxidation of organic matter, and the destruction of pathogenic bacteria—can all be attained by practically efficient methods, but there are minor difficulties to be overcome in each case. The pathogenic bacteria can be destroyed when necessary by disinfection with bleaching powder, but the dosing devices available for use in small plants are very defective. The oxidation of organic matter can be economically effected by the use of sand, contact or trickling beds. Distribution systems for trickling filters require further study. The intermittent sand filter has possibilities of intensive action not attained in plants operated at present largely at haphazard. It is the accumulation of suspended solids on the surface which keeps the capacity of sand beds down to 100,000 gal. per acre per day and experiments carried out at the Sewage Experiment Station of the Massachusetts Institute of Technology show that the rate can be raised to 200,000 gal. or 300,000 gal. if suspended solids are removed by careful preliminary treatment. Sludge disposal is still the most serious of the unsolved problems of sewage treatment. It has been minimized at the Technology Experiment Station by the use of a deep tank on the septic principle, but so operated that the sewage rises and washes the sludge continually, and thus keeps it from becoming overseptic. By this means the liquefying process is so accelerated that over 70 per cent of the deposited total suspended solids have been liquefied. The utilization of sewage sludge, though not yet economically realized, furnishes promising possibilities to the industrial chemist in the future.

Mr. NICHOLAS S. HILL, JR., presented a paper on sanitary conditions in their relation to water supplies in the vicinity of New York City, touching upon the principal legal questions involved in the problem.

Recent Metallurgical and Chemical Patents

Gold and Silver.

Clancy Cyanide Process.—The details of the improvements which have been made in the cyanide process by JOHN COLLINS CLANCY were given in our issue for November, 1910, page 623. They are again mentioned here in order that those desiring to secure copies of the United States patents may do so. (976,043-4-5, Nov. 15, 1910.)

Zinc.

Production of Zinc Powder.—In the process of first making zinc oxide and then reducing the oxide in a separate process, OLIVER B. DAWSON, of El Paso, Tex., proposes to introduce into the zinc vapor, at a point just above or close to the bed of the mixture under treatment, sprays of water gas. The object of this is to supply carbon monoxide to combine with oxygen present, either free or combined with zinc, forming carbon dioxide, and leaving metallic zinc in the form of fume which is subsequently condensed. (976,557, Nov. 22, 1910.)

Rare Metals.

Pyrophoric Mass.—An improved invention relating to those substances which give off sparks of great intensity when rubbed with a metallic substance has been patented by DR. CARL AUER VON WELSBACH, of Vienna. These substances are of value in igniting liquids, solid and gaseous bodies, as well as for the purposes of illumination, signaling and photography. The inventor has discovered that cerium, or metals of the cerium group, may be treated to form a pyrophoric body which is hard and durable, resisting the decomposing action of air.

Since the pyrophoric property of the metal seems to lie in its suboxides or nitrides, the procedure is as follows: The metal is first reduced to fine powder, then compressed into suitable molds by a pressure of from 1000 kilos to 25,000 kilos per square centimeter. This mass is slightly pyrophoric, but the property is further increased by heating with an equal volume of air in a metal casing having thin walls. The temperature is raised gradually to incipient red heat, when a lively reaction ensues and the mass frits together. The fritted mass is stable and is so highly pyrophoric that when a knife-blade is drawn across it an actual sheet of flame is produced. For varying the hardness of the alloy certain other metals such as zinc, tin, lead, calcium, magnesium and aluminium may be added. (976,760, Nov. 22, 1910.)

Melting Furnaces.

Crucible Furnace Cover.—Furnaces adapted to the melting of metals in crucibles are not always provided with covers which prevent the flame of the furnace coming in contact with the metal, thereby oxidizing it and causing loss. It is the object of a recent invention by Julian L. R. Brown, of Bridgeport, Conn., to provide a cover for a furnace, which shall at the same time act as a cover for the crucible and prevent the deterioration of the metal by the action of the flame. The device will be understood by reference to Fig. 1, which shows a plan of the cover and a section of the furnace. The cover may be made of any suitable material, such as fire-clay, graphite or brick. The essential features are the central portion adapted to cover the crucible, and escape holes for the flame, in alignment with the flame gap surrounding the crucible. Various modifications of the idea are also patented. (975,506, Nov. 15, 1910.)

Charging Melting Furnaces.—In practice there is considerable wear on melting furnaces, and especially on electric furnaces, by throwing in the rough scrap and metal, which digs into the sides of the furnaces and weakens them. To obviate this difficulty Michael R. Conley, of New York has patented a cartridge of sheet metal adapted to fit the melting pot of the furnace and contain the charge to be melted. The entire surface of the cartridge comes in contact with the

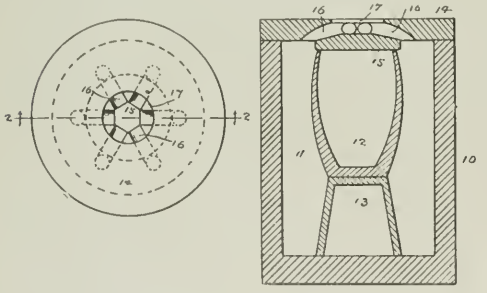


FIG. 1.—CRUCIBLE FURNACE WITH COVER

hot furnace wall and the charge begins to melt evenly and quickly. In practice the cartridge may be made of any desired composition, and thus not disturb the nature of the melted product. (975,751, Nov. 15, 1910.)

Reversible Heating Furnaces.

Furnace Head for Gas Furnace.—Relating to furnaces in which gas and air are simultaneously introduced and in which the stream of gas and air is periodically reversed, Bruno Versen, of Dortmund, Germany, has patented an improved furnace head, illustrated in Fig. 2. An object of the invention is to improve the caloric efficiency of the furnace by providing apertures in the furnace head for the escape of the "heat," i. e., the combustion mixture of air and gas, said apertures being at about the same height as the apertures for the introduction of the gas. By this means the air which served for raising the temperature of the heating flame is maintained within the said flame in the entire interior space of the furnace.

In the improved head air is conveyed by vertical lateral flues L which open into the top of the furnace by a common port L'. Gas is allowed to enter by a flue between the air flue and by a port below the common air port. Below the latter branch flues Z extend from the lateral flues L and open into the furnace as the gas flue G. When the head is acting as an exhaust, the heat on the hearth of the furnace will be conveyed through Z. When the head serves as an intake the blast of

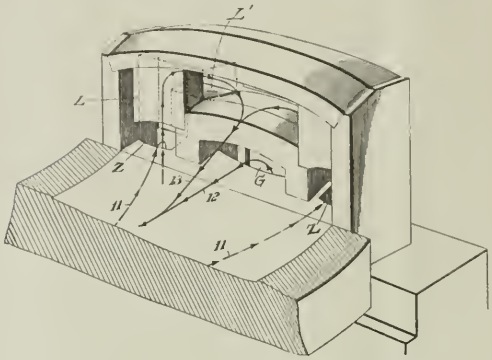


FIG. 2.—FURNACE HEAD FOR GAS FURNACE.

air will come in through L, and owing to its momentum will pass Z and enter through L'. For the purpose of protecting the furnace head against undue deterioration the inventor places cooling pipes in various portions of the head, not shown in this illustration. The system of piping also serves as a reinforcement, resisting vibrations and jars. On account of the advantageous distribution of the air the weight of the head may be decreased, so that even in very large furnaces the head may be easily removed. (975,738, Nov. 15, 1910.)

Roasting Furnaces.

Roasting Sulphides or Sulphates.—In a furnace of the superposed hearth type, Utley Wedge, of Ardmore, Pa., has introduced a modification designed to facilitate the elimination of sulphur from ore, particularly in the latter part of the roast. In the drawing, Fig. 3, is shown a furnace of the type mentioned, with the addition under consideration. The furnace is divided into two sections, upper and lower. In the upper section the roasting operation proceeds as usual, but on coming to the lower section the ore is fed through a seal formed by a shelf below the passage from upper to lower sections. The ore is scraped from the shelf by an arm attached to the upper rabble of the lower section. Between the arch of the upper chamber of the lower section and the hearth of the lower chamber of the upper section is provided means for introducing a carbonaceous reducing agent such as coke breeze or pulverized coal. This will reduce any sulphate compounds which may have been formed and will permit the elimination of the remaining sulphur by the application of a temperature approximating 1200° F., as compared with 1600° F. or 1800° F. commonly required in the lower portion of the furnace as usually operated. Arrangement is made to permit of either mixing the gases from both sections, or conveying them from the furnace separately. By preference the lower section of the furnace is of the muffle type, the heating chambers being supplied with the products of combustion from a heating furnace shown at the side. (976,525, Nov. 22, 1910.)

Roasting Furnace.—An improvement in his well-known roasting furnace has been patented by John B. F. Herreshoff, of New York. The object is to improve the life and efficiency

of the apparatus by proper cooling arrangement. The hollow shaft of the furnace is closed at its upper end and has a surrounding sleeve and rabbles attached to the shaft. Connection is made between the rabbles and the shaft and sleeve, whereby air may travel upward through the shaft, thence outward into each rabble arm and back into the sleeve, which is open at the top. By this circulation of air, which is driven by a blower, the hot parts of the furnace are cooled. (976,175, Nov. 22, 1910.)

Annular Roasting Kiln.—A furnace of this type with a number of novel improvements has been patented by JOHN

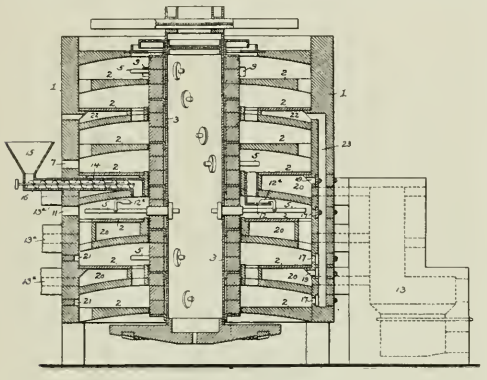


FIG. 3.—ROASTING FURNACE.

ZELLWEGER, of St. Louis, Mo. The principal features are means for recuperating heat from the fire gases; the use of warm air for oxidizing the charge; separation of sulphur gases from the products of combustion; and variation of temperature in the several zones of the hearth, whereby the ore may be locally heated to different degrees of temperature. (976,769. Nov. 22, 1910.)

On account of limitations of space a very large portion of Synopsis and Patent Abstracts had to be reserved for our February issue.

Vacuum Evaporator.

Evaporators are employed in many industries, often in the recovery of valuable constituents of solutions ordinarily considered worthless; frequently, too, unutilized exhaust steam occurs. In such cases a multiple-effect evaporator operated by this exhaust can be utilized to the greatest advantage. As an instance, it is only within the past few years that the tank-water which occurs in packing houses, etc., has been evaporated and its valuable solids recovered.

This tank-water averages from 5 per cent to 7 per cent solids; the smallest packing and rendering plants produce several hundred gallons daily and the value of the solids contained averages $35 per ton, whereas the cost of producing them, by means of the multiple-effect evaporator, will not exceed $10 per ton. An evaporator should, therefore, pay for itself in from six to eight months, depending upon the quantity and quality of the water.

The following notes refer to the construction and uses of the Redfield "standard evaporator," built by Messrs. Robert S. Redfield & Company, 90 West Street, New York City, for the evaporation of tank-water, glue, gelatine, garbage liquors, spent lyes, caustic potash, beef extract, cane and beet sugar, glucose, tanning extract, brine copperas, and many other special liquors occurring in chemical works and other establishments.

The Redfield standard evaporators are of the vertical tube type with circular steel shells or square cast-iron shells, both types being fitted with removable heat chambers, built with copper, steel or charcoal-iron tubes, as necessity requires. The entire heating chamber is removable en bloc from the body of the evaporator.

The shells are fitted with manhole and side glass fixtures, affording at all times an easy access to and a clear view of the interior of the shells.

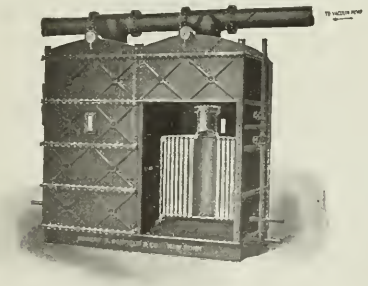
The fundamental distinction between the Redfield standard and other types of evaporator lies in the fact that in the Redfield apparatus the steam is outside instead of inside the tubes. The liquor therefore is afforded a free circulation upward through the tubes and returns to their base by way of the open space between the shell and the steam drum proper.

When the steam is caused to enter the steam chamber the heated liquors at once rise in the vertical tubes. There is nothing to impede this movement and with increasing temperature and higher vacuum the movement of the liquor becomes so rapid that jets will be thrown up often to a height of 2 ft. or 3 ft. It is obvious that this hot liquor must in some manner return to the base of the evaporator, which is cooler than the tube passages from which it emerged. This passage is provided by the space between the steam drum and shell, with the result that there is a constant and exceedingly rapid and automatic circulation of the water rising through the tubes, thence around the heating chamber to the base of the tubes.

By virtue of this vertical-tube construction the tubes automatically clean themselves due to the extremely rapid passage of the liquor throughout their length while the high speed at which the liquor circulates entirely prevents incrustation, resulting in freedom from tube repairs and saving in steam.

In the latest type of the Redfield standard evaporator the steam in each effect enters the center of the steam chamber. This construction at once insures a perfectly even steam distribution, which in part accounts for the remarkable reserve capacity. The condensation being automatically and immediately removed the full efficiency of the steam is constant. For this reason this type of evaporator is particularly well adapted for such liquors (like glue and gelatine) where quick evaporation is required, as too long contact with hot surfaces would deteriorate the quality of the product.

The tubes in the steam chamber are expanded into the tube



EVAPORATOR.

sheets by the same method that boiler tubes are expanded into the tube sheets of high-class boilers.

If steel tubes are employed they have a practically unlimited life. Copper tubes, owing to the great expense, are never used heavier than 16-gage, and these light tubes cannot be kept tight for any great period of time nor can they stand up as would a heavier tube. Practically the only other maintenance expense is on the pumps.

The fact that the steam chambers in the Redfield standard evaporators are entirely separate from the main body permits the removal of the entire steam chambers and the replacement of steam chambers with from 50 per cent to 100 per cent increased capacity.

These evaporators are virtually automatic in operation. The feed valves may be so regulated that a constant supply of fresh tank water is drawn in to counterbalance the rapid evaporation. Each plant is provided with a stick testing funnel so constructed that a quantity of the concentrated liquor may be withdrawn from the last effect quickly and easily to be tested for density. Opening a valve at the base of this funnel allows the vacuum to pull the tested liquor back into the evaporator. This method prevents all loss of concentrated liquor and is extremely simple.

An entire evaporating plant, either single or multiple effect,

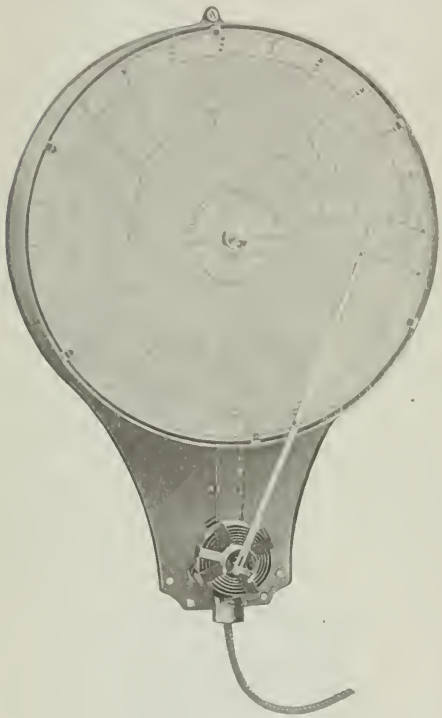


FIG. 1.—INTERIOR CONSTRUCTION OF RECORDING THERMOMETER.

may be easily handled by ordinary labor. In plants up to 2000 gal. only a part of one man's time is required.

To save floor space in multiple-effect evaporators all space may be eliminated between different effects The length of the large vapor piping as well as all other piping connected with a multiple-effect plant of this type is reduced to a minimum.

A double-effect Redfield standard evaporator is shown in the adjoining diagram. Messrs. Robert S. Redfield & Company build their standard evaporators in any capacity from 100 gal. to 5000 gal. per hour.

High-Pressure Tanks.—For containing gases or liquids under pressure cylinders that are merely riveted and calked have often proved unsatisfactory; such a cylinder may even be perfectly tight under a high-pressure water or liquid test, yet may be found to leak badly when only a slight air or gas pressure is applied. For this reason brazed tanks or cylinders are now generally preferred for high-pressure tanks. With a brazed tank it is well known that the brazed joints or seams are the strongest parts, as they form a perfect union when properly brazed. Concerning such brazed high-pressure tanks interesting information may be found in the catalog of the manufacturers, the Wm. B. Scaife & Sons Co., of Pittsburgh.

Compensated Gas-Filled Recording Thermometers

During the last 15 years recording thermometers of the Bristol Company, of Waterbury, Conn., have been constructed in various different forms, depending for their operation on the expansion of a liquid, the expansion of the vapor of a liquid, or the expansion of a gas. These thermometers have been used very extensively for ranges of temperature up to 800° Fahr., but the models equipped with flexible connecting tubes between the sensitive bulb and recording instrument and depending for their operation on the expansion of a vapor or a gas have not until recently been adapted for recording the lower ranges of temperature.

A new compensated gas-filled Bristol recording thermometer has recently been developed for recording the lower ranges of temperatures, such as atmospheric temperatures, temperatures of water, temperatures of brine in refrigeration systems, etc., and found satisfactory in numerous tests. These thermometers are equipped with a patented compensating device which automatically corrects for changes of temperature at the recording instrument.

Bristol's gas-filled class III recording thermometers are equipped with sensitive bulb and flexible capillary connecting tube and a pressure tube, as shown in Fig. 1, the sensitive bulb and flexible connecting tube and spiral pressure tube all being filled with an inert gas under pressure. Changes of temperature at the sensitive bulb cause corresponding changes in the pressure of the confined gas and these changes in pressure are measured and recorded by the recording instrument. The sensitive bulb is usually about 10 in. long and ¾ in. in diameter and the volume of gas contained in this sensitive bulb is very large in proportion to the volume of gas contained in the fine capillary connecting tube between the sensitive bulb and the recording instrument, this making the error due to changes of temperature along the connecting tube negligible.

Scientists have long recognized the fact that an air or gas

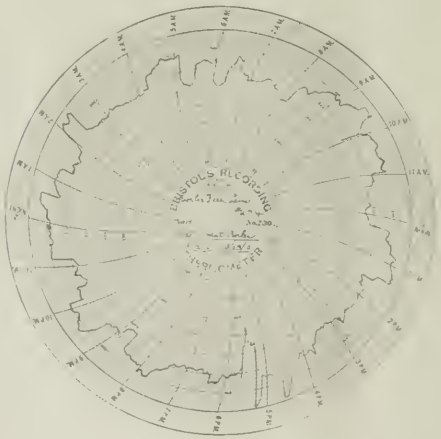


FIG. 2.—RECORD OF FEED WATER TEMPERATURE.

thermometer is the ideal type of apparatus for measuring low temperatures, provided that the air thermometer can be manipulated in the laboratory, where it is possible to make various corrections. The very important new feature of the Bristol compensated gas-filled recording thermometer is the patented compensating attachment for the spiral pressure tube, shown in Fig. 1, since a thermometer equipped with this compensator gives the same readings or record when the temperature at the recording instrument changes as it would if the temperature at the recording instrument remained constant.

The need for such a compensator can be illustrated by the

application of a recording thermometer for recording temperature of brine in a refrigeration system. The temperature of the atmosphere at the point where the recording instrument was installed might change, although the temperature of the brine at the point where the sensitive bulb was installed remained constant and a recording thermometer for brine temperature should, of course, be so constructed that it would be affected only by changes of temperature at the sensitive bulb.

Fig. 2 shows a record of feed water temperature as recorded by one of these instruments, the chart having range of 40° Fahr. to 225° Fahr. It will be noted that the scale of this chart has uniform graduations over the entire range and this makes it possible to use the Bristol-Durand radii averaging instrument for computing the average of the feed water temperature for the 24 hours.

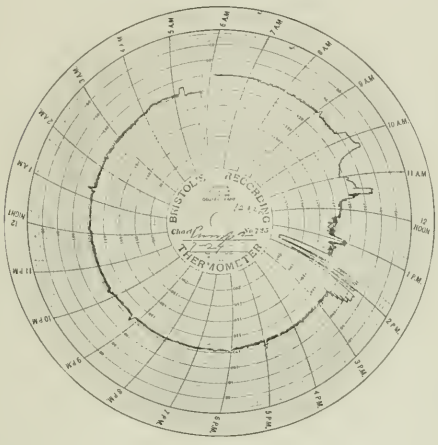


FIG. 3.—RECORD OF CONDENSER WATER TEMPERATURES.

Fig. 3 shows a record of condenser water temperatures, as recorded by one of these thermometers on a chart having range of 40° Fahr. to 220° Fahr. It will be noted that this chart has a reversed scale of uniform graduations. For condenser water temperatures the reversed scale is a desirable feature, as the average temperatures are usually lower than 125° Fahr., whereas the temperatures are likely to reach boiling point occasionally, at times of loss of vacuum in the condenser, as shown by the record in Fig. 3, between 1:30 p. m. and 1:45 p. m.

American Museum of Safety.—The formal exercises in connection with the permanent exhibition of safety devices of the American Museum of Safety were held on Nov. 21 at the Auditorium of the Engineering Societies Building, 29 West Thirty-ninth Street, New York. The object of the Museum is the conservation of human life by means of a permanent exhibit of the best and most practical safety devices for making safe the dangerous parts of machines and processes. Mr Philip P. Dodge presided. Among the speakers were the director of the Museum, Dr. W. H. Tolman, who discussed Germany's system of accident prevention; Prof. F. R. Hutton, Mr. T. C. Martin, Mr. E. S. Lott and Dr. N. E. Ditman. Mr. S. C. Dunham, president of the Travelers' Insurance Company, presented the gold medal offered by his company to the individual industrialist or corporation that has done the most for the protection of the lives and limbs of their workmen by means of safety devices for machines or processes to the United States Steel Corporation Mr. W. B. Dickson, first vice-president, accepted the medal on behalf of the Steel Corporation.

Personal

Dr. **Leo H. Baekeland,** of Yonkers, N. Y., has been awarded by the City of Philadelphia, on the recommendation of the Franklin Institute, the John Scott Legacy Premium and Medal for his invention of bakelite, in accordance with the report of the Institution's committee on science and arts, adopted May 4, 1910.

Mr. **Walter Brinton,** superintendent of the manganese steel department of the Taylor Iron & Steel Company's plant at High Bridge, N. J., since 1895, has resigned and has accepted a position as consulting engineer for the Edgar Allen American Manganese Steel Company, which is manufacturing manganese steel at Chicago Heights, Ill., and at New Castle, Del. Mr. Brinton's headquarters will be at the New Castle plant.

Dr. **Allerton S. Cushman** has resigned his position with the United States Department of Agriculture to found the Institute of Industrial Research in the City of Washington. The object of the Institute will be to undertake investigations of industrial problems in all lines, but more particularly of investigations concerning the protection and conservation of industrial metals; agricultural, land and fertilizer problems; hydraulic cements; paint technology; chemical industry, chemical engineering and electrochemistry. The Institute will be located in the heart of the City of Washington, near the White House, and fronting upon the Potomac River Park. Dr. Cushman has associated with himself Mr. Henry A. Gardner, who will have charge of the division of paint technology, and Dr. N. Monroe Hopkins, who will be in charge of the division of electrical engineering and electrochemistry.

Mr. **I. A. Goodell,** well known among smelting men in the West, was in Denver last month, having completed the work on which he was engaged in Montana.

Mr. **H. A. Guess,** manager of the Federal Lead Company, Flat River, Mo., recently spent some time in Salt Lake City on professional business for the interests with which he is connected.

Mr. **George A. Guess** is now metallurgical superintendent for the Cerro de Pasco Mining Company, Cerro de Pasco, Peru.

Dr. **Leo F. Guttmann** and Mr. **Camillo J. Guttmann** announce that the practice of their father, the late Mr. Oscar Guttmann, is now being carried on by them under the name of Oscar Guttmann & Sons. Specialties of their consulting practice are the erection of explosive plants, acetone, sulphuric acid, nitric acid and acetic acid plants, the fixation of atmospheric nitrogen as nitric acid, nitrates and cyanamides, the design, erection, remodeling and supervision of chemical factories, etc.

Mr. **D. C. Jackling,** of Salt Lake City, who is in charge of the operations of several of the large Western copper propositions, has found it necessary to deny a persistent rumor to the effect that he would assume general managership of all of the Guggenheim Western interests.

Mr. **Robert Kann** has formed partnership with Mr. R. J. Wysor in order to extend the scope of his laboratory. Mr. Wysor has been connected for a number of years with the Carnegie Steel Company. The new firm is established at 62-64 Gold Street, New York City, and is prepared to do general metallurgical work and to take charge of weighing and sampling of shipments.

Senor **Ramon F. Lujan,** of Mexico, has been spending some time in Denver on business relating to zinc production in Mexico.

Mr. **Charles Mindeleff,** chief chemist for the A. S. & R. Company, Perth Amboy, N. J., has been making a tour of the Western United States Mints in a professional capacity.

The Colorado School of Mines has issued a biennial report on the condition of the institution. In the last report, 1908, the enrollment was 307 students, of whom 46 per cent were from outside the State. At present the enrollment is 331 students, of whom 53 per cent are registered from outside the State. Six years ago only 26 per cent of the students were from outside the State. Thirty-six States, the District of Columbia, Alaska and the Philippine Islands are represented in the student body, in addition to seven foreign countries. The total number of graduates, including 1909, is 580. The inventory of property shows a total value of $723,841, a gain of 11.6 per cent in the biennial period. The report contains an appendix on non-metallic minerals of Colorado, by J. C. Bailar.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Electric Furnaces (Continued).

695,939, March 25, 1902, Clarence S. Loman, of Everett, Mass.

Resistance type, for the production of oxides of tin and lead. The furnace is a horizontal trough having its ends widened and deepened to receive vertical terminal plates of copper. The narrow intermediate portion of the trough is covered by a collecting chamber, through one side of which project tuyères. A flue or chamber for the oxidized products opens from its other side. For the manufacture of binoxide of tin the metal is preferably maintained at a temperature of about 1200° C. and air is suplied at a temperature of about 400° C. To produce putty powder—a mixture of the binoxide and protoxide of tin—the molten tin is maintained at about 1200° C., and cold air is introduced. Cold air is also used for the manufacture of lead oxides.

696,004, March 25, 1902, George D. Burton, of Boston, Mass.

Arc-in-liquid type, for the smelting of ores of gold, silver, lead, etc. The charge is distributed in receptacles molded from fireclay and railroad cinders and baked. These receptacles have perforated flat bottoms dipping into a solution contained in a tank. The level of the solution is varied by a displacing plunger. The positive electrode is a vertical lead plate in one end of the tank. Each receptacle contains a negative electrode, imbedded in the charge, being a disk of carbon on one of copper, carried by a swinging arm. The receptacles are covered. In operation an arc is established between the liquid and the charge in the receptacles. The molten products pass through the perforated bottoms and are chilled in the liquid. The volatile products escape through a pipe leading from the cover to a condenser. Baffles in the pipe retain solid products. For the treatment of ores containing nickel, iron, sulphur and arsenic the solution may consist of water, 30 gal.; salsoda, 10 lb.; and sodium chloride, 2 lb. For the treatment of lead or silver-galena ores the solution consists of water, 30 gal.; and sodium chloride, 10 lb. For the treatment of silver-galena ores an electrode disk of antimony may be substituted for that of carbon. Ores containing sulphur, antimony and arsenic are fluxed with chlorides of sodium and ammonium. It is stated that ores containing gold, silver and lead give separate globules of these three metals. Oxygen may be injected into the escaping sulphurous fumes.

697,810, April 15, 1902, Michael R. Conley, of New York.

Resistance type, for the reduction of iron ores and production of steel for ingots. The furnace is a vertical stack, having a constricted portion toward its bottom. Radial electrodes of mixed plumbago and clay project through the walls at its neck. Other radial electrodes pass through the walls of the crucible beneath, which has tap holes for metal and slag. The metal decarburized at the neck can be recarburized or treated with manganese in the bath below.

698,766, April 29, 1902, A. Voelker, of Ehrenfeld, Germany.

Combined arc and resistance type. Materials for making glass are forced upward through conduits, by screw-conveyors, into electric arcs. The fused product runs downward into a refining pot, wherein it is heated by a current passed directly through it, between carbon-plate electrodes situated behind perforated partitions. The current density is 5 amp per square millimeter of electrode surface. The refined product overflows into a third chamber beneath. The walls of the melting and refining chambers are of chamotte.

BOOK REVIEWS.

More Recent Cyanide Practice. Edited by **H. Foster Bain.** Second edition. 418 pages, 88 illus. Price, $2. San Francisco: Mining and Scientific Press.

This is an appendix to (not a second edition of) "Recent Cyanide Practice," and is a reprint of articles on this process which have appeared in the *Mining and Scientific Press* from 1907 to 1910. In the 77 articles here reprinted is to be found some of the most recent information on all phases of the process. While too much in detail for general reading or students' use, yet the collection will be of great value to every practical user of the cyanide process.

Van Nostrand's Chemical Annual. A handbook of useful data for analytical, manufacturing, and investigating chemists and chemical students. Edited by **John C. Olsen,** A.M., Ph.D., professor of Polytechnic Institute, Brooklyn. Second issue, 1909. 580 pages. Price, $2.50 net. New York: D. Van Nostrand Company.

The first edition of this "annual" appeared in 1907, the second around January, 1909. Thus the term "annual" is misleading. However, there is really no inherent reason why a book of this kind should be republished every year. In spite of the rapid advance of chemistry in many fields a very large part of the book must always remain unchanged in succeeding editions; and many of the minor changes, especially those caused by the yearly changes in the "international" atomic weights, would be too insignificant to induce American chemists to buy a new edition every year.

The book is decidedly useful and has deserved the favorable reception which has been accorded to the first edition. It is what it is intended to be, a convenient reference book of numerical data, and it is especially to be recommended that all tables and numerical data have been quoted from the original source wherever possible.

There are introductory tables of atomic weights, physical constants of elements, etc. (47 pages). Then follow tables for the calculation of volumetric analyses (30 pages) and gas analyses (18 pages), physical constants of inorganic compounds, organic compounds, alkaloids and essential oils, and melting points and composition of lead-tin-bismuth-cadmium alloys (280 pages), specific gravity tables (64 pages), equivalents of weights and measures (10 pages), thermochemical data (12 pages). All this is excellent and very useful, though one might wish to have the thermochemical part considerably enlarged.

The book is concluded by a review of chemical literature and new books. This is well prepared. But as the review covers only "the more important articles" and is therefore acknowledged to be incomplete and as this information is easily available elsewhere (for instance, in "Chemical Abstracts"), the editor may find it useful to drop this part of the book when he needs the space for more tables in later editions. As it is the size of the work is just within the limits of a handy handbook. To enlarge the volume would make it without doubt unhandy.

Besides the editor, Dr. J. C. Olsen, the list of contributors includes Dr B. B. Boltwood, Dr. V. J. Chambers, Dr. C. A. F. Kahlbaum, Dr C. H. Lips, Dr J. L. R. Morgan, Dr. E. E. Reid, Dr A. Rogers, Mr. A. F. Seeker, Dr. A. Seidell and Mr. W. C. Whipple.

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The Conquest of the Air by the Chemist.

If we leave aside early and isolated experiments, the endeavor of the chemist to transform the inert elemental nitrogen in atmospheric air into some nitrogen compound of commercial value, has extended on a systematic basis and on a relatively large scale over a little more than the last decade. This issue contains two articles particularly interesting with respect to the Birkeland-Eyde and Badische Company plants in Norway and the new Pauling plant in France for the production of nitrates by arc discharges. The more we learn of the different processes—differing essentially in the type and design of the arc furnace employed—the more it appears that with respect to the yield of grams of fixed nitrogen in form of HNO_3 per kw-hour there is not much of a choice between them; hardly much if any progress has been made in this respect over the unlucky Niagara Falls pioneer plant of Bradley and Lovejoy. Perhaps this is not so very astonishing since in the essential principle all the processes are alike.

But progress has been made in raising the concentration of NO in the gas mixture which leaves the furnace, and very much progress in simplifying the design. Without making any hideous comparisons, the enormous advance made in simplifying the construction is evident: this must find its expression in dollars and cents in the first cost of the plant. How important this is for a new industry will be clear to anybody only slightly acquainted with the ways of capitalists, who are, after all, very human beings like the rest of us.

With respect to the yield of fixed nitrogen per kw-hour we may compare the nitrate process with the calcium cyanamide process, which, though essentially different, is also a method of atmospheric nitrogen fixation. If 1 kw-hour produces 70 grams HNO_3 it binds 16 grams of nitrogen. On the other hand, in good commercial cyanamide practice 1 kw-hour binds some 70 grams of nitrogen in form of cyanamide. Naturally such a comparison, though interesting, is quite one-sided.

In a recent discussion before the New York Section of the American Electrochemical Society Dr. M. Loeb pointed out that as the formation of NO from air at the temperatures of the arc represents equilibrium, the attempt should be made to remove the NO in some way from the air by absorption in some suitable body so that in order to re-establish equilibrium new NO would be formed, etc. Dr. Whitney pointed out that such absorption might be very difficult at the enormous temperature of the arc. Yet the principle is plausible. In Haber's synthesis of ammonia—another instance of fixation of atmospheric nitrogen—such absorption is carried out and makes the process really practical. But here we work in the cold and absorption is comparatively easy. Clearly, the whole subject is in a state of evolution. It was first vigorously attacked by electrochemists with their inborn insurgent spirit. It is now being taken up by conservative business men. Such a combination must prove invincible in the end.

The Tremendous Magnitude of Western Copper Operations.

During 1910 no metal was the subject of such earnest and careful thought as copper. Overproduction, which had been freely predicted, finally became so great that insistent demands were made upon producers to reach some agreement for curtailment of output. One of the large determining factors was the tremendous development of Western low-grade copper properties, commonly spoken of as the porphyry coppers, which permitted operations on so large a scale that great quantities of copper were available at the remarkably low figures of 7, 8 and 9 cents per pound. The unquestioned ability of these companies to produce largely and cheaply, and the preparations being made by similar companies to enter the list of producers during 1910 and 1911, placed the prospects for the copper market in a bad light. Curtailment was necessary, and was accomplished in some measure, about 10,000,000 pounds per month, although the net result was not as encouraging as had been hoped for. Curtailment was urged as the best temporary method of dealing with a bad situation, for the well informed knew it to be a statistical fact that low prices of themselves would not curtail production and that it would be foolish to continue normal or increased production in the mere hope of a recovery of prices and the probability of causing considerable copper to be sold at a loss.

The development of the Western coppers was an important factor for another reason. On account of the character of the deposits and the nature of the treatment, it was possible to place them on a sound financial basis as investments, making them partake more of the nature of an industrial or manufacturing enterprise with known quantities of raw material and fixed costs of production than was usually possible in a mining venture. In brief, it was possible by calculation to see the end from the beginning and determine the probable life of the business. Furthermore, by combining in one company or group of associated companies the mining, concentrating, smelting and transportation features of the business, the profits of each transaction contributed to the net result.

The Utah Copper Company is the largest present producer of copper concentrate. Originally owning two plants, it later acquired that of the Boston Consolidated and then dismantled the smaller of the original two. The original Utah copper mill at Garfield, Utah, had a daily capacity of 6000 tons of ore averaging 2 per cent copper. Improvements made since then have raised this to 10,000 tons, which is the official figure given in the company's last quarterly report. The Boston Consolidated mill had an original capacity of 3000 tons per day, but after its acquisition by the Utah Copper Company it was wholly remodeled so that its capacity was raised to 6000 tons. This gives the Utah company a total capacity of 16,000 tons of ore per day, with the possibility of forcing the tonnage to a higher figure. Operations of such magnitude become a forcible factor in any field, and when it is considered that the cost of production after smelter deductions are made is from 7½ cents to 8¼ cents per pound of copper, it readily will be understood why a policy of curtailment was inaugurated awaiting the normal recovery of the market.

The Nevada Consolidated Copper Company, operating in White Pine County, Nevada, concentrates its ore in the Steptoe Valley mill and is at present treating about 5000 tons of 2 per cent ore daily. If plans are carried out it will eventually have a daily capacity of 15,000 tons. It is constructed in units of 1500 tons capacity, and 10 such units are embodied in the plans for the complete plant. The cost, as given in the reports of the company, is about 7.9 cents per pound of copper produced, after crediting the cost of copper with the income from the railway and from investments in two associated companies.

The Ray Consolidated Copper Company, Arizona, is one of the porphyry coppers which has not yet come into production, but it gives promise of being a large factor. The assured ore in Ray properties has been brought up to 59,000,000 tons, and the acquisition of the stock of the Gila Copper Company adds about 16,000,000 tons, or a total of more than 75,000,000 tons of 2 per cent copper ore. The mill under construction is planned to have a capacity of at least 8000 tons of ore per day, which should give an annual smelter output of 80,000,000 lb. of refined copper. This present estimate may be increased as development proceeds, for in the company's first report it was stated that the mill would have a daily capacity of 5000 tons, and this figure has now been exceeded. The estimate of cost is not in excess of 9 cents per pound of copper.

The Chino Copper Company will operate in New Mexico and is expected to commence production early this year. Its developed tonnage is 19,000,000 tons of 2.59 per cent copper ore, which is exceptionally high grade for deposits of this character. The concentrating plant is designed to treat 3000 tons per day, but, as has been the case with other mills of this character, the estimate probably will be exceeded. On this basis, however, the company will add from 35,000,000 to 40,000,000 lb. of copper to the 1911 production at a cost not over 9 cents.

The newest copper prospect of the first magnitude is the Inspiration, near Globe, Ariz. The ore is a disseminated deposit about 180 ft. thick. The tonnage which has been proved is 19,000,000 tons of 2 per cent copper ore, and this on an area of only 29.9 acres of land. An experimental mill of 75 tons daily capacity is practically completed, and from the results therein obtained will be planned a mill of 5000 tons daily capacity. Undoubtedly the Inspiration will join the goodly company of low-cost producers mentioned above.

The advent of all of these Western companies into the field under full production will be a factor to be reckoned with. It was pointed out a year ago that their output would be taken care of in the normal course of trade if a policy of curtailment were inaugurated to materially reduce visible stocks before the full flood of Western copper began to find its way into the market, but even now signs are not wanting that a copper convention must be entered into if the integrity and prosperity of the copper industry are to be maintained.

Double-Stage Crushing.

In double-stage crushing as practised on the Rand the stamp has been held in higher esteem than the tube mill as a mechanically efficient machine. The supporters of the tube mill have pointed out, however, that in usual practice the stamp is favored by receiving material which does not require reduction in the first stage, or rather, which is already reduced, while the tube mill is discriminated against by charging it with all the material requiring second-stage reduction.

In this connection recent experiments in South Africa by E. H. Johnson have shown a remarkable performance in double-stage crushing. By first separating the fine ore from the stamp feed and delivering it direct to the tube mill he has obtained from 10 stamps and one tube mill a duty of 26.1 tons per stamp in 24 hours, and a final pulp containing 0.5 per cent remaining on a 90-mesh screen.

Compared with single-stage crushing by stamps and fine screens, the above combination showed an increase of 32 per cent in horse-power efficiency and 31 per cent in grinding efficiency. This indicates possibilities for the tube mill which have not yet been fully appreciated, and which will tend to place double-stage crushing on a firmer basis than ever. As a fine comminuter the tube mill is practically sure of its position in any system of stage reduction, regardless of the machines which may be used in connection with it. Differences of opinion exist regarding the value of stamps as coarse crushers, and some metallurgists believe that rolls and chilean mills would serve the purpose better, but in regard to the tube mill opinion is hardly divided.

Some Handicaps to Iron Exports.

Considerable satisfaction is felt over the fact that our exports of iron and steel products in 1910 made a new record for tonnage by fully 15 per cent, the total of scrap, pig iron, unfinished steel, finished rolled steel, wire products, pipe, etc., amounting to about 1,540,000 gross tons, against a previous record of 1,325,740 tons, made in 1906. Of course, every step forward is a gain, but our export movement is still very small compared with our production. Much of the material exported is well advanced in stage manufacture, the declared value averaging about $40 per gross ton, and the tonnage of pig iron represented in our total exports is considerably in excess of 1,540,000 tons, perhaps 1,700,000 tons, but that is only 6 per cent of our total production in 1910.

Now, 6 per cent is a pretty low figure in view of the fact that in the Lake Superior region we have, as is generally admitted, the finest series of iron ore deposits in the world, and that in the Connellsville region we have as fine coking coal as is available anywhere, and elsewhere we have coal which in the by-product oven makes an entirely satisfactory coke. Copper, with which Nature has also bountifully supplied this country, we export very largely, as contrasted with the meager 6 per cent which obtains in iron.

The divergence in the movement of iron and copper suggests that our iron exports are not what they might be under better domestic conditions, and a glance at the world movements in iron confirms the suggestion. For we observe England making about 10,000,000 tons of pig iron a year and Germany 12,000,000 to 13,000,000 tons, and each exporting about 4,000,000 tons of iron and steel products, or from six to seven times as large a percentage of the production as we export of ours. These countries labor under certain disadvantages. England brings much of her ore from over the sea, her own ores being relatively lean, while Germany's ores are lean and far from tidewater.

It is true that our exports are in a measure restricted by commercial considerations. Our iron and steel export movement is not a "dumping" process but a regular commercial operation. Once our iron and steel exports were strictly of a dumping character, when they took the form chiefly of pig iron, steel billets and sheet bars and rails. In recent years a fairly definite understanding has been built up in the international markets whereby better prices, relative to manufacturing conditions, are obtained than formerly, and our exports cannot suddenly be increased without disturbing the markets, to the disadvantage of ourselves as well as of those sellers whom we should thereby displace. Nevertheless, our exports would have grown more rapidly had we not been subject to certain serious handicaps, the chief of these being the high freight and high labor costs.

There is little doubt that we are at the peak in the curve of labor costs in the United States. It is not reasonable to suppose that the constant increase for a dozen years presages a similar advance for another dozen, and the law of compensation seems even now to be preparing to effect an adjustment. To the iron and steel industry, however, relatively little relief is to be expected, for the movement to shorten hours and provide injury and death reliefs promises a counter influence.

In the matter of reduction in cost of transporting materials some relief is promised as the years pass. The suggestion may sound strange at the moment, when the railroads are so strenuously insisting that they must have higher freight rates in order to maintain their position, but we speak of adjustments as between commodities. A peculiar position prevails as between the railroads and the iron and steel producers. The railroads were long the iron trade's best customers, and in the eighties at least the railroads easily consumed 60 per cent of the product of the iron industry. In the nineties they consumed nearly as large a percentage, and their purchases up to the early part of 1907 were also large, though much smaller, in point of percentage, than in the eighties.

Through the fact of the railroads being the iron trade's best customers and through the movement of iron and steel materials being of a mixed character, it came about that freight rates upon these materials were established at a higher level than would likely be selected were the whole problem to be worked out in the light of present-day conditions. The ton-mile rates paid by the iron and steel industry upon ore, coal, coke, limestone, pig iron, unfinished steel and finished steel average fully as high as the average of all rates paid upon all freight moved over the railroads of the country. This is unnatural. The commodities involved are among the least subject to injury, and are among the most easily handled, of all freight. Enormous tonnages are moved in steady processions, frequently in solid trains involving the minimum expense. Of the value of a ton of finished steel at seaboard ready for export the freight paid in producing it forms a very large part, and there is reason to hope that in course of time this part will be reduced, so that the iron industry will pay no more than its fair share of the expenses of the railroads in conducting their operations.

We do not wish to be understood as intimating that American railroads would suffer by a comparison with those of other countries, for that is not our belief. But we do not think that at present the iron industry is enjoying its full share of the economies effected in American railroading, whereby they are able, as we understand, to stand comparison with English and continental European practice.

Minnesota Meeting of the American Chemical Society.

The forty third general meeting of the American Chemical Society, held at Minneapolis, Minn., from Dec. 28 to Dec 31, was attended by 275 members and guests and, like all recent meetings of the Society, it was an echo of the general enthusiasm that pervades its membership.

Owing to the fact that over 175 papers were presented, it was necessary to hold meetings of all of the Divisions and the meetings of the Division of Agricultural and Food Chemistry, Division of Fertilizer Chemistry, Division of Pharmaceutical Chemistry, Division of Industrial Chemists and Chemical Engineers, Division of Physical and Inorganic Chemistry and Division of Organic Chemistry were well attended.

A point especially worthy of note was the success of the Biological Section under the chairmanship of Dr Carl L. Alsberg, of Washington, D. C. Forty-four papers were presented before this section of a strictly chemical nature bearing upon biochemical problems.

The meeting of the Section of Chemical Education, held at a time when all the industrial chemists could meet with them, proved decidedly interesting and the discussion was general.

On the first day of the meeting the chemists all gathered in general session to listen to eight general addresses as follows:

The Lost Arts of Chemistry, by W. D. Richardson.

The Basis of Industrial Efficiency, by A. D. Little.

Synthetic Metals from Non-metallic Elements, by Herbert N. McCoy

Progress in Food Chemistry, by H. E. Barnard.

Mechanism of Cell Activity, by Carl L. Alsberg.

Waste Wood and Some of Its By-Products, by Geo. B. Frankforter.

The Formation of Carbohydrates in the Vegetable Kingdom, by Wm. McPherson.

The Efficiency of the College Graduate in the Chemical Industry, by Chas. F. Burgess.

The address of the retiring president, Wilder D. Bancroft, on "A Universal Law" was an exposition of the general application of the theorem of LeChatelier to all kinds of natural phenomena as well as those of a chemical nature.

Possibly the "complimentary smoker" in the rooms of the Commercial Club was as thoroughly enjoyed as any one feature of the meeting, for an unusually witty program had been arranged and original songs bearing upon individual chemists and chemical phenomena were thoroughly enjoyed; also the mementos furnished to the visiting chemists were pleasing to all. This, however, was not the only social function of the meeting, for several receptions and teas were arranged for the ladies and the banquet on Friday night, which was attended by some two hundred members, was also graced by their presence.

On Saturday afternoon a large number were treated to a sight seeing trip covering both the cities of Minneapolis and St. Paul. A number of the chemists also had the privilege of viewing the wonderful paintings of La Farge in the State Capitol of Minnesota. Excursions were made to the following works Russell-Miller Milling Company, Pillsbury Flour Mills Company, Washburn-Crosby Company, Consolidated Milling Company, Minnesota Linseed Oil and Paint Company, Archer-Daniels Company, Midland Linseed Oil Company, St. Anthony Falls Water Power Company, Cream of Wheat Company and International Stock Food Company.

The announcement of the election of Professor Alexander Smith as President of the Society and of Dr Louis Kahlenberg, Dr. Frank K. Cameron, Dr. Geo. B. Frankforter, Dr. F. C. Franklin and Dr. E. G. Love as Councilors-at-Large met with hearty approval and applause

Prof. M C Whitaker was elected editor of the *Journal* of Industrial and Engineering Chemistry and the report of a committee recommending a decided broadening of the policy of that journal was adopted.

The Secretary, Dr Charles L. Parsons, Durham, N. H., announced a net gain of 550 members for the year, the Society now having a membership of over 5100.

Inventors' Guild.

In the early part of 1910 several men who had done work along the line of invention and who, in developing and patenting their inventions, had come to realize the difficulties and disadvantages under which the inventor labors, instituted a movement for the formation of a society looking toward the betterment of these conditions. The result of this movement was the formation and incorporation in New York City of the Inventors' Guild, the object of which is briefly outlined in the following quotation from the constitution of the society.

"The object of the guild is to advance the application of the useful arts and sciences, to further the interests and secure full acknowledgment and protection for the rights of inventors, to foster social relations among those who have made notable advances in the application of the useful arts and sciences."

Some of the handicaps to which the inventor is subject, other than the proverbial one of never having any money, are the delays in the Patent Office and the ineffectiveness of its work, due to overcrowding and lack of proper facilities in the Patent Office; the expense and tardiness of litigation and the possibility under which a rich corporation, by delaying and prolonging a suit, may increase the expenses to a point which makes such suits prohibitive for a poor inventor; the disadvantage to which the American inventor is subject in foreign patent offices as compared with the liberality of the American Patent Office toward the foreign inventor, and many other conditions militating against the American inventor which should be remedied.

The membership of the Inventors' Guild is limited to 50. The idea of limiting the membership is that with a small society of this sort it is easier to accomplish real results than with a larger organization, hampered as it must be by unwieldiness and red tape. Further, with a small organization each man will feel that he is a working unit and that he will be depended upon to do real work, whereas in a large organization the general feeling is that there will be plenty of other men to do the work, and that lack of assistance from any particular member will make little if any difference. The result is that in the large organization the work, if any, is usually done by even a smaller number of members than that provided for in the Inventors' Guild.

It is proposed to select the membership of the guild carefully, and to this end a member must be formally proposed by a member of the guild, must be personally known to five members of the guild, must pass the membership committee and board of governors, and finally must be voted upon by the whole membership, 4 per cent of the votes cast being sufficient to reject a candidate. The object of such discrimination is to include among the members of the guild men who not only have made inventions, but who have achieved some measure of success in connection therewith, and who will therefore be capable of exerting some influence; and also that no one shall be admitted not congenial to practically the entire membership.

The officers of the Inventors' Guild are as follows: Ralph D. Mershon, president; Chas. W. Hunt, first vice-president; Chas. S. Bradley, second vice-president; Thomas Robins, secretary, and Henry L. Doherty, treasurer.

At the present time the guild has 29 members, as follows: Bion J Arnold, Dr. L. H. Backeland, W. H. Blauvelt, Chas. S. Bradley, Alex. E. Brown, Henry L. Doherty, Thomas A. Edison, Carleton Ellis, Stephen D. Field, James Gayley, Edward R. Hewitt, Peter Cooper Hewitt, Chas. W. Hunt, Dr. John F. Kelly, T. S. C. Lowe, Ralph D. Mershon, Ambrose Monell, Dr. Edwin F. Northrup, Prof. G. W. Pierce, Chas. E. Pope, Dr. Michael I. Pupin, Thomas Robins, Dr. F. Schniewind, C. H. Smoot, Prof. Carl Thomas, F. L. O. Wadsworth, Arthur West, Dr W. E. Winship, B. F. Wood.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our London Correspondent.)

Magnetic Properties of Iron and Its Alloys.

At a meeting of the Institution of Electrical Engineers early in December, Sir ROBERT A. HADFIELD and Prof. B. HOPKINSON presented a paper on "The Magnetic Properties of Iron and Its Alloys in Intense Fields." The materials for the research carried out by the authors were a series of alloys prepared at the Hecla Works, which had already been extensively investigated and many of their physical properties had been ascertained. They included iron-carbon steels, nickel steels and manganese steels in varying proportions.

The most important results obtained were: (1) Every alloy has a different saturation intensity of magnetism, which occurs mostly in a field of 5000 units, but in a few there is a slight increase between 5000 and 25,000. All the materials exhibit the behavior of a mixture of magnetic substances with non-magnetic substances, the permeability of which does not vary materially from unity. The non-magnetic or nearly non-magnetic nickel and manganese alloys show no exception. In none of the alloys examined does the permeability differ from unity by much more than 2 per cent. (2) There is in the series no alloy having a higher specific magnetism than pure iron. (3) The saturation value for pure iron of specific gravity 7.8 is 1675 absolute units, within 1 per cent. (4) In an annealed iron-carbon steel containing a small proportion of other elements the specific magnetism is less than that of pure iron by a percentage equal to six times the percentage of carbon. It is deduced that the magnetism of carbide of iron is about two-thirds of that of pure iron. (5) Quenching an iron-carbon alloy from a high temperature largely reduces this specific magnetism, but the amount of reduction is rather uncertain. (6) The addition of silicon or aluminium to iron reduces the specific magnetism nearly in proportion to the amount added, as if the addition behaved as an inert diluent; but if carbon is present silicon appears to neutralize its action to some extent. (7) In the case of alloys of iron with nickel and with manganese, or with both, no simple relation between their magnetism and composition has been observed. This might be due to the peculiar effects of heat treatment on these alloys. The authors, in conclusion, direct attention to the following general points: Among the alloys there are only one or two whose magnetism is greater than the sum of the magnetism of the separate constituents, a specimen containing 11.4 per cent of nickel being especially noticeable. Secondly, the magnetic properties of every alloy, even those which are almost non-magnetic, are very closely expressed by the equation $B = H + 4\pi I$, where I is the constant designating magnetism, provided that H exceeded 7000 C.G.S. units. Thirdly, comparison with Barrett's results in a field of 40 units, or less, shows that in some cases the magnetism of an alloy may be considerably greater, relative to that of pure iron, in low than in high fields. Barrett found that when the value of H is small several silicon-steels and one aluminium-steel are more magnetic than pure iron; but all these materials are less magnetic than pure iron when saturated. In these cases the coercive force is small. In some alloys with a very high coercive force, such as some of the manganese steels, the magnetism relatively to pure iron is much greater when saturated than in a field of 40—in some cases more than twice greater.

The discussion of the paper was opened by Prof. Silvanus P. Thompson, who said the facts put forward were a strong confirmation of Ewing's theory that the saturation value of intrinsic magnetism was practically a constant. If the research had done nothing beyond establishing the fact that the magnetic behavior of iron is as that of the sum of its constituents, yet a most important point would be established.

Mr. J. T. Morris thought that the research would materially assist in setting up a sound theory for the guidance of experimental work. The influence of temperature was most important. It was interesting to know that this point was now under investigation.

Professor Hopkinson in his reply said he hoped it would be possible to investigate the phases of iron by the magnetic properties of the metal. In the case of annealed carbon-iron alloys he thought it had been shown that the magnetism was a function of the amount of carbon. He did not think that the iron carbide was affected by the presence of iron; and it was clear, both from chemical and microscopical results, that in such alloys they were dealing with mixtures. As to the effect of temperature, he could say that in the case of pure iron a rising temperature lowered the saturation intensity. The reduction increased up to the critical point of 750° C., when the magnetism suddenly disappeared. It seemed probable that temperature affected the molecular constituents; and the question arose whether the reduction of magnetism resulted from the total destruction of the magnetism of certain molecules or whether it was consequent on a gradual change taking place in the whole. He thought the former explanation was more probably correct

Non-Ferrous Alloys for High-Pressure Castings.

At a recent meeting of the Institution of Mechanical Engineers, Prof. H. C. H. CARPENTER and Mr. C. A. EDWARDS contributed an interesting and valuable paper on "The Production of Castings to Withstand High Pressures." The first experiments were made with pure gunmetal containing 85 per cent of copper and 15 per cent of tin, a small quantity of copper phosphide being subsequently added to remove oxygen and leave not more than 0.1 per cent of phosphorus. With this alloy the authors repeatedly reproduced perfectly sound castings which, after being machined, withstood an average pressure of 18 tons before they burst.

Experiments with aluminium bronze followed. The first alloy contained 91 per cent of copper and 9 per cent of aluminium, and the mixture was poured at 1125° C., or about 80° above its solidifying point. Only one unsuccessful casting was obtained out of six. After machining they were tested up to 12 tons to 13 tons, when slight stretching ensued and water came through the valve joint; but by tightenng up the joints the pressure could be raised to 14 tons to 15 tons, when further stretching resulted; but, after again tightening up, the pressure was increased to 18 tons, at which none of the four castings fractured. Consequently, 9 per cent aluminium bronze was capable of withstanding as high pressure as 15 per cent gunmetal; and, moreover, the former had greater ductility.

Up to this point all the castings had been made in iron chills. Further castings were made in green-sand molds, the pouring, as before, being from the bottom. Three castings failed to pass the preliminary test, and the other three were tested up to 10 tons when tightening of the joint was necessary. At 13 tons it stretched again, and, with alternate stretching and tightening of the valve joint, a pressure of 19 tons was attained. The walls were then machined down to varying thicknesses and further experiments carried out which appeared to indicate that if these tests had been carried far enough the cylinders would only have been fractured after the hardening produced by the water pressure had raised the yield stress to the same figure as the ultimate stress.

These results showed that cylinders of aluminium bronze could be cast, both in chills and in sand molds, to withstand a water pressure of 18 tons per square inch, but they had the disadvantage of beginning to stretch at a pressure of 10 tons to 12 tons—the sand casting giving the lower figure. A bronze containing 9.5 per cent of aluminium stood up to 11 tons before bulging, and a 10 per cent bronze was tested up to 12.5 tons before any expansion, while 10.5 per cent bronze stood up to 14 tons before any tightening was necessary. Finally, an 11 per cent aluminium bronze allowed hydraulic pressure to be taken up to 26 tons without any sign of leakage.

In their summary the authors stated that conditions had been

worked out for the production of non-ferrous hydraulic castings capable of withstanding pressures of from 14 tons to 20 tons. These results had been obtained with pure copper-aluminium bronzes containing 9 per cent to 11 per cent aluminium. Their largest experimental castings weighed 36 lb., including gate and riser, and the best proportion of aluminium for these was 9 per cent to 10 per cent. For smaller castings a higher percentage could be employed. The essentials for success were (1) very slow pouring; (2) casting at a temperature not higher than 50° C. to 80° C. above the initial freezing point; (3) the metal must enter the mold at the lowest possible point; (4) green-sand molds must not be too damp, and with close sand the lower part of the mold should be dried; (5) with large castings risers must be used to overcome shrinkage. The average of clean, sound castings was not less than four out of six. Their homogeneity appeared perfectly satisactory, and their corrosion in fresh or in sea water was extremely slight. So far as the authors were aware, no other bronzes possessed this combination of properties.

The discussion was opened by Mr J. Archbutt, of the Midland Railway, who remarked that the paper brought out the importance of attention to small details for the production of good castings. He wished to ask why the authors selected the 15 per cent tin alloy, which was very brittle. Quenching at above 500° C. had a very great influence on the properties of this 15 per cent alloy.

Mr. Bateman said his experience of hydraulic pressures for valves showed that phosphor-bronze would stand three times the working pressure for a long time, while ordinary gunmetal would stand that pressure for only a few weeks.

Mr. Harrison pointed out that commercial engineers had to adhere very rigidly to specification for such customers as the Admiralty. He had found that the metal put into a large casting amounted to 15 times the weight of the finished article.

Mr. C. A. Edwards, in the course of his reply, said it was better to use a big head and ensure a good casting. In the pressure tests they filled the castings with water and pumped up the pressure on top with oil. They had tried quick pouring, and that was why they had so many bad castings at the outset. As to adhering to specifications, if castings would stand higher pressures and be perfectly safe the users should not quibble about other matters.

The Estimation of Phosphoric Acid.

At a recent meeting of the Society of Public Analysts Mr. G. F. Wesley Martin described a new method. The phosphoric acid is first precipitated at ammonium phosphomolybdate, which, after washing, is dissolved in excess of caustic alkali, and the ammonia is then distilled into excess of standard acid and titrated in the usual way. This method has the advantage of fair rapidity, and appears to be sufficiently exact.

Market Report.

December, 1910.

Copper. Opening at £57.5.0 copper remained fairly steady until the close of the month when it showed a downward tendency, closing at £52.15.0.

Tin opened at £169.15.0 and rose smartly to £175 by the 12th, thence dropping to £171.10.0 by the 21st. After this recovered to £175.10.0 (28th).

Lead has been practically fixed at £13.11.3.

Hæmatite opened at 64/6 and appreciated slowly to 66/11½ (28th), the rise being more pronounced after the 9th.

Scotch Pig opened at 55/9 and has kept fairly steady, dropping slightly on the 6th and 13th. Closed at 55. 10½.

Cleveland Warrants opened at 49/9, and with the exception of small fluctuations, on the 6th and 13th, have remained steady, closing at 49/10.

Sulphate of ammonia, f.o.b. Liverpool, per ton	£12	17	6
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphide powder, per ton	22	..	..
Borax, British refined crystal, per ton	16	..	..
Sal ammoniac, lump, firsts, delivered U. K.	42	..	..
Copper sulphate, per ton	19	15	..
Camphor, 1-oz. tablets	..	1	7
Caustic soda, ash, 48 per cent, ordinary, per ton	5	10	..
Shellac, standard, T. N., orange spots, per cwt.	4	4	..
Sulphur, recovered, per ton	5	..	..
Carbolic acid, liquid, 97/99 per cent, per gal.	..	1	1
Hydrochloric acid, per cwt.	5	..	
Creosote, ordinary good liquid	..	..	2
Naphtha, solvent, 90 per cent at 160° C., per gal.	..	..	10
Petroleum, Russian, spot, per gal.	..	..	5½
Copper ore, 10 per cent to 25 per cent unit	9/1½ to 10/1½		
Tin ore, 10 per cent, per ton	£110 to £111		
Mica, small slab, in original cases, per lb.	6d to 1/-		
Litharge, flake, per ton	£15	15	..
Red lead, genuine, per ton	16	5	..
Bleaching powder, 35 per cent. per ton	4	5	..
Platinum, per oz. nominal	8	..	..

India Rubber, Para fine, opened at 6/3½, had dropped by the 7th to 6/1½, and thence to 5/10½ on the 14th and to 5/10 by the 21st, ultimately closing at 5/9 per lb.

Compared with prices for the end of December the following have appreciated:

Antimony has risen per ton	£2	..	..
Copper sulphate	..	5	..
Shellac, per cwt.	..	1	6
Carbolic acid, per gal.	..	..	1½
Creosote, per gal.	..	..	⅛
Naphtha, solvent, per gal.	..	..	½
Petroleum	..	..	¼
Tin ore, per ton	5	..	..
Litharge	..	10	..
Red Lead	..	10	..
Platinum, per oz.	..	6	..

Sulphate of ammonia only is quoted 2/6 cheaper.

London, England.

The Western Metallurgical Field.

Conservation in Metallurgy.

In discussing the efficiency of present metallurgical processes and the possibility of avoiding waste in their operation, Mr. Franklin Guiterman presented some forceful considerations in his annual address as retiring president of the Colorado Scientific Society. By way of introduction the speaker defined the meaning of the word "waste" on account of the "confusion which has arisen in the public mind from the reckless use of the word by superficial magazine writers and that brotherhood of doctrinaires which acclaims itself "conservationists." The waste of a product in whole or in part involves the sacrifice of something which might be profitably produced and which, if saved, could be of use to the community. It therefore follows that while waste is always non-utilization non-use is not necessarily waste.

The metallurgical industry is always at that state of efficiency which limits the further use of raw products of inferior grade, or which makes it inadvisable to carry the process to a higher degree of perfection, even if this be possible. Hence the "waste" of yesterday was due to limiting conditions which may be removed by the progress of to-day, while the improvements of to-morrow are anxiously awaited to reduce to-day's losses. In this sense so-called waste has been merely non-use, and many examples are available to show that the material which was not used years ago is to-day the source of remarkable profits due to improvement in processes of treatment.

An exceptional example of involuntary conservation of an unused product for over 20 years was the retention of some mill tailings from a Leadville mill. Cribbing was erected to prevent the tailing flowing onto neighboring property and into a stream. The product was the waste from concentration of a mixed sulphide ore, of which only the lead and iron content was then useful. The waste assayed Au, 0.04 oz. per ton; Ag, 6 oz. to 7 oz. per ton; Zn, 26 per cent to 27 per cent; Pb, 6 per cent

to 7 per cent; Fe, 20 per cent, and SiO_2, 8 per cent. The zinc ore, blende, was discarded for obvious reasons known to all lead smelters. To allow it to remain in the ore would have been to invite penalties which would have reduced profit to loss. Hence it was eliminated and only retained on the property unwillingly. Twenty years later, owing to the development of electromagnetic separation of pyrite and blende, the 80,000 tons of material in the dump netted the owners over $325,000.

This is probably not typical of anything that will occur 20 years hence with waste or unused products of to-day, for metallurgical progress is more rapid now, and, furthermore, any unused material which has a potential value is experimented with and, generally speaking, is always saved.

Among the unrecovered metals which are either lost in the metallurgical practice of to-day, or recovered only as a contaminating impurity of a more valuable product, are cadmium, bismuth and antimony. Cadmium occurs in almost all Colorado blendes in small quantities. It is a most useful metal for some purposes, but a detriment to its extended use is its high cost of production. Its tendency to rapid oxidation makes its recovery difficult and inefficient, and as it occurs only in small quantities as compared with the zinc, it is practically "wasted."

Bismuth and antimony are recovered in the blast furnace with lead, and contaminate it to the detriment of the price. Bismuth in amounts exceeding 0.4 per cent in lead excludes it from the market for corroding purposes, and the small amount of bismuth could, therefore, be well dispensed with. Here is an example of recovery of a product entailing a financial loss. The same applies to antimony occurring in lead ores, for it increases lead losses in refining and could be dispensed with to commercial advantage.

Another form of so-called waste which has been a fruitful field for those who like to reduce a loss to dollars and cents without regard to its other features is the elimination of sulphur during the smelting of sulphide ores. It would be possible to convert the sulphurous gases into sulphuric acid, but unfortunately there would be no market for this product under governing conditions. The smelters could produce more sulphuric acid in a day than is now made and sold by Western manufacturers in a month. As sulphurous acid is a gas which can be discharged into the air under proper precautions with comparatively little damage to vegetation, and as sulphuric acid is a liquid which must be contained in iron vessels, it is readily seen that the production of the latter would be a matter of large expense rather than of profit, and of serious embarrassment than of congratulation.

Arsenic also is contained in smelting ores, and is now recovered at little or no profit. If this element could be discharged to waste without damage its loss would hardly constitute a failure to observe the laws of conservation. When the time comes that the uses of arsenic warrant an increased production it can readily be supplied at short notice, but until then its recovery is hardly a commercial necessity as far as use is concerned.

Summing up, Mr. Guiterman believes that with respect to gold, silver, lead and copper, and the working costs obtaining in the processes, the limit of efficiency seems to have been reached. There is undoubtedly much room for improvement in the metallurgy of zinc, not only in point of metal recovery but also in the present high cost of smelting operations. But the glaring weaknesses in methods are to be found not so much in the larger as in the smaller enterprises, and it is in respect to the latter that the metallurgical engineer will be called upon to give his counsel and aid.

Progress in Copper Milling and Smelting.

At Needles, Cal., the United States Smelting, Refining & Mining Company has just completed and put in operation a 250-ton concentrator to handle low-grade copper ores. The company also has a smelting plant at Needles and the product of the concentrator will be treated locally.

The initial run of the first unit of the new Miami concentrator in Arizona was to have occurred in January, but delays incident to such a project have postponed operations until the present month so that the operation of the first unit may be coincident with this issue of Metallurgical and Chemical Engineering. The Miami is one of the large low-grade propositions that are being brought into production this year and is modeled on the lines which have proved satisfactory in older mills.

At the Garfield smelter, Utah, reverberatory copper smelting is being improved by running harder silica slags than were formerly considered standard practice. It has been customary to run lower silica slags in reverberatory furnaces than in blast furnaces, as experiment had shown that when silica rose above 38 per cent to 40 per cent in the former the capacity and efficiency of the reverberatory were decreased. Recently, however, reverberatory slags at Garfield have carried as high as 45 per cent silica, and at the Steptoe plant in Nevada as high as 43 per cent, with satisfactory results. This is made possible by exercising greater care in mixing and preparing the charge, and results in the advantage of reducing cost of operation and of fluxing material.

As previously stated in these columns, the California copper smelters have been operating at reduced capacity, or not at all, on account of injunctions obtained by the agriculturists requiring the smelters to abate the smoke and fume nuisance. The U. S. Smelting Company, at Kennett, put a bag house in commission, but it was able to handle the fumes from but two furnaces. Three had been tried but it was found impossible to cool the fumes sufficiently to prevent disintegration of the bags. Later reports, however, indicate that the system has been improved and that three furnaces are now in operation.

The First National Copper Company, owning the smelter at Balaklala, was closed pending the installation of the Cottrell process of electric fume condensation. It was expected that the plant would resume operations in January, as the necessary improvements would be installed by that time. The California metal production has suffered a serious setback as a result of the injunctions obtained by the farmers, and it is doubtful whether, under present market conditions for copper, some of the plants will resume operations with any kind of system of fume recovery.

Montana-Tonopah Mill Operations.

In the eighth annual report of the Montana-Tonopah Mining Company, operating at Tonopah, Nev., some data are given regarding the milling operations. During the year an average of 38.5 stamps out of 40 were dropping continuously. The tonnage milled was 50,245, and the average stamp duty was 3.67 tons per 24 hours. The average value of ore milled was $15.22 per ton (0.206 oz. gold and 21.139 oz. silver). The average value of tailing was $1.43 (0.0146 oz. gold and 2.168 oz. silver). The gross mill extraction was 90.8 per cent, and the cost of milling $3.37 per ton. The bullion produced cost 53.4 cents per ounce, and brought 67.4 cents per ounce.

The Non-Ferrous Metal Market.

With the exception of tin, all the metals suffered a reduction in price during the past month, and while fair business has been reported in all lines, only tin has enjoyed a heavy trade. The prospect for any material change is not considered good.

Copper.—It is generally believed that practically all the business transacted in sales of this metal has been consummated at concessions in price. Even the agencies which have been holding aloof from the market have shown a tendency to enter at current prices. Lake copper is quoted at 12⅝ to 12¾c and electrolytic at at 12.35 and 12.40c.

Lead.—A fair business has been transacted at the following prices: 4.32½ to 4.35c in St. Louis and 4.5c in New York.

Spelter.—Immediate and future deliveries of this metal have combined to make a fair volume of business. St. Louis prices have been about 5.35 to 5.37½c and New York 5.50 to 5.52½c.

Tin.—The continued strength of the foreign market has produced a local conviction that it is due to legitimate causes and buyers are entering the market more liberally. Prices rose somewhat during the month and the ruling price for the month was 30.5 to 40c.

The Iron and Steel Market.

The new year opened with the iron and steel industry running at between 40 and 50 per cent of capacity. Out of a total 33,000,000 tons, the capacity of blast furnaces nominally in blast was about 19,000,000 tons a year, but a capacity of 2,000,000 or 3,000,000 tons was banked to avoid accumulation of pig iron while the attendant steel works were idle. There had been a sharp slowing down during December, on account of seasonable conditions, but there has been relatively little speeding up during January and the gait at the close of January is little if any over 50 per cent of full.

A condition of extreme nervousness prevailed among buyers in December on account of widespread reports that prices of finished steel products might be reduced at the meetings of manufacturers which were expected to be held in connection with the dinner called for the night of Jan. 11. Among producers there was practically no expectation that any change would occur in prices, the dinner being intended rather to encourage adherence to established prices. After Jan. 1 orders began to come in more freely, chiefly in the form of specifications against contracts, and representing requisitions which had been held back to avoid complicating the annual inventories. After the dinner of Jan. 11, with its failure to develop any change in prices, the flow of business experienced further, although slight, improvement.

The total of January bookings in finished steel products promises to show a slight gain over December, and, as January production has been less than December, a gain in unfilled orders on books is likely to be shown. A test of this is furnished by the Steel Corporation's reports. During the whole of 1910 its unfilled orders decreased, but the December decrease of 85,656 tons was the smallest. The total on Dec. 31, 1910, was 2,674,757 tons, the smallest ever reported. A gain is expected to be shown during January of between 60,000 and 120,000 tons.

All expectations of lower prices for finished steel products in January were disappointed, while a surprise, at least to the general trade, occurred in the other direction, since effective Jan. 23 all wire products were advanced $1 per ton, making plain wire 1.55c, base, wire nails $1.75, base, painted barb wire 1.75c and galvanized barb wire 2.05c. The advance was effected by the leading interest announcing its new price, the independents promptly following suit. A conference between a few of the larger interests had previously been held.

The wire advance was made, as such wire advances frequently are, for the purpose of stimulating specifications. The large jobbers had previously been well covered by contracts, permitting specifications to be filed for 60 days, and toward the close of the period it will depend upon the condition of mill order books how much tonnage they will permit to be specified.

The wire advance of $1 a ton follows a decline of $3 a ton during the first seven months of 1910 and successive advances totaling $5 a ton after the great reduction of May 1, 1909, so that the new prices are $3 a ton higher than the lowest prices made after the great market break which started Feb. 10, 1909. Prior to the advance just made the wire mills were running very slack, much less than 50 per cent of capacity, but have been doing better since and will probably be running practically full by the beginning of March.

It seems to be the general opinion that the decline in activity has carried the iron trade well below the normal and that a marked upturn is a certainty, the time only being in doubt, and predictions of a distinct improvement have been made, naming all the dates from Feb. 1 to May 1.

Railroad buying showed a slight improvement in January and there are hopes of further improvement upon the settlement of the freight rate issue by the Interstate Commerce Commission. This is now expected not later than March 15. The Pennsylvania Railroad Company and the rail mills have at last agreed upon specifications for the 150,000 tons of rails for which the railroad offered orders for this year's delivery. This company has ordered about 2000 freight cars and a few other car orders have been placed. No very heavy railroad buying is expected, but a moderate improvement over conditions in the past eight months is to be expected, as many of the roads are short of equipment in one respect or another. There is no prospect of a big rail year. Orders thus far placed indicate a total of between 1,000,000 and 1,500,000 tons for replacement purposes on the steam roads, and with the various other demands the total rail production of the year bids fair to be between 2,500,000 and 3,000,000 tons, not far from the output in 1909 and contrasting with almost 4,000,000 tons produced in 1906, the record year.

Prices of finished steel products are being held quite well. There is occasional shading in one product or another by small interests, but the leading producers, including the Steel Corporation and some eight or ten independents, are holding their prices rigidly, and the attitude of these interests is that they will hold prices irrespective of occasional shading by smaller interests, but will abandon all reserve should any case develop of shading by any large interest. In other words, divergences from the present schedule will hardly prompt an attempt to make a fresh stand upon some level a little lower, but will provoke an open market.

Pig Iron.

Prices have yielded to the extent of about 50 cents in the Philadelphia, Buffalo and Chicago markets, the Pittsburg and Alabama markets being unchanged. There was heavier buying in January than in December, but chiefly by the action of a few consumers who make a practice of buying at this particular time. Merchant pig iron stocks are still accumulating, on the whole, but in some districts stocks are about stationary. Prices at valley furnaces, 90 cents higher delivered Pittsburg, remain as follows: Bessemer, $15; basic, $13.50; No. 2 foundry, $13.75; malleable, $13.50; forge, $13.25. No. 2 foundry is $15.50 at Philadelphia and $14 at Buffalo, No. 2 being $14.25 at Cleveland, $15.50 at Chicago and $11 at Birmingham.

Steel.

Very little business has been done in billets and sheet bars on account of so much tonnage being held over on contracts which should have expired Dec. 31. Prices are very well held on the basis of $23 for billets and $24 for sheet bars, Bessemer or open-hearth, f.o.b. Pittsburg or Youngstown. Forging billets are $28, Pittsburg, with the corresponding price of $30.40, Philadelphia, nominally quoted, while actual sales have been made at $27, Chicago. Rods are $27.50, Pittsburg.

Finished Steel.

Finished steel prices are unchanged, except as to wire products, and are as follows, f.o.b. Pittsburg except where otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3-inch to 15-inch inclusive, zees and angles 2 x 3 and larger.

Steel bars, 1.40 cents, base; iron bars, 1.35 to 1.40 cents, Pittsburg; 1.30 cents, Cleveland; 1.30 cents, Chicago.

Wire nails, $1.75, base, per keg; plain wire, 1.55 cents, base; painted barb wire, 1.75 cents; galvanized barbed wire, 2.05 cents.

Black sheets, 28 gage, 2.20 cents; galvanized, 3.20 cents; blue annealed sheets, 10 gage, 1.65 cents; painted corrugated roofing, $1.55 per square; galvanized, $2.75 per square.

Tin plates, $3.60 for 100-lb. cokes.

CORRESPONDENCE

Copper Poisoning.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—We have just had an experience here that seems to warrant the publication of this warning to any one who contemplates working with copper in the electric arc furnace.

We melted 5000 lb. of electrolytic copper scrap—uncontaminated except for a small amount of admixed iron—in a three-phase arc furnace of the ordinary steel furnace type, the experiment lasting some five hours. A few hours after pouring off all of the ten men in the furnace building suffered inconvenience in breathing. For the 24 ensuing hours severe nausea was experienced in each case and a soreness throughout the entire system similar to that of acute grippe.

Since in all other respects the run resembled an ordinary steel run, I attribute the trouble we experienced to copper which was shown to be present in the furnace fumes. The temperature of the copper bath as a whole at no time exceeded 1300° C, so that probably the copper was volatilized only at points directly beneath the electrodes.

From the severity of the attack, coupled with the short time we were exposed to the fumes, and that in a building which was well ventilated, I am of the opinion that very serious results might obtain under less favorable conditions.

I also attach more importance to the above from the fact that in five years of continuous research work on a large scale with furnace processes which involved, among others, tellurium, tellurides, arsenic, arsenides, zinc and sulphide ores, the only physical inconvenience suffered has been an occasional carbon monoxide headache. C. A. HANSEN.

General Electric Company, Research Laboratory, Schenectady, N. Y.

The Origin of Hydrogen in Blast-Furnace Gas.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In the July issue, 1910, of this journal the authors published an article entitled "A Quantitative Investigation of the Source of Hydrogen in Blast-Furnace Gas." Considerable interest was manifested in the contents of this paper, as it was demonstrated, from the basis of the authors' experiments, that the source of only about half of the hydrogen in the furnace gas could be identified. It will be recalled that the authors suggested that the remainder of the hydrogen in the exit gas might be due to the reaction of hot carbon with combined water in the burden, previously unexpelled by heat.

The above-mentioned article has been reviewed by H. Le Chatelier in the *Revue de Metallurgie* of October, 1910. The main feature of his paper is an explanation of the source of the remaining hydrogen, which he maintains is very simple. He states that the decomposition of water may be readily effected by carbon monoxide in the hot top gases according to the following reaction: $CO + H_2O = CO_2 + H_2$. Le Chatelier states that this reaction begins at 200° C. and occurs to a considerable extent at 300° C., the temperature of the top gas in the authors' experiments, and proceeds to show that at this temperature the entire water content of the gas must be dissociated if the gases are in equilibrium. He admits, however, that in this experiment only half the entire water introduced into the top of the furnace is dissociated, but concludes by stating that the amount would be greater the higher the temperature of the furnace top and the longer the gases remain at this temperature in passing through the mains to the gas-cleaning plant.

The writers have referred to the well-known work of Neuman and Pistor[1] and from their published statements, or from similar work by other authors, fail to confirm the figures in Le Chatelier's article. Neuman and Pistor passed steam and carbon monoxide through a porcelain tube heated to different temperatures and determined the carbon dioxide formed Their results are as follows.

Temp. in Degrees C.	Per Cent CO Converted into CO_2
560	trace
600	2 per cent
900	8 per cent
954	10.5 per cent

These results would show that but little hydrogen would be formed, according to the previously mentioned reaction, below 600° C. But hydrogen may be liberated from water by carbon at a lower temperature. According to Ledebur[2] the decomposition of water by solid carbon begins at 500° C., the product at that temperature being mostly carbon dioxide. From these results it would seem that water would be decomposed by solid carbon rather than by carbon monoxide, since the former reaction sensibly occurs at 500° C. and the latter not until 600° C. is reached.

It would also seem that no reaction could occur in the extreme top of the furnace or in the mains leading from the furnace. The average temperature of the top gas during the experimental work conducted by the authors was only 290° C. The moisture in the gas was not determined in this experiment. Many tests, however, showed the moisture content to vary from 7 per cent to 15 per cent by volume, the hydrogen being at the same time from 4 per cent to 5 per cent.

The recognized complexity of reactions in a blast furnace must cause the solution of the present problem to be difficult. (Even the conditions at the very top of the stock descending into the furnace may be subjected to a sudden change due to slipping.) The writers have been interested in this review by Le Chatelier, but do not think his statement is well substantiated by other published evidence obtainable.

Carnegie Steel Co., Duquesne, Pa. R. J. WYSOR and W. D. BROWN.

Electrode Construction for Furnaces.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—Your January issue, page 42, contains a short article by Dr. Mahlke on a new form of electrode which may involve some features of interest. Permit me to call attention, however, to the prevalent misconception, which I find also exists in this article, that a part of the furnace in which the current density is greater will necessarily be heated to a higher temperature. This is unquestionably an error, even though it has long been supposed to be true. I had called attention to it in some publications on the losses in electrodes.

It was there shown that the stable temperature which any part of the furnace will reach depends not only upon the heat set free in it, but also, and quite decidedly so, on the relative rate at which the heat thus generated internally, or reaching it from other places, will leave that particular part. It was shown that it was quite possible and practicable to use small electrodes with a very high current density, provided they are made so short that the heat will be led off as rapidly as it is formed, and in such a case the temperature will not rise higher than with small current densities.

It would be quite possible, therefore, that the polar enlargements proposed by Dr. Mahlke (and already existing in slightly different forms in other furnaces) could be heated to a higher temperature than the end of the electrode in contact with them, even though the cross-section of the enlargement is much greater than that of the electrode. All that is necessary is that the proportions be such that up to this enlargement all the heat generated in the electrode, and no more and no less, flows out at the cold end. The heat generated in the enlargements then cannot flow out through the electrode, and if it cannot

[1] *Berichte der Deutschen Chemischen Gesellschaft*, XVIII, p. 2894.

[2] "Gasfeuerungen," p. 69.

flow out into any other part as rapidly as it is formed it will accumulate and raise the temperature. Of course, when its temperature gets higher than that of the electrode then there will be a slightly increased heat flow through the electrode, whereby the portion of the enlargement immediately surrounding the hot end of the electrode proper will be slightly chilled by the electrode. It would therefore be possible (whether it would be practicable or not cannot be said off hand) to make this enlarged terminal very great in bulk and actually use it to generate much of the furnace heat, as the heat generated in it could be made to flow into the furnace product lying between these enlarged electrode terminals; the resistance of the product itself might then even be negligibly small. This, therefore, illustrates again the fallacy of our former conception that the temperature must be greater the higher the current density.

Concerning the practicability of his arrangement it must not be forgotten that the electrode generally burns away and must therefore be fed into the furnace; hence anything but a uniform cross-section would in many cases be impracticable. To use the enlargements at the end of electrodes in arc furnaces seems to me to be of theoretical rather than practical interest. It is quite true that the electrode losses in arc furnaces are greatly increased by the fact that the hot end generally has a very much higher temperature than the treatment of the product requires, as, for instance, in iron furnaces; yet to get over this difficulty by the proposed enlargements would probably introduce insurmountable practical difficulties; the furnace, for instance, would have to be opened to replace such an enlargement, not to mention the difficulties in screwing a piece of carbon with a carbon screw to the rounded end of a white hot electrode, from which the stump of the old screw must first be taken out without injuring the thread, all of which seems to be a practical impossibility.

Dr. Mahlke begins his article by saying that this point had been "overlooked" in recent articles on electrode losses. As far as I am concerned I certainly did not overlook it, but in fact even called attention to such enlargements and the possibility of higher temperature in them by referring to the pole plates in the Roechling-Rodenhauser modification of the induction furnace, although I have since then learned that these plates are not counted upon as heat producers.

Philadelphia, Pa. CARL HERING.

Technical Training.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—Co-operation, the slogan for the advancement of the age, hand in hand with specialization, has never before in history been carried on to so great an extent. In the commercial world specialists in each particular line work together toward an ultimate end. The same can be said of the educational world, especially as pertaining to higher education, yet, further, it has been extended to the high schools and, in a moderate degree to the grades. In fact, it has become a recognized principle. When, because of this spirit, such remarkable strides in progress have been made during the last century in each separate field, why, then, arises the question, have they not joined forces? This can practically be accounted for by a certain amount of suspicion with which a member of one fraternity looks upon a member of another. This, of course, has for a basis a slight foundation in that the scientific man's knowledge as to practical things, so thinks the practical man, is about on a par with the farmer's opinion of the city chap and vice versa.

However, the practical man, who has had a practical education and has not been afraid to soil his hands by beginning at the bottom of the ladder, and thus getting a working knowledge of affairs, is the nearest to a combination of the two which we have in this age. Yet, even this man, unless his training has been such as to form in him the student habit, will very soon become so engrossed in the immediate work at hand that he will fail to realize the advance that science is making, and that he is falling behind the procession. No matter how great a success he may be making he may be greater. He must broaden himself. He must know more of the scientific things pertaining not alone to his but as well to allied lines.

The strictly scientific has its place and a very important one in both the advancement of pure and applied science, and where can it be better worked out than in our educational institutions? In fact, the more research, the better, without neglect of the other duties of the instructor, is getting to be a well-recognized principle in the higher educational world. The main reason that more of this is not done is that an inadequate salary occasions the necessity of outside work in order to make both ends meet. Secondly, he is too lazy. Thirdly, he does not have the requisite brain power. When asked why he did not do more in his line I once heard the head of a department in a State university reply, "that it was all worked out and there was nothing more to do." The work for this type of man is digging post holes where there will always be plenty left to dig.

There is rarely found combined in a person the power of original thought, especially when it is logical, and the ability to take a given problem and carry through an investigation to its logical end, whether that end be positive or negative. Generally speaking, there is but one place where it is possible for men to get this training at the present time and this is in the graduate schools of our various educational institutions. It is very seldom that a man who has finished the ordinary four years of any one of our numerous scientific or technical courses in either a college or university has this faculty, at least to any degree.

This may be accounted for, to a certain extent at least, by the fact that the first thing which appeals to the student in engineering and most of the applied sciences is whether or not the particular subject in hand will be of any practical value, i. e., he is looking to his college education as a means to an end and that, in many cases, being the making of a living for himself and later for his family. Now, the courses, for the most part, were planned by men of vast experience who were able to see and know what were the subjects best suited for a man who wishes to prepare himself for a particular profession. Where it is equally possible to appeal to this practical instinct of the student, thus holding his interest in the work at hand without detriment to his training, the easier and further can he be made to progress in a given time.

A plan to bring about this sort of a thing in such a way that it would be possible for the student engaged in obtaining an engineering education to come in contact with and observe men who had obtained the training which he is striving for is by carrying on research of direct practical application. There should be a corps of instructors who are able not only to carry on industrial research themselves and to impart their knowledge to the student, but who can likewise interest men outside who are engaged in technical work to bring their problems to the university or technical school for solution. There they can either avail themselves of the facilities offered by the institution or establish a fellowship for their particular problem similar to those which have been established at the University of Kansas or the University of Pittsburgh.

There will be thus benefited—

(1) The student, who, kept at work by the constant force of inspiration, will learn by association to appreciate and strive to obtain the power of experimentally carrying on a logical line of thought to the desired end. (2) The instructor, who will come in contact with the practical application of scientific ideas, keep pace with the times and thus avoid fossilization. (3) The practical man, by including in his environment men possessed of a fund of scientific knowledge along lines in which he is quite likely to be rusty. Access to a library will do away with useless duplication and he will keep up to date along the line in which he is most interested.

University of Arizona,
Tucson. R. C. BENNER.

Presentation of the Perkin Medal to Dr. Charles Martin Hall.

At the meeting of the New York Section of the American Electrochemical Society, held in the Chemists Club in New York on Jan. 20, 1911, the Perkin medal was formally presented to Dr. Charles M. Hall.

The meeting was well attended, especially by members of the electrotechnical fraternity, who had come to honor Mr. Hall, many of them from outside of New York, Niagara Falls being, of course, well represented. A pretty touch of friendly sentiment and an international flavor were given to the proceedings by the presence and speech of Dr. Paul Héroult, of France.

The Perkin Medal.

The chairman, Dr. CHARLES F. MCKENNA, made the following introductory remarks which we give in abstract:

"The foundation of the Perkin medal was a happy act of scientific benevolence. Perkin's contemporaries in America, wishing to do something which would honor him in life and continue to keep his memory illustrious in science even after he should have passed away, chose to establish this medal and to see to it that in recurring years it should go as a reward to the ones who had struggled, striven and fought and forced nature to yield up further favors to men.

"Sir William Perkin has passed away and now after some further years in which we have witnessed the awarding of this honor we find the following result:

"Each year the achievement of Sir William himself stands out bolder on the pages of chemical history, each year the new award of this medal adds a link to the chain connecting the great work of industrial chemical advancement in America with the earliest and most pronounced advance made in England when aniline yielded up mauve. Each year the appreciation of the honor grows and the Perkin medal with its annual award is coming to be looked upon as the 'Concours' of chemical industry.

"And the thoughtful men who in founding this benevolence, drew up its guarantees and conditions, builded carefully and perhaps better than they knew. Besides leaving the selection of the recipient open to nomination by a large group of American societies interested in chemistry, they set no restriction as to the period in which the discovery may have been made. It is not necessary that the special work shall be that of the preceding year, or of any immediately preceding year. This leads to careful judgment and brings forward for the palm many a struggling chemist who in years past had known the disadvantages of inconspicuousness and the hardships of passing along the dusty road of conflicting interests.

"It happened that the importance of Sir William's own discovery was immediately grasped and that it traveled quickly throughout the world. But this is not always so in our chemical discoveries, and I think it can be said that the full appreciation of the significance of an American college student's experiments in electrolyzing a bath of alumina and kryolite was long in dawning.

"Therefore we must rejoice that the Perkin medal gives the chemical fraternity the opportunity to honor one of our confrères whose modesty has been as singular as his patience has been marked."

CHARLES M. HALL.

Dr. McKenna then introduced the doyen of the chemical profession in America, Dr. Charles F. Chandler, who made the formal presentation speech.

Address of Presentation.

Dr C. F. CHANDLER spoke in part as follows:

"It is my privilege and very pleasant duty as the senior past president of the Society of Chemical Industry to present to Charles Martin Hall, LL.D., the fifth impression of the Perkin medal in recognition of his most valuable work in applied chemistry.

"This work relates particularly to his discoveries and inventions in connection with aluminium. This is a subject which has interested me deeply from my student days. In 1854 I listened to Woehler's account of his discovery of aluminium in 1827, and the following year (1855) I was shown by Rose aluminium which he extracted from cryolite in the spring of that year. In 1869 I had the pleasure of making the acquaintance of Henri St. Claire Deville, who in 1854 and the following years prepared aluminium on a manufacturing scale. Later I had among my pupils at the School of Mines my warm friend Hamilton Y. Castner, who for two years monopolized the manufacture of aluminium by means of his cheap sodium, and finally for the past twenty years I have been very intimate with Dr. Hall, whose electrolytic process at Niagara Falls has produced the metal in such enormous quantities and at such a low price that it has monopolized our market and taken the place for innumerable purposes of other metals formerly employed.

"The name 'aluminium' is derived from alumen, which included alum, in which Margraf in 1754 recognized a peculiar earth which we call alumina and which was supposed a century ago to contain a metal. Sir Humphrey Davy, who in 1807 succeeded in extracting sodium from soda and potassium from potash, and a little later the metals of lime, magnesia, glucina and strontia, endeavored to reduce alumina by heating with the aid of 1000 galvanic plates an iron wire in contact with alumina in an atmosphere of hydrogen. The iron fused, became whiter, and on solution in acid yielded the earth alumina. He evidently produced an alloy of the metals. Nothing further was done till 1827, when Woehler reduced aluminium chloride with potassium and obtained aluminium as a gray, metallic powder, and later in 1845 by acting on aluminium chloride with potassium vapor obtained the metal in tin-white globules.

"In 1854 Henri St. Claire Deville became imbued with the idea of making aluminium a useful metal. He secured the interest of the Emperor Louis Napoleon, and with liberal grants of money was able to work on a factory scale. He employed aluminium chloride and sodium, introduced many improvements in processes for their preparation and application, and manufactured the metal in what for those days seemed very large quantities. Among other improvements he introduced cryolite as a flux. This mineral from Greenland was used experimentally to produce aluminium by the action of sodium in 1855 by both Rose and Percy. Deville's factories continued to produce aluminium on a moderate scale till about 1888, reaching a maximum output of over 5000 lb. per year along in the middle eighties. A few other manufacturers established works. The process of manufacture was, however, so expensive that aluminium was really a precious metal, used chiefly for fancy articles.

The price of metallic aluminium varied before 1890 as follows:

1855	$90	lb.
1856	27	"
1857	22.50	"
1862	12	"
1886	12	"

"In 1869 I purchased a little basket in Paris at the aluminum salesroom for $9. It weighed 4 oz.

"In 1886 Hamilton Y. Castner invented a new process for making sodium. He employed caustic soda in place of the carbonate and iron carbide in place of carbon as a reducing agent. As a result he reduced the cost of sodium from $1 per pound to 25 cents, a very important item, as it was never possible to make a pound of aluminium with less than 3 lb. of sodium. By June, 1888, his works at Oldbury, near Birmingham, were in working order, with a daily output of 250 lb., sold at $5 per pound. In 1889 the daily output was 500 lb. at $4 a pound.

"And then the manufacture of aluminium was discontinued because the new electrolytic manufacture from alumina produced the metal at $1 per pound.

"We now come to the achievements of Charles M. Hall, but before I go any further I wish to say that whatever praise I shall bestow on this great inventor I do not in the slightest degree disparage that other great inventor, Paul L. Y. Héroult, who worked on the same problem at the same time, across the Atlantic, made the same discovery independently and worked out an analogous process which has played the same important part in Europe that Hall's process has played in America.

"Charles Martin Hall was born in Ohio, Dec. 6, 1863, was educated in Oberlin and graduated there in June, 1885, when 21 years and 6 months of age.

"He studied chemistry while at school, became interested in the aluminium problem while in college and made many experiments in the hope of finding a more economical process for its production. Finally his thoughts turned to electrolysis, and about Oct. 1, 1885, he began to seek a suitable anhydrous solvent for alumina, operative at a practicable temperature, which would bring the alumina into a fluid condition in which it would yield to the electric current.

"Fluorspar and the fluorides of magnesium, sodium, potassium and aluminium were successively tried, but were found to be too infusible and to dissolve little, if any, alumina.

"On Feb. 10, 1886, he tried cryolite, the double fluoride of sodium and aluminium. This fused readily at a very moderate temperature and when he dropped in the white powder of alumina it dissolved and disappeared as does powdered sugar or salt in boiling water. It dissolved to the extent of more than 25 per cent and the solution was clear and limpid.

"He thus made his first great discovery. He proceeded at once to apply the electric current to this solution. His first experiments with clay crucibles were not successful; but on Feb. 23, 1886, he employed a carbon-lined crucible and was successful in obtaining aluminium. He had achieved success in inventing a process for making aluminium directly from alumina by electrolysis under conditions which promised to revolutionize the industry and furnish the metal at so low a price as to enable it to replace copper, tin, lead, zinc and other metals for an endless variety of purposes.

"Hall's age at this time was two months over 22 years and he had discovered and invented what had escaped Wochler, Rose, Deville and many of the other world-renowned chemists who had busied themselves with aluminium over a period of half a century.

"On July 9, 1886, Hall applied for his broad basic patent which was granted April 2, 1889, No. 400,766, with the following claims:

"'1. As an improvement in the art of manufacturing aluminium the herein-described process, which consists in dissolving alumina in a fused bath composed of the fluorides of aluminium and a metal more electro-positive than aluminium, and then passing an electric current through the fused mass substantially as set forth.

"'2. As an improvement in the art of manufacturing aluminium the herein described process, which consists in dissolving alumina in a fused bath composed of the fluorides of aluminium and sodium and then passing an electric current, by means of a carbonaceous anode, through the fused mass, substantially as set forth.

"'3. As an improvement in the art of manufacturing aluminium, the herein described process, which consists in dissolving alumina in a fused bath composed of the fluorides of aluminium, sodium and lithium, and then passing an electric current, by means of a carbonaceous anode, through the fused mass, substantially as set forth.'

"He also received four additional patents for modifications of his original process. Hall did not escape the usual experience of the successful inventor. He was obliged to defend in court the originality of his invention. The suit was brought in the United States Circuit Court of the Northern District of Ohio and after an exhaustive discussion of the prior art the patent was sustained. The opinion was written by Judge William Howard Taft, now the President of the United States. He wrote: 'Hall was a pioneer and is entitled to the advantages which that fact gives him in the patent laws.'

"The court declared the patent valid and issued a perpetual injunction against the defendants. For various reasons there was considerable delay in devising suitable means for applying the invention on a manufacturing scale and securing the necessary capital. But at last, in November, 1888, he began work in his factory at Kensington, near Pittsburgh, producing 50 lb. of metal a day. Since that date his progress has been wonderful. He soon erected larger works at Niagara Falls as the first consumer of the electricity supplied by the Niagara Falls Power Company. At the present time the company has three establishments at Niagara Falls, one at Massena in New York on the St. Lawrence and one at Shawinigan Falls in Canada. The total consumption of electricity is 140,000 hp, probably a larger amount than is consumed by any other electrochemical works in the world. The annual output of aluminium is now 40,000,000 lb. The first metal produced at Kensington was sold at $2 per pound; the price now is about 22 cents.

"The early dreams of the young undergraduate have certainly been realized.

"Hall has made many other inventions, several of which have been patented. Six of these are for improved methods for preparing pure alumina, one of which is still employed for preparing this material for use in the works. He has also patented improved methods for making improved carbon anodes for use in the works. There are very many other matters in connection with Hall's work to which I should be glad to call your attention did time permit. I think, however, that I have laid before you a most creditable record of discovery to satisfy you that your committee has been wise and just in awarding to Dr. Hall the gold medal which was established to honor forever the memory of the great chemist, Sir William Henry Perkin, and by association with him to honor our own most distinguished fellow chemists.

"Dr. Hall's achievements certainly entitle him to a place in the front rank of electrochemists and his youthful successes must ever prove an encouragement to other young men who develop a love for this branch of applied science. Personally Dr. Hall is a man of the greatest modesty and lovable character."

In turning to Dr. Hall, Dr. Chandler concluded: "Charles Martin Hall, Doctor of Laws, my dear friend, it gives me the greatest pleasure as the representative of the affiliated chemical and electrochemical societies to place in your hands this beautiful token of appreciation and affection of your fellow chemists."

Personal Reminiscences of Dr. Hall.

Dr. Charles M. Hall expressed in a few graceful sentences his thanks for the medal. "I am glad to have this medal on

account of what it represents and on account of the manner in which it has been awarded, and I shall treasure it among my most valued possessions."

Dr. Hall then gave an account of some personal and unpublished facts in connection with his work on aluminium.

"My first knowledge of chemistry was gained as a school boy at Oberlin, Ohio, from reading a book on chemistry which my father studied in college in the forties. I still have the book, published in 1841. It is minus the cover and the title page, so I do not know the author. It may be interesting now to see what this book, published 70 years ago, says about aluminium. It states: 'The metal may be obtained by heating chloride of aluminium with potassium in a covered platinum or porcelain crucible and dissolving out the salt with water. As thus prepared it is a gray powder similar to platinum, but when rubbed in a mortar exhibits distinctly metallic luster. Fuses at a higher temperature than cast iron and in this state is a conductor of electricity but a non-conductor when cold.'

"Later I read about DeVille's work in France and found the statement that every clay bank was a mine of aluminium and that the metal was as costly as silver. I soon after this began to think of processes for making aluminium cheaply. I remember that my first experiment was to try to reduce aluminium from clay by means of carbon at a high temperature. I made a mixture of clay with carbon and ignited it in a mixture of charcoal with chlorate of potassium. It is needless to say that no aluminium was produced. I then thought of cheapening the chloride of aluminium then used as the basis for aluminium manufacture and tried to make it by heating chloride of calcium and also magnesium chloride with clay, following the analogy by which iron chloride is produced when common salt is thrown into a porcelain kiln. A little later I worked with pure alumina and tried to find some catalytic agent which would make it possible to reduce alumina with carbon at a high temperature. I tried mixtures of alumina and carbon with barium salts, and also the same mixture with cryolite and carbonate of soda, hoping to get a double reaction by which the final result would be aluminium. I remember buying some metallic sodium and trying to reduce cryolite, but obtained very poor results. I made some aluminium sulphide, but found it very unpromising as a source of aluminium then as it has been ever since.

"On a later occasion I tried to electrolyze a solution of aluminium salt in water, but found nothing but a deposit of alumina on the negative electrode. I did not give a great deal of time to these experiments, as I was a student in college at the time and worked on three or four other attempted inventions. I had studied something of thermochemistry and gradually the idea formed itself in my mind that if I could get a solution of alumina in something which contained no water and in which the solvent was chemically more stable than the alumina, that this would probably give a bath from which aluminium could be easily obtained by electrolysis. The idea remained in my mind and in February, 1886, I began to experiment on this plan. The first thing in which I tried to dissolve alumina for electrolysis was fluorspar, but I found that its fusing point was so high as to be rather discouraging. I next made some magnesium fluoride, but found this also to have a rather high fusing point. I then took some cryolite and found that it melted easily and melted dissolved alumina in large proportion. Then I rigged up a little electric battery—mostly borrowed from my professor of chemistry, Professor Jewett, at Oberlin College, where I had graduated the previous summer of 1885. I melted some cryolite in a clay crucible and dissolved some alumina in it and passed an electric current through it for about two hours by means of carbon electrodes. When I poured out the melted mass I found no aluminium. It then occurred to me that the operation was interfered with by impurities, principally silica dissolved from the clay crucible, so I next made a carbon crucible, inclosed it in a clay crucible and repeated this experiment. This time, after passing the current for a couple of hours, I poured out the material and found a number of small globules of aluminium. I was then quite sure that I had discovered the process that I was after.

"I then undertook to broaden and improve the process and found that I could use, instead of cryolite, other double fluorides, particularly a double fluoride of potassium and aluminium. The most important change, however, which I made at this time was in the material used as an anode. I wanted to get rid of the burning up of the carbon anodes. I tried a platinum anode and found that it seemed to work all right, but it was too expensive. Then I discovered that if I used a fusible bath of a potassium double fluoride with a sodium double fluoride I could use a copper anode, which immediately became coated with a thin film of copper oxide and acted like a permanent platinum anode. This was not a step in advance as I thought it was, as more or less copper gets into the reduced aluminium and the use of a copper anode led me to use very fusible baths which on the whole do not work as well as the less fusible baths. It is probable that this change delayed a successful result for a year or two.

"When worked on a small scale this process with any of the baths I have mentioned, and with either copper or carbon anodes, is not apparently promising. The ampere-hour efficiency is low, sometimes zero, and the bath, whether composed of sodium or potassium salt, becomes filled with a black substance which accumulates and renders the process very difficult. I presume that my friend, Dr. Héroult, whom I have the pleasure of seeing here to-night, who invented the process independently in France about the same time as I did, encountered the same difficulties I did. In spite of the difficulties mentioned, however, I had great faith in the theoretical possibilities of the process and believed that the practical difficulties could be overcome, so I stuck to it from the start.

"On the financial side I had in the course of three years three different sets of partners or backers, of whom two sets became discouraged and gave up. In the summer and fall of 1886 I worked with some people in Boston with whom my brother had made some financial arrangement. The results there obtained were not satisfactory to them and in October, 1886, my Boston friends declined to go further and I returned to my home in Oberlin, Ohio. In December, 1886, I found that a bath composed of a very fusible double fluoride of aluminium and potassium, with copper anodes, worked much better than anything I had before tried. I have here a number of buttons of aluminium made by this method at this time. The larger one was made with current from a galvanic battery on Dec. 7, 1886, and weighs about eight grams. After this I negotiated with the head of one of the large chemical manufacturing companies of the United States with headquarters at Cleveland, but we could not finally agree. I thought then, and have thought since, that this gentleman was the kind of man whom the proverbial inventor meets when he is deprived of the fruits of his labor. He was extremely careful and conservative and was always wanting to take a few days more to think the matter over. He kept me on the string six months and the kind of contract which he finally offered seemed to me too entirely one-sided. The gentleman told a friend of mine afterward that I was no business man, which was no doubt true, but I received what seemed to me much fairer treatment from all with whom I afterward had business dealings and found much more liberal associates and friends.

"The Cowles Electric Smelting & Aluminium Company, who were then making aluminium alloys at Lockport, N. Y., were the second set of people who became interested in my invention. They took an option on it and I spent a year with them, from July, 1887, to July, 1888. They finally threw up their option. The baths which I used at Lockport, at first, worked well for a few days, but after a time they would become less efficient. I finally worked out a system by which the difficulties were overcome. This was by making a bath consisting partly of calcium fluoride or fluorspar and adding 3 per cent or 4 per cent of calcium chloride and using carbon anodes. I reasoned that

chlorine was evolved and burned up the objectionable compound which spoiled the bath. After finally overcoming the difficulties which I have mentioned I made several pounds of aluminium in small crucibles which I showed to Mr. Alfred Cowles and gave him all the facts in relation to the same, but he was not interested.

"I then sent a representative—Mr. Romaine Cole, now dead—to Pittsburgh, who got together the gentlemen who formed the Pittsburg Reduction Company, now the Aluminium Company of America, and we immediately started in the summer of 1888 to build and operate a commercial plant on Smallman Street in Pittsburgh. We had at our disposal about 50 hp in electrical current of 2000 amp. It took a few weeks after starting to get the dimensions of our baths and so forth just right, and then the difficulties which I have referred to disappeared as if by magic. The clogging and spoiling of the bath, which had caused trouble for the last three years, did not occur on a large scale. No calcium chloride was required. It seems that this is a process, unlike a good many others, which works badly on a small scale and well on a large scale. I accounted for this by the fact that on a large scale the electrodes are further separated and there is less circulation between the positive and negative electrodes, which lowers the efficiency and favors the formation of the clogging black compound.

"We also found, as I had predicted nearly three years before and stated in a patent application filed two years before, that on a commercial scale no external heat was required to keep our baths in fusion. This was a great advantage, but I believe that it resulted from a law of nature and not from any invention at the time, as we did not use any excess of current for maintaining fusion, but only the normal current and voltage for electrolytic purposes, and the use of the electric current for fusing and heating in connection with electrolysis was a thing which had been disclosed and published almost a century before.

"The business of manufacturing aluminium has now grown to a great commercial business. Many workers have contributed to it and the credit is to be divided among many. Our financial people, particularly the gentlemen of the Mellon Bank of Pittsburgh, have given indispensable aid in purchasing our stock and in loaning us money most generously. Our patent attorneys and experts, our engineering and chemical staff, sales agents, superintendents, foremen and others have all done their share."

"In the commercial development of the business," Dr. Hall concluded, "the greatest credit is due to the late Captain Alfred E. Hunt and to Mr. Arthur V. Davis."

Personal Reminiscenes of Dr. Héroult.

Dr. Paul Héroult then made the following address:

"There is a verse of Kipling which reads as follows:

> "'Ship me somewhere east of Suez,
> Where the best is like the worst;
> And there ain't no ten commandments,
> And a man can raise a thirst.'

"I was in that particular mood when I made my first acquaintance with aluminium in large quantities. At the time I had a friend who since then became my partner, but for the time being we were both dead broke. We had pawned everything in sight and also other things which were not in sight.

"Finally my partner had a bright idea. He brought back from home a stick of aluminium about 6 in. long which was valued very high in his family as a personal souvenir of Saint Claire Deville.

"As we handed it to the pawnbroker, the latter said: 'What is that, bar silver?' We said: 'Better than that; that is aluminium.' 'Aluminium,' he said, 'what is that?'

"He weighed it in his hand and said: 'Why, is this hollow?' We said: 'No, that is aluminium and it is worth 120 francs per kilo.' After some thought he said: 'Well, I will give you 2 francs for it,' say 40 cents in American money.

"On a hot summer's day it was better than nothing, and we took the money with the firm intention of buying the stick back, which we never did. Maybe that was one of the reasons why, later on, I had to make good and replace it.

"No material change took place in the industry of aluminium for three years after that event, but the actual fact of the development of said industry was preceded by rumors to the effect that something new was coming.

"I can remember there was a shop on the Grand Boulevard in Paris where there was a display of aluminium goods, mostly thimbles, if I remember rightly; besides wedding rings, brooches, statuettes, a few coils of wire and quite some interesting specimens of aluminium bronze in the form of coffee pots, sugar bowls, etc.; none of the cheap stuff which is sold nowadays. I bought once a thimble for my mother out of my savings. I was a good boy in those days. Every one of those articles had on the Paris finish and sold like silverware.

"In the nineties this same shop practised the same prices, although the cost for ingots had come as low as 25 francs per kilo.

"This was a wonderful time for the aluminium makers; they were making money hand over fist, at least on the books. There was no use for any syndicate at that time. Aluminium was being piled up in the store and the stock of finished product reckoned at above-named price. But this did not last long; customers were scarce and finally the bankers got tired of lending money to people who piled up stock; it became necessary to sell.

"Another price was established at 8 francs per kilo which did not prove successful either. Then came the 5 marks which originated with the Swiss Company; finally it seemed to open the bunghole of the cask.

"Now after various vicissitudes the price in the old country looks pretty near settled around 20 cents per pound and will vary probably according to the price of copper.

"Twenty-five years ago the aluminium industry employed about 10 men; the production was 1½ tons a year, the total value of which was $30,000. Nowadays the same industry produces 50,000 tons at a value of $20,000,000 and employs from 15,000 to 20,000 men.

"If we take the average consumption of copper at 900,000 tons a year at a value of 13 cents per pound, the total value of ingot metal will be $234,000,000 a year comparing with the $20,000,000 for aluminium, say one-tenth. But 13 cents is pretty near the rock bottom price for copper. At 20 cents per pound aluminium presents a great advantage for most uses.

"Aluminium is slowly wedging its way in the metal market. It looks probable that in a course of 10 or 15 years the consumption of the new metal will be equal to that of copper and that after the golden age, the stone age, the bronze age and the iron age we will have the aluminium age.

"My friend Hall and myself have been fighting for 15 years, most of the shots going wild on account of the long reach over the Atlantic.

"Since we met, however, we conceived a better opinion of each other and I take great pleasure to-night in extending to my friend Hall my sincere congratulations on the award to him of the Perkin medal."

Concluding Speeches.

Dr. Jos. W. Richards then gave some reminiscences of his friendship of many years with Dr. Hall. He emphasized that Hall had taken at Oberlin College the classical, not the scientific, course. It seems as though a broad foundation such as given by a classical course is a very suitable basis for later technical work.

Dr. Richards then gave a brief review of the various uses of aluminium. First, in the steel industry as a deoxidizer. Second, as an electric conductor. Third, in the reduction of refractory oxides to carbon-free metals and in welding by the aluminothermic method of Dr. Hans Goldschmidt. Fourth, for culinary utensils aluminium seems to be the ideal material; it is practically indestructible and its use for this purpose should very largely increase. Dr. Richards also referred briefly to

the use of aluminium in the chemical industries, to the use of stiffened aluminium for automobile and airship construction, to the use of aluminium bronze, etc. There was quite an interesting collection of various aluminium products on exhibition.

Dr. Richards concluded that the invention of aluminium ranked with the invention of the Bessemer steel process as the most important technical achievement of the nineteenth century.

In a few graceful words congratulations were expressed to Dr. Hall by Dr. Wm. H. Walker for the American Electrochemical Society and by Dr. Chas. Baskerville for the American Chemical Society.

After the formal meeting was over there was an informal session of the Verein Deutscher Chemiker.

Nitric Acid and Nitrates from Air.

At the recent Königsberg meeting of German Naturforscher und Aerzte, Dr. J. ZENNECK gave an interesting outline of the different processes for the fixation of atmospheric nitrogen by electric discharges through air. The paper is printed in full in *Phys. Zeit.*, Oct. 15, 1910. (See also p. 99 of this issue.)

Fig. 1 shows the arrangement of the principal apparatus for demonstration purposes. The air first enters a blower (in practice a powerful turbo-compressor is used which supplies some 20,000 cu. m of air per hour to the plant).

The air then enters the "furnace" in which a small portion

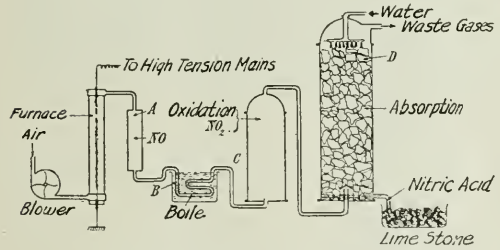


FIG. 1—APPARATUS FOR DEMONSTRATING THE MANUFACTURE OF CALCIUM NITRATE FROM ATMOSPHERIC NITROGEN.

of the nitrogen and oxygen combine to NO. The colorless gas passes off through the glass tube *A* into the boiler *B*, where the heat in the gas is utilized. In the model for demonstration purposes it is simply a water tank with copper tubes. In practice the apparatus has the same form as any large boiler. The steam produced in it is utilized for heating and evaporating.

After the gas has cooled off it has an opportunity of rest in a large vessel, the "oxidation chamber," in which the NO combines with free oxygen in the air to form NO_2.

The gas then enters a glass cylinder *D* filled with coke particles, through which water runs down, absorbing the NO_2 and forming nitric acid. In practice the nitric acid formation is carried out in very large towers.

The formed nitric acid is finally passed off into a receptacle filled with limestone for the formation of calcium nitrate.

The essential difference of the different commercial processes is to be found in details of the furnace construction. But in all cases it is an arc furnace. Convenient electrode holders for high-tension and arc experiments are shown in Fig. 2, *a* and *b*.

To get as much air as possible in contact with the arc the Pauling Brothers employ an arrangement similar to the horn lightning arrester, Fig. 3. (Details of the Pauling process, as carried out in a commercial plant at Innsbruck, Tyrol, were given in our October issue, 1909, Vol. VII, page 430.)*

Birkeland and Eyde deflect the arc, which is produced between two horizontal electrodes by magnetic means so as to form a flaming disk. The form of their furnaces is shown in Fig. 4. (See also our Vol. II, page 399; Vol. IV, pages 126, 295 and 360; Vol. VII, p. 304 and 305; Vol. VIII, p. 451.)

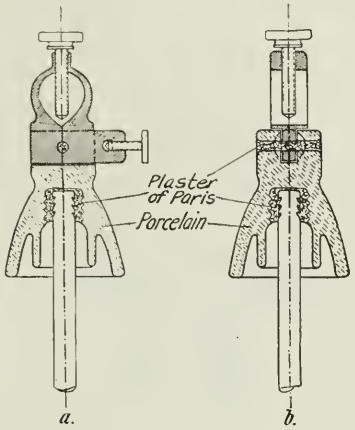


FIG. 2—ELECTRODE HOLDERS.

Petersson, Brion, and Moscicki employ a similar principle. One electrode is in form of a rod, the other is a coaxial cylinder. In the annular space between the two the arc is playing and is caused to run around continually by means of a magnetic field, the lines of flux of which have approximately the direction of the axle of the two electrodes. This is shown in Fig. 5. (See also our Vol. V, page 492.) The furnace of Moscicki

FIG. 3—PRINCIPLE OF PAULING APPARATUS.

has been further developed in conjunction with the Aluminium Industrie Gesellschaft in Neuhausen.

The furnace of the Badische Company is shown in Fig. 6. The long arc plays in the central tube and the air passes around it in a spiral path from the bottom of the tube to the top. (For further details see our Vol. VII, page 245.)

*In connection with this and the following processes the article by Lamy and Lamb on page 99 should be compared.—EDITOR.

This arc is interesting in various respects from the standpoint of the physicists. While long arcs are usually very unstable the arc is in this case maintained in a very quiet state by means of the spiral air current encircling it. Further, it is unusually long, for instance 8 m (26 ft.) at an effective tension of not over 4000 volts. Further, the conditions of pressure

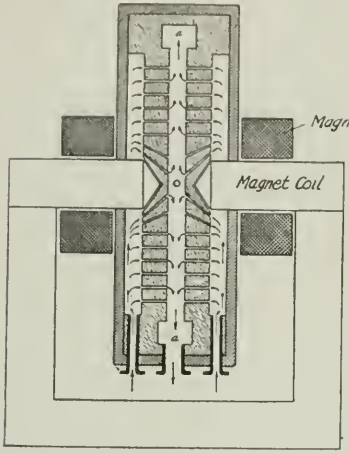


FIG. 4—BIRKELAND-EYDE FURNACE.

and temperature in such an air vortex are peculiar; in the axle of the vortex the pressure is well below atmospheric pressure. Such an air vortex acts somewhat like a centrifuge, the heavier and colder air being pressed to the outside against the walls of the tube. In consequence of this the exceedingly hot internal core with the arc is surrounded by a layer of relatively cold air.

This fact explains why Dr. Zenneck could carry out his experiment in a glass tube, although more than 10 hp were used

FIG. 5.—PICTURE OF MOSCICKI ARC.

so that considerable heat was developed. For the same reason comparatively narrow tubes can be used in the commercial furnaces with their enormous temperatures, and these tubes can be made of iron without being destroyed by the heat.

Further, the success of this type of furnace seems to contradict the hypothesis that the formation of the nitrogen oxide by the arc is a purely thermal phenomenon. According to this hypothesis it is not only necessary to heat the air to an exceedingly high temperature, but also to chill it as suddenly as possible. This latter condition is not very well fulfilled in this type of furnace.

The air to be treated is preheated by the gases leaving the furnace. The path of the air through the furnace is clearly shown in Fig. 6.

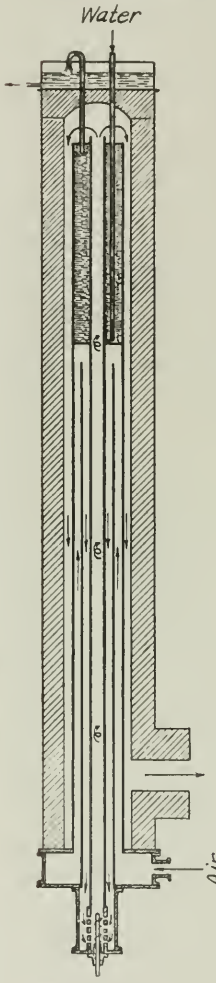


FIG. 6—FURNACE OF BADISCHE COMPANY.

The method of starting the arc is very simple. Through a little hole in the bottom a little rod of wood is introduced wound with wire so as to produce a conducting contact between the lower electrode and the wall of the tube. An arc is formed which is quickly carried upward by means of the air vortex until it is in its normal central position between the lower and upper electrodes.

Dr. Zenneck gave finally a brief outline of the electrical engineering features of the plant of the Badische Company. In the hydroelectric power plant the three phases of the large three-phase alternators are connected to the three transmission cables. A fourth thinner cable connects the neutral point of the alternator with the earth in the furnace house.

The three main transmission cables are connected each to one electrode of one furnace, the other electrode being earthed. But in each of the three main cables a large inductance coil is inserted in series with the arc. This is absolutely necessary, since without the inductance coils there would be danger of the arc being extinguished or the machine burning out. The size of these inductance coils is many times larger than ordinary inductance coils as usually employed.

These inductance coils fulfil the same purpose as a series resistor, but they represent a reactance instead of resistance—that is, they consume very little energy, since the phase difference between e.m.f. and current is very nearly 90 deg.

A novel engineering feature of the installation is the direct use of high-tension currents in the plant. The high-tension currents enter directly the works and right at the furnace electrode there is a tension of several thousand volts. This involves great danger and has necessitated special precautions.

Although the arcs are very quiet and steady once in a while an arc may be extinguished and a load of at least 1000 hp is thus suddenly taken from the generator. In the same way when the arcs are started the generators are suddenly loaded. Starting appliances, such as are used with motors, can be employed only to a small extent and are used only with very large furnaces: several inductance coils are connected in parallel and one is switched in after the other.

Further, the current and voltage conditions in the furnaces are very different from usual electrical engineering practice and great precautions must be taken in the measurements to get exact results. This becomes evident when it is found that two precision instruments, which were both calibrated by the Reichsanstalt and which agree absolutely with each other under ordinary conditions, gave readings different by 4 per cent. Commercial instruments, which are ordinarily exact by 1 per cent or 2 per cent, were found to give readings which were wrong by 12 per cent or more.

Even the connections of the instruments to the line conductor involve difficulties, as the line conductor carries a current of 200 to 300 amp and is at some thousand volts against earth.

Dr. Zenneck emphasizes these difficulties because the fundamentally important determination of the furnace efficiency depends on an exact measurement of the electrical energy. He concludes: "Since the measuring methods mostly employed in engineering fail certainly in this case, all statements in the literature on the efficiency of the different furnace systems

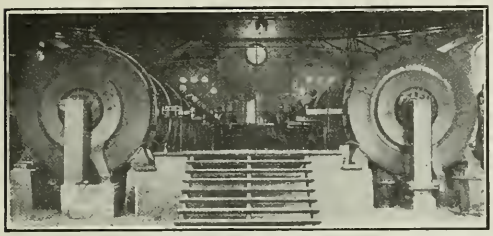


FIG. 7—FURNACE HOUSE, NOTODDEN.

should be regarded with unlimited skepticism if the statements do not prove clearly that the measurement of the electrical energy was exact."

* * *

One special difficulty of the measuring methods is discussed by Dr. J. ZENNECK in an article in *Physikalische Zeitschrift*, Oct. 15. In ordinary engineering practice measuring transformers are widely used, the measuring instruments being connected to the low-voltage terminals. But in connection with long arcs as used in the furnace of the Badische Company the employment of measuring transformers may involve unduly large errors. Dr. Zenneck claims that to be sure of the result it is necessary to insert the current coil of the electrodynamic wattmeter directly into the line conductor and to connect the voltage coil in series with non-inductive resistors across the line.

But this involves the difficulty that as the instrument is charged to a high voltage purely electric (electrostatic) forces are acting on the movable system of the instrument. Errors due to this fact must be eliminated. In the article referred to a simple method is described how to calibrate the instrument so as to eliminate this error once for all.

* * *

The works for the production of calcium nitrate from atmospheric nitrogen at Notodden, Norway, using the Birkeland and Eyde process, and especially the electric power plant, are the subject of an article in *Teknisk Ugeblad*, reproduced in London *Electrical Review*, Dec. 30. The power plant contains four alternators, generating three-phase currents at 10,000 volts, 600 amp per phase, the frequency being 50 cycles per second. All four generators work in parallel on a system of busbars, which are connected with Notodden by three cables. The current is transformed by oil-cooled transformers and then supplied to the furnace, inductance coils being arranged in series.

The furnace house (Fig. 7) is a building with a floor area of about 21,500 sq. ft., constructed entirely of brick and iron. The number of furnaces is 36. "The diameter varies from 6 ft. to 8 ft., but the length is comparatively small. The electrodes are placed in the center of the furnace, diametrically opposite each other, and between them the electric arc is formed. The air is conveyed to the interior of the furnace through the walls of the chamber, which is made of firebricks, and from this chamber the nitrous gases pass through a fireproof channel. The temperature of the chamber is generally about 3500° C., while that of the escaping gases varies from 800° C. to 1000° C. The furnace can be kept going for weeks without interruption; the brickwork lasts from four to six months, while the electrodes have to be charged every three or four weeks.

"The necessary supply of air is obtained by means of centrifugal fans. The gases from the furnaces are collected and conveyed through two large pipes, each with a diameter of 6 ft., to the boiler house, where the heat contained in them is utilized for various purposes.

"From the furnace house the gases pass to a cooling house through large pipes of aluminium, placed in flowing water, and having been sufficiently cooled pass on to vertical iron cylinders, lined with acid-proof stone, where the oxidation takes place."

The information given concerning the absorption of the oxide vapors and production of calcium nitrate is meager. The production at Notodden during the first year amounted to 20,000 tons of nitrate of lime, but when all the projected works have been completed the minimum production will be 100,000 tons per year. The total capital invested when the new buildings and extensions are completed will be about $17,500,000.

Concentration of Platiniferous Copper Ore at the Rambler Mine, Wyoming.

Editorial Correspondence.

Historical.

In the winter of 1901 it was discovered that platinum occurred in the copper ore of the Rambler mine, located in Albany County, Wyoming, about 50 miles southwest of Laramie. The occurrence was first suspected from the difficulty experienced in making satisfactory assays for gold on shipments to the Denver smelters. In the usual gold assay platinum would be collected in the final cupellation, and it is difficult to see how it could have escaped detection for so long a time, for up to the date of actual discovery in 1901 nearly 4000 tons of 25 to 30 per cent copper ore had been shipped from the mine. Of this quantity, about 2500 tons had been shipped to the smelter at Encampment, Wyoming, and smaller tonnages to other plants. The incident is more striking in view of the fact the Rambler Company had never received pay for gold in its ore, as it was presumed that this metal occurred in quantities smaller than was paid for in ordinary smelting practice.

The first tests to prove the discovery were made in the laboratory of Henry E. Wood, Denver, and published by Prof. Wilbur C. Knight, of the University of Wyoming.[1] These tests showed that various copper minerals from the mine carried from 0.10 oz. to 0.70 oz. platinum per ton, while several carloads of covellite ore carried from 0.40 oz. to 1.40 oz. platinum per ton. About the same time Wells and Penfield[2] determined the mode of occurrence of the platinum to be sperrylite, $PtAs_2$. Their work was done on a sample of covellite, from an ounce of which they recovered six brilliant crystals of sperrylite.

Subsequent to this time many confirmatory tests were made by members of the company's staff, the United States Geological Survey, and others. The later investigations gave evidence of the presence, in considerable quantities, of other members of the platinum group of metals. The first commercial results were obtained from consignments of high-grade copper ore to the Chicago Copper Refining Company, Blue Island, Ill. On the assumption that all of the noble metals ultimately would be collected in the copper base in the smelting process, and later precipitated as anode slime in the electrolytic refining process,

[1] *Eng. & Min. Journ.*, Dec. 28, 1901.
[2] *Amer. Journ. Sci.*, Feb. 1902.

arrangements were made to recover this slime for further treatment. Assays of this slime showed the presence of considerable quantities of platinum and palladium, the values ranging from $5.50 to $6.50 per pound. One lot assayed 226 oz. platinum and 166 oz. palladium per ton of slime. From two portions there were actually recovered 16.74 oz. of platinum, 37.5 oz. of palladium, 5.33 oz. of gold and 38.55 oz. of silver.

Investigation of Treatment Methods.

The definite and rather extensive occurrence of the metals of the platinum group having been established, the company was confronted with the problem of ore treatment. It was apparent that much had been lost in the several thousand tons of ore shipped to copper reduction works, and that some system must be devised for saving the precious metals. A small matting furnace which had been erected at the mine had not been operated successfully, as the present appearance of the old slag dump, copper bottoms and matte shells now indicate. The failure of this plant is not surprising in view of the abundance of decomposed diorite, or so-called talc, in some of the ore, carrying 55 per cent silica and 30 per cent alumina and iron. Some of the highest grade platinum ore is especially refractory in the smelting process, one analysis showing 12.75 per cent insoluble and 52.85 per cent alumina and iron, with no lime and magnesia.

The complexity of the problem of suitable ore treatment will be better understood from a consideration of the variety of copper minerals existing in the mine, much of it low grade and requiring concentration. The Rambler has yielded, up to date, the following copper minerals:

TABLE OF MINERALS OCCURRING IN THE RAMBLER MINE.[3]

Mineral.	Composition.	Sp. Gr.
Sulphides		
Chalcocite	Cu_2S	5.7
Chalcopyrite	$(Cu_2)S_2(Fe_2)S_3$	4.2
Covellite	CuS	4.0
Carbonates		
Azurite	$(CuOH)_2Cu(CO_3)_2$	3.7
Malachite	$(CuOH)_2CO_3$	3.8
Oxides		
Tenorite	CuO	5.8
Cuprite	Cu_2O	6.0
Chalcotrychite	Cu_2O	6.0
Chloride		
Atacamite	$Cu(OH)_2Cu(OH)Cl$	4.1
Sulphate		
Chalcanthite	$CuSO_4\ 5H_2O$	2.2
Silicate		
Chrysocolla	$CuSiO_3 . 2H_2O$	2.1
Native Copper		
	Cu	8.9

These minerals occur also in diorite in all stages of alteration, even the socalled talc carrying copper and precious metals. One of the minerals, chalcanthite, is soluble in water, and the carbonates and oxides do not yield readily to concentration. Furthermore, all of these copper ores occur closely associated, one mineral sometimes coating another, and it would be almost impossible to sort them. Finally, all of them, but particularly the covellite, carry the precious metals of the platinum group very finely divided, the largest crystals of sperrylite isolated by Wells and Penfield measuring only 0.12 m in diameter. Sperrylite has a specific gravity of 10.6 and the metals of the platinum group have specific gravities ranging from 16 to 21.

The following table[4] shows the distribution of precious metals in the ores:

Identification of Sample.	Value per ton in Precious metals.	Copper.
High-grade platinum, circular drift	$36	$3.28
Hard talc, N. W. drift	12	2.75
Sulphide, black drift	26	1.45
Talc, black drift	20	5.88
Talc, covellite drift	32	12.34
Black material, black drift	88	4.65
High-grade platinum, opposite talc contact	100	75.00
White talc at contact, first level	26	5.75
Covellite, covellite drift	80	59.00
Brown talc at contact, first level	12	4.90
Precipitate above talc, covellite drift	116	68.60
General sample, stope level	26	16.40
Low grade, second level	12	2.00

From these considerations it will be seen that the Rambler ores offered a most interesting problem in metallurgy.

As a result of experiments made in 1906 under the direction of Dr. David T. Day, of the United States Geological Survey, it was found that when the ore was crushed to 2 mm and concentrated on a Wilfley table about one-half the platinum content of the ore was recovered in the concentrate, the other half being lost in the middling and tailings. The test further indicated that the precious metals occurred very finely divided and it was Mr. Day's opinion that grinding to ½ mm would result in a saving of 80 to 90 per cent of the precious metals.

In the spring of 1909 concentration experiments were carried out at the ore testing works of H. E. Wood, Denver. The results obtained on ore crushed to 20-mesh were comparable with those obtained by Mr. Day. About 56 per cent of the precious metals were obtained in a concentrate assaying 1.70 oz. platinum per ton. On reconcentrating this concentrate it was possible to obtain a precious metal product carrying 13.06 oz. platinum per ton, the ratio of concentration being 150 of crude ore assaying 0.69 oz. platinum per ton into 1 of precious metal concentrate assaying as stated above. A second test on material ground to pass a 16-mesh screen gave a concentrate assaying 1.38 oz. platinum per ton, which on reconcentration yielded a product assaying 14.62 oz. platinum per ton. This final product represented a concentration of 191 tons of crude ore into one of precious metal concentrate.

Later tests conducted by Prof. A. C. Dart, of the School of Mines, University of Wyoming; Mr. Robert A. Marr, consulting metallurgist, Union, Ore., and FitzGerald & Bennie, electrometallurgists, Niagara Falls, N. Y., indicated the possibility of profitably concentrating the low-grade copper ore by grinding fine enough to liberate the precious metals, making a high-grade precious metal concentrate and a copper concentrate carrying some precious metals. Following these opinions the company has erected a concentrating mill at the mine.

Rambler Experimental Mill.

The mill was erected adjoining the old ore bins, so that the ore could be handled from them by gravity. The mill equipment consists of crusher, rolls, trommels, classifier, Wilfley tables and canvas tables. As the plant is being operated in an experimental way, the system of treatment cannot be said to be definitely settled. In fact, the mill has been in operation just long enough to present a number of problems that have not been solved as yet, and it is expected that experimental operation will continue for some time. It will be necessary to determine what commercial products can be made most economically and in what quantity they can be produced. it will then be possible to negotiate the sale of the concentrates or consider their treatment locally, although the latter is improbable.

The mill was designed and erected by Mr. R. A. Marr, but its present operation is under the direction of Mr. Dart, who is acting as consulting engineer to the company. On page 77 is a diagram of the flow of ore through the mill.[5]

The crusher is set to crush to ½ in., and the rolls to grind to 20-mesh. The trommels are fitted with punching screens to deliver 20-mesh material. Three Wilfley tables are fed from

[3] Composition and specific gravity taken from Frazer & Brown Tables for the Determination of Minerals, 6th Edition.

[4] Assays by Fitz Gerald & Bennie. Precious metals calculated at $20 per ounce and copper at 14 cts. per pound.

[5] The abbreviation P.M. used in the diagram and following tables indicates "precious metals."

the three compartments of the classifier, and the overflow from the last compartment runs to two canvas tables. At present three products are being taken off the tables, viz., a high-grade precious metal concentrate comprising a streak of about 1 in. wide at the top of the table; a copper concentrate, and a combined middling and tailing, the last being impounded for further treatment. Owing to the lack of a regrinding machine the middling is being combined with the tailing for the present, although conditions indicate the advisability of regrinding the middling. The precious metal streak is made up of very finely divided particles, and carries the major percentage of the platinum group metals in the ore; the copper concentrate carries the bulk of the sulphides, and the middling is composed largely of 20-mesh grains of the carbonate copper ores.

The following table gives the assays of the various table

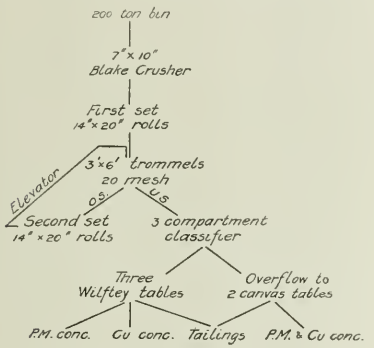


DIAGRAM OF FLOW OF ORE THROUGH MILL.

products, as made June 25, 1910, by Mr. L. S. Worthington, the company's chemist and assayer. The methods of assay will be discussed later.

	Ounces per Ton.					Per cent
Description.	Total P. M.	Au.	Pt.	Ag and Pd.	Ir.	Cu.
Mill feed	1.27	....	.....	....	....	2.8
P. M. conc. table No. 1....	20.19	4.01	11.21	1.76	3.21	19.7
Cu conc. table No. 2.....	1.57	....	0.36	1.21	....	7.3
Middling table No. 1.....	1.15	....	.....	1.01	....	2.7
P. M. conc. table No. 2....	7.96	2.06	3.24	1.63	1.03	15.4
Cu conc. table No. 2.....	1.24	....	0.12	1.12	....	7.9
Middling table No. 2.....	1.07	....	.....	0.91	....	1.8
Mill tailing	0.90	....	.....	....	....	2.3

On June 30, 1910, Mr. Dart and the writer jointly sampled and assayed the various table products. The following table gives the results obtained:

	Ounces per Ton.						Per cent
Description	Total P. M.	Au.	Pt.	Ag&Pd.	Ir.	IrOs & Rh.	Cu.
P. M. Conc. table No. 1...	28.00	4.66	11.82	11.08	0.08	0.36	23.1
Cu Conc. table No. 1.....	1.86	Trace	0.06	1.26	Trace	.....	7.2
P. M. Conc. table No. 2...	17.68	2.36	4.86	10.36	0.10	Trace	22.4
Cu Conc. table No. 2.....	2.40	Trace	0.14	1.76	Trace	.....	12.6
P. M. Conc. table No. 3..	72.50	6.92	7.28	11.22	13.00	34.08	23.4
General P. M. Conc. tables Nos. 1 and 2.....	15.28	2.64	5.78	5.86	0.38	0.62	18.4

An imperfection in table No. 3 made it necessary to take the sample of precious metal concentrate periodically by hand. When the sample was not being taken the precious metals ultimately combined with the copper concentrates and were, therefore, not included in sample six of the above table. For the same reason no assay appears for the copper concentrate of table No. 3. In preparing the samples for assay no metallics were observed on a 100-mesh screen. Of the precious metal concentrates 33 per cent of the sample from table No. 1 passed a 100-mesh screen; 25 per cent of sample No. 2 and 90 per cent of sample No. 3 were also 100-mesh and finer. It will be observed that the richness of the sample varies directly with the fineness, the finest material being the richest.

Assay Methods.

In view of our comparatively meager knowledge of assay methods for metals of the platinum group, it would be unfair to all concerned to present the foregoing figures without an accompanying statement of the methods employed. The assay for platinum is usually made on sands containing metallics, or on alloys of the metal, and it is seldom that ores carrying the platinum group metals are presented for assay. The figures given in the foregoing tables do not represent *direct weights* of the metals indicated, but are calculated from *differences in the weights* of the buttons after successive partings with different acid solvents which are supposed to extract certain metals from the button.

The procedure is based on the following statements made by Lodge[6] and Fulton.[7] In an alloy of the precious metals,

(1) Sulphuric acid will dissolve silver and palladium together;

(2) Nitric acid will dissolve platinum when the ratio of silver to platinum is 12 or 15 to 1, or when the ratio of silver to gold to platinum is 15:2:1;

(3) Dilute aqua regia, 1:5, will dissolve gold;

(4) Concentrated aqua regia will dissolve iridium, and,

(5) The final residue insoluble in the foregoing treatment will be iridosmium and rhodium.

The method of assay used at the Rambler mine at present is as follows: Crucible fusions of ½ A.T. of material are made in duplicate. To one crucible is added silver in quantity about four times the weight of the precious metals believed to be present. As there is considerable copper and precious metal in the lead button, the cupellation is carried out at a high temperature, and if the resulting bead is large it is recupelled with an addition of lead to clean it thoroughly. The high temperature employed probably volatilizes any osmium and rubidium as oxides, so that these do not remain in the bead. The beads are dark gray in color, flat and rough.

The button to which no silver was added is weighed and calculated as total precious metals. The button alloyed with silver is parted with sulphuric acid and weighed. The difference in weight between this residue and the total precious metal button is calculated as silver and palladium. No further use is now made of this residue, unless it should be needed to replace a button lost in subsequent operations, or to check later results. The total precious metal button is now alloyed with silver, using 12 to 15 times the weight of residue from which silver and palladium have been dissolved.

This button is parted with nitric acid, specific gravity 1.20, and weighed, recording the differences in weight between it and the next preceding weight as platinum. The residue is then warmed with dilute aqua regia, 1:5, which will dissolve the gold, the weight of which is calculated by difference after another weighing. Finally the residue is boiled with strong aqua regia, dissolving iridium, which is estimated from the difference in weight between the residue and the next preceding it. The final weight gives iridosmium and rhodium. In alloying the cupelled buttons with silver in the first stage of the work it is noticed that small dark spots appear on the molten button, finally settling on the bottom when cold. These are probably iridium, which is exceedingly difficult to alloy. In making the partings it may be necessary to repeat the operation if for any reason it is feared that solution is incomplete, and this may be continued until a constant weight is obtained.

There are points in this procedure about which questions can be raised. It is doubtful whether the solvents used are without action other than that attributed to them, so that the final assay figures for any one of the metals may be high or low, due to incomplete solution of one of the constituents of the residue, or to partial solution of a constituent assumed to be insoluble. In any event, the total precious metal button can usually be relied upon, and the various constituents are reasonably correct.

[6] Notes on Assaying.
[7] A Manual of Fire Assaying.

This does not obviate the necessity or desirability of more exact procedure, and it is hoped that the production of high-grade precious metal concentrate at the Rambler will stimulate research in this direction.

Mention has been made of smelting operations carried on formerly at the Rambler mine. A matte smelter was erected in 1902 and operated for six or eight months, but finally abandoned as unsuccessful. Probably suitable fluxes were not available, and as the ore does not carry much iron, it was difficult to smelt. Samples of matte and copper bottoms taken from material on the dump showed, respectively, the presence of 17.00 oz. and 32.52 oz. of precious metal per ton of such material, thus showing the probability that precious metals have always occurred in the ore and have been lost in most of the previous shipments to copper reduction works. The data further indicate the possibility of gathering the precious metals in copper matte and blister copper. These facts, together with some work done for the Rambler company by the Chicago Copper Refining Company and FitzGerald & Bennie lead to a suggested system of metallurgy. The work referred to consisted in reducing some of the Rambler high-grade ore, producing blister copper. This was analyzed by FitzGerald & Bennie, with the following results:

Copper	91.1 per cent.
Precious metals	3.91 oz. per ton.
Silver	6.42 oz. per ton.
Lead	2.17 per cent.
Nickel	0.60 per cent.
Antimony	1.95 per cent.
Iron	0.90 per cent.

The lead was probably gathered from previous operation of the furnace, as it does not occur in the Rambler ore in any appreciable degree. Nickel has been identified in Rambler ores and antimony also occurs in small quantities, some authorities giving it as a constituent of sperrylite. It was from the electrolytic refining of such blister copper as this that the anode slime previously referred to was recovered and subsequently refined by Baker & Company and the S. S. White Company. .

In consideration of the foregoing data it would seem as though a rational system of metallurgy for Rambler ores would be the following: Direct smelting of the high-grade (25 to 30 per cent) copper ore, producing a matte subsequently to be blown to blister copper. Concentration of the low-grade (2 per cent) copper ore, recovering a high-grade precious metal concentrate which could be refined chemically, and a copper concentrate which could be smelted, blowing the resulting matte to blister copper; or, all of the valuable concentrate could be combined for smelting. The copper obtained from all smelting operations could be refined electrolytically and the anode slime containing the precious metals could be treated chemically. The copper carbonate and oxide minerals in the low-grade milling ore, which pass into the tailing pond, could be leached with the mine water, which is quite acid and also carries some copper in solution. The copper dissolved in this process could be precipitated on scrap iron, but the precious metal in the tailing would, for the present, have to await the development of some method of recovery. Or it might be found that it would pay to smelt 2 per cent copper tailings if the silica were not too high and the precious metal content were sufficiently valuable. The question of transportation would play a large part here. It will be apparent from the assays previously given that the concentration of the copper values in the low-grade ore would scarcely be profitable were it not for the precious metals recovered, but since the copper lost in the tailing and middling is almost all in a soluble form, the process outlined is expected to prove profitable and may be adopted.

The writer is indebted to Mr. W. J. Sherwood, mine manager for the Rambler Copper & Platinum Company, for permission to publish the data given herewith, and to Mr. A. C. Dart, consulting engineer, for courtesies extended.

Exploitation of Peat Bogs.

The exploitation of Canadian peat bogs for the production of fuel for domestic and industrial purposes was the subject of an interesting address, delivered last year before the American Peat Society by Dr. Eugene Haanel, Director of Mines, Ottawa, Ontario, Canada. This address has recently been published in form of a pamphlet, from which the following extracts are taken.

A former extended investigation of the manufacture of peat fuel, carried out by the Department of Mines of Canada, has demonstrated, firstly, that for the economic production of fuel from peat machinery driven by power must be substituted as far as possible for manual labor; and, secondly, that processes so far invented for removing the water content of the peat by pressure and artificial heat have not led to commercial results, and after trial have been abandoned.

Two promising processes employing artificial heat in the production of peat fuel have given disappointing results.

The Electropeat Syndicate erected an extensive plant at Kilberg, in the County of Kildare, Ireland, for the production of a substitute for coal from peat, which they expected to sell at six shillings per ton. No money was spared in trying to make the experiments a success, and most expensive machinery was put down. Experts were brought from Germany to look after the work under the superintendence of Captain Verey, R. E. Large shipping companies on the strength of samples submitted, and representations made to them, promised some very large contracts for fuel, but the company, at the end of over one year's experimentation, found that its process was a failure. The peat made by it looked like coal and burned well, but was hygroscopic, and after a short time crumbled to powder. All efforts to overcome this defect failed, and the project was permanently abandoned.

The Elkeberg process of the wet carbonization of peat is exceedingly ingenious, and has aroused great expectations, but is still in the experimental stage after the expenditure of 200,000 kronen in Sweden to place it upon a commercial basis.

The endeavor to accomplish *economically* by artificial means in a *short time* what has been accomplished by nature in exceedingly long periods of time, the change of peat into a substance similar to coal, has so far apparently not been attended with success. Dr. Haanel would not like to say that it cannot be done, since it is unsafe to make any statements regarding the possibilities of future achievement, but at present the outlook in this direction is not encouraging.

The proper course for Canada is, as Dr. Haanel points out, to introduce such processes and such machinery as have proved successful and are now in actual commercial operation in Europe.

To re-establish the confidence of the people of Canada in the value of peat as a domestic and industrial fuel and to stimulate renewed activity in the development of the great peat resources of Canada the government has acquired 300 acres of peat bog, with an average depth of 9 ft., for the purpose of manufacturing peat fuel on a commercial scale, and by a method which has proved successful in European practice. At this plant interested parties have an opportunity of ascertaining for themselves the working of the bog as well as the suitability of the peat fuel produced. The capacity of this plant is a production of 30 tons per day. For a large commercial plant mechanical excavators should replace the manual labor employed at this plant, if the bog to be exploited is suitable for this class of labor-saving machinery.

The plan at Alfred is to serve as a model of a successful process, and not for the production of peat fuel on an extensive scale. It was expected, however, to manufacture during the first season about 2000 tons of peat fuel, part of this to be used in their peat-gas producer at Ottawa.

There is nothing artistic about the appearance of the fuel produced at this bog. It has not the regular geometric form

of briquettes nor their smooth exterior, but it serves the purpose for which fuel is intended as well as briquettes, and has the advantage of being low in cost of manufacture.

Allowing 140 days for a season's operation, the cost per ton of air-dried machine peat, including interest on capital invested, amortization, oil and repairs, is given by Dr. Haanel as follows:

Cost of fuel on the field	$1.40
Cost of fuel stored in shed	1.65
Cost of fuel loaded on car	1.65
Cost of fuel in stack	1.70

By the employment of mechanical excavators and the manufacture of peat on a large scale, the cost of production per ton should be considerably less than the figures here given.

The objection to the air-drying process practised at this plant is that it is not a continuous process, that it can be worked only during the summer months, and that the amount of fuel which can be produced during one season is dependent upon weather conditions. But air-drying is employed successfully on a large scale in Europe, and especially in Russia, and the weather conditions in Canada are at least as favorable, if not more so.

The economy which may be effected by the use of peat in Canada is readily understood when it is stated that in Ontario and Quebec the average price of anthracite with a high percentage of ash is $7.50, and in Manitoba, $10.00 per ton, whereas air-dried machine peat, containing only about 5 per cent of ash can be manufactured at a cost of considerably under $3.00 for an amount having the same calorific value as a ton of the anthracite which must be imported. This could be sold at places conveniently situated as regards transportation facilities, and not too far from the place of manufacture for a little more than one-half the price paid for anthracite in Ontario and Quebec, and for a little more than one-third of the price paid for anthracite in Winnipeg.

Air-dried peat is not alone an excellent fuel for domestic use, but for the production of power it proves an ideal fuel in the peat-gas producer, which is to-day as reliable and efficient in its operation as the coal-gas producer. Dr. Haanel did not hesitate to say that it is an ideal fuel because the peat from most bogs is free from a clinkering ash and yields on combustion a fine, white residue, which readily allows of the thorough cleaning of the fire, and the property of not fusing or caking in the producer assures regular operation.

Moreover, since gas leaves the producer with a high degree of sensible heat, which must be cooled to the temperature of the atmosphere before being used in the gas engine, it is exceedingly important that as much of this sensible heat as possible be utilized in the producer itself, in order to increase its thermal efficiency. This is accomplished in coal-gas producers by the introduction of water vapor which passes through the incandescent fuel with the air supplied for combustion. This water vapor is decomposed, yielding hydrogen and oxygen. The latter combines with the carbon of the fuel, forming carbon monoxide.

This chemical reaction absorbs a large amount of heat and lowers the sensible heat of the gas, but the heat absorbed in liberating the hydrogen is to a large extent restored and utilized when the gas enriched by hydrogen is burned in the gas engine or other apparatus. With peat containing from 25 to 30 and more per cent of moisture the latter is sufficient to accomplish all that is required without the introduction of water in the producer from an outside source.

To demonstrate the value of peat for the production of power for industrial purposes a modern German peat-gas power plant has been erected by the Department in Ottawa. Its capacity is 60 hp, and consists of a double firezone Koerting peat-gas producer, with the necessary gas-cleaning apparatus, and a Koerting four-cycle single-acting gas engine direct connected to a Westinghouse 50-kw, direct-current generator.

The consumption of peat per brake horse-power is expected to be in the neighborhood of 2 lb., definite figures not yet being available.

The erection of gas producers designed for the recovery of by-products is not recommended except in localities where such by-products would command a ready and profitable market. In Canada it is far more economical to aim at the complete gasification of all the heat elements in the fuel.

Nucleus Action and Grain Growth

By HENRY M. HOWE.

The purpose of this note is to consider the explanation which Mr. E. Cohen gives of the interesting cases of nucleus action which he has observed.[1] He finds that tin which has been dulled in a certain way, when pressed against bright cold worked tin, such as tinfoil, or tin plate, inoculates these latter with like dullness, and that these infected spots thus set up can carry the infection to still others, slowly at room temperature (§ 4) and more rapidly at higher temperatures. (§§ 6, 13, 14.) What is the exact nature of this infection? The bright surface which he dulled in most of his experiments had been made by cold rolling or like action, and he explains the infection as an undoing of this cold rolling, reasonably calling the cold-rolled state metastable, and the non-cold-rolled state, that is, the dulled state to which infection brings it, the stable state ("forciert" and "nicht forciert," which we may translate as "overstrained" and "not overstrained," "ecroui" and "non-ecroui").

A prominent effect of overstrain, by whatever form of cold-working it is caused, is to distort the individual crystalline particles of which the metal is normally composed. Most metals, iron included, belong to the cubic (monometric) system; and the distortion incident to overstrain, of course, changes by different amounts the length of the several axes of any given crystalline grain, so that these grains, normally "equiaxed," become "inequiaxed." So we often refer to the "inequiaxing" caused by overstrain, and the "equiaxing" caused by annealing, one feature of which is that it restores the crystalline grains or particles to their normal state. But as tin belongs to the tetragonal (dimetric) system, with axes normally unequal, "equiaxing" and "inequiaxing" are not strictly applicable. Let us rather speak of "crystalline distortion" or "abnormal axing" and "crystalline readjustment," or "normal axing."

Applying these ideas to Mr. Cohen's experiments the initial bright state of his tinfoil and tin plate is one of crystalline distortion or abnormal axing, the diseased state; and the dulled state is one of cure, readjustment, or normal axing, in spite of the fact that here the cure is, as far as the practical applications of the tin are concerned, far worse than the disease.

The question which I wish to consider is, whether under Mr. Cohen's various conditions we can properly regard crystalline distortion and readjustment as the chief causes of the phenomena. Let us consider these phenomena.

A first case is that of the dulling of tin plate by reheating. Simple exposure to 161°, 176° and 184° did not cause dulling, though like exposure when coupled with contact with dulled tin did, and though Behrens found that bright faces of very greatly overstrained tin dulled at 110° and very rapidly at 150° (§17). This case falls readily into line. The molecular mobility caused by heating, and the nucleus action of dulled (readjusted or normal axed) crystals, are clearly allies, jointly favoring the readjustment (dulling) of distorted crystals. Hence it is but natural that their joint action should cause the crystalline readjustment of which dulling is but an evidence, even when heating alone fails to; and that the greater instability of Behrens's greatly distorted grains might cause them to submit to the tendency to readjustment, which heating to 150° or even only to 110° sets up, even though the less unstable grains of Cohen might defy this tendency to readjustment even at

[1] *Zeitschr. phys. Chemie*, 1909, **68**, *p.* 214. See also this journal, Vol. VIII, p. 510.

184°, unless it were reinforced by alliance with the nucleus action of dulled and normal-axed particles.

A second case is that careful oblique hammering on the bright surface of tin plate so as to destroy the polish sets up the dulling, which in turn dulls tinfoil pressed against it at 100°, setting up centers whence the dulling spreads. (§ 21.) This case, too, falls into line, if we assume that an exaggeration of the overstrain creates a degree of instability, an abnormality of axing, so great that the readjustment and normal axing occur, as in Behrens's case, without nucleus action. In short, the instability, when exaggerated, from being metastable becomes labile.

A third case may be considered. The watered (moiré) surface of etched tin plate dulls a bright tin plate when pressed against it at 184°, though two bright tin plates thus pressed do not dull each other, and though the watered surface is not affected by contact with dulled tin. (§§ 20-21.) To bring this into line we have to suppose that this watered look caused by etching is due to the readjustment and normal axing of the crystals, so that etched tin and dulled tin are alike in being normally axed.

A fourth case is less clear. The bright surface of a quietly frozen tin ingot is readily dulled by contact with dulled (infected) tin, whereas the surface and even the interior of a tin ingot shaken during freezing has the dulled look of infected tin, and has the power of dulling bright tin. In order to bring this into line we seem forced to assume that the bright surface of the quietly frozen ingot is in effect overstrained. When the metal is first poured freezing is likely to take place first at the bottom and sides of the ingot in contact with the cold mold. Later a thin skin freezes over the upper surface; and though this skin remains bright, its crystalline structure may misfit that which, spreading up quickly from bottom and sides, approaches it from below, so that the very skin may find itself in unstable crystalline equilibrium, not because itself abnormally axed, but because discordantly axed with regard to the mass underlying it. Or its own surface tension may so interfere with the crystallization of the very skin as to make it unstable. Because of this instability the surface may recrystallize and dull when to the influence of the underlying discordant crystals, or of its own instability, is superadded the influence of the dulled tin pressed on it from above.

But this explanation may be accused of bearing too much good will; and we find ourselves left with a desire either to clinch it by other and corroborative evidence, or to seek some further explanation. The amorphous state may give us the key.

In this connection there is great interest in Mr. Charpy's[1] discovery that previous overstrain accelerates enormously the growth of crystalline grain (Stead's brittleness) which occurs when very low carbon steel is held between 650° and 800°. Thus two like pieces of very low-carbon and moderately phosphoric steel, one wire drawn and the other not, were held in this range of rapid growth between 650° and 800°. After this the grains of the wire-drawn piece were about 10 times as wide as those of the other, implying a difference in volume of 1000 to 1. Steel which when wire drawn could be bent considerably without cracking, after later heating at about 650° became so brittle that it broke when dropped on the ground. Thus, though one effect of reheating wire-drawn steel is truly to anneal it, by normal axing and relieving stress, and perhaps in other ways, yet in case of very low-carbon steel an undue exposure to this very annealing temperature may cause a grain growth far more harmful than the abnormal axing and the stress which it removes. The cure of two diseases may here give rise to a third worse than those two.

That this growth should be rapid in this very low-carbon metal so heated is indeed natural. In the presence of much carbon the growth of the ferrite, whether alpha below Ac2 or beta between Ac2 and Ac3, is naturally impeded by the presence of the cementite. In very low-carbon steel the relative freedom from cementite removes this obstacle to growth. Of course, if the temperature rose past Ac2 so that the iron became beta the particles of this modification would be nascent during the change, and this nascency might well lead to extremely rapid crystalline growth. That the coarse grain should be effaced when, on rising past 900°, the iron changes into the gamma state is natural enough. The new state might well form new crystals of its own.

The reason for the acceleration of the crystalline growth caused by overstrain remains to be determined. Of course, the abnormally axed overstrained crystalline particles are initially in unstable equilibrium, so that to the impulse toward crystalline readjustment due to the mobility of the high temperature, say 750°, is added the impulse due to the initial instability, resulting in a greater rapidity of readjustment. One could hardly foresee that this should lead to a more rapid growth of grain; but the fact that it does is not unnatural, and may perhaps be likened to the lessening of friction which motion causes. Just as the mechanical friction of motion is far less than the mechanical friction of rest, so the anti-crystallogenic friction may be less in the overstrained metal because of the added rapidity of molecular motion caused by the instability due to initial overstrain.

Columbia University.

[1] *Rev. Metallurgie mem.*, 1910 **7**, p. 655.

The Death of Dr. F. A. Kjellin.

We regret to report the sudden death of Dr. F. A. Kjellin at the age of 38 years on Dec. 30, 1910. His name will be forever remembered for his connection with the development of the electric induction furnace. While others before him, especially Mr. E. A. Colby in this country, had invented and tried an induction furnace, Kjellin was the first to use it on a commercial scale, and he did important pioneer work in electric steel melting and refining.

THE LATE DR. F. A. KJELLIN.

Frederick Adolf Kjellin was born in the county of Vardinge, not far from Stockholm, Sweden, on April 27, 1872. He was educated at the Royal Technical Institute in Stockholm from which he graduated in 1893. During subsequent years he carried out a great deal of experimental work in chemistry and metallurgy, and worked out a new method for the electrolytic precipitation of zinc, on which method he obtained a patent. During the years 1893-1895 he assisted Professor Wiborgh in developing the well-known Wiborgh termofon for the determination of high temperatures. From 1895 to 1899 he was engaged in developing the Lux light now used extensively.

In the year 1899 he began at Gysinge his work on the use of the electric induction furnace for steel melting. In 1904 he became connected with the Swedish Gröndal-Kjellin Company, in the capacity as chief of the electrometallurgical department. During the last few years he acted as consulting engineer in this department. The honorary degree of Doctor of Philosophy was bestowed upon him by the University of Upsala in 1907.

The Manufacture of Ammonia-Alkali in England.

By Wm. Mason.

There is no branch of chemical industry in which the principles of engineering science have been so profitably applied as that of the manufacture of ammonia-alkali. The world's production of this article, which in 1883 was only 160,000 tons, has steadily increased until at the present time it is estimated to amount to upward of 2,000,000 tons per year, about nine-tenths of which is said to be produced by the Solvay system. The English Solvay works of Messrs. Brunner, Mond & Company, Ltd., in the Cheshire salt district, are the largest in the world and since the year 1881 have continued to show the most magnificent financial results.

It has been thought by some that the Solvay system was a secret method, but that is not so. This system works the method first made known by Schloesing and Roland in 1854, which is a modification of that practised by earlier workers, such as Dyar and Hemming, who obtained the first patent for this process in 1838. It is represented by the following equation:

$$Na\,Cl + NH_3 + CO_2 + H_2O = Na\,HCO_3 + NH_4\,Cl.$$

The ammonia-alkali process is so simple, so unlike the Leblanc process, that any one with only a little experience in chemical manipulation can produce sodium carbonate by it.

Solvay's first patent in 1863 was for plant, and a few years afterward he established works at Couillet, in Belgium, making in the first year 179 tons of soda ash. He was the first to make the manufacture of ammonia-alkali a financial success, increasing the production each year, until in 1874 he manufactured upward of 4678 tons, when it was introduced into England by the late Dr. Ludwig Mond with a success that is without a parallel in the history of chemical industry. The English Solvay works have paid an average dividend for the last 28 years of 31 per cent, the lowest in 1881 of 19 per cent, and the highest of 100 per cent in 1893.

Notwithstanding such success it can be shown that in the works of this successful company, when they were annually making a profit of nearly £200,000, in trying to improve the chemical side of this manufacture, they adopted and persistently worked a method which must have added at least 10 shillings per ton to the cost of production.

It has often been stated that in the carbonating part of this process a quantity of sodium bicarbonate remaining dissolved in the residual chloride liquors would always be lost, and to recover this bicarbonate Solvay suggests in one of his patents the placing of salt in the carbonating tower; but ordinary salt, as supplied to chemical works, contains impurities equal to 0.8 per cent, which increases the percentage of insoluble matter in the finished soda ash. In 1878 Dr. Mond obtained a patent for a method for purifying the brine by adding to it a kind of water-softening mixture; by this method he obtained a very pure salt. This process was worked for some time and then discontinued, ordinary salt being afterward used.

Now there is no evidence whatever to show that any of the salt added in this way was ever decomposed, or was necessary to prevent the loss of bicarbonate in these liquors. For this purpose about 10 cwts. of salt per ton of soda ash produced was required; this salt contained 10 per cent of water, which diluted the liquors, the dilution being further increased by 50 per cent more water being required to wash out the excess of salt from the bicarbonate. This water carried away bicarbonate in solution, thus making more liquor to be passed through the ammonia still, which increased very considerably the cost of manufacture.

Below are given some analyses of the residual liquors: No. 1 from Solvay towers having had salt placed in them; No. 2, from another works which did not use salt in the carbonating apparatus, while No. 3 gives the analysis of liquors from Solvay towers after finding out that salt was not required to obtain a residual liquor from the carbonating tower, with little or no sodium bicarbonate in solution:

No. 1. Solvay Tower Liquor with Salt Added.

Percentage alkalinity calculated as NH_3	1.49	1.19	1.61	2.22
Ammonium chloride	16.53	17.12	16.42	14.29
Sodium chloride	13.11	12.98	12.29	14.98

No. 2. Carbonator Liquor without Salt.

Percentage alkalinity calculated as NH_3	1.65	1.87	1.56	1.63
Ammonium chloride	18.88	17.97	18.08	17.33
Sodium chloride	6.88	7.28	7.74	8.00

No. 3. Solvay Tower Liquor without Salt.

Percentage alkalinity calculated as NH_3	2.02
Ammonium chloride	17.22
Sodium chloride	8.45

Since for every equivalent of ammonium chloride produced an equivalent of sodium carbonate is precipitated as sodium bicarbonate, analyses Nos. 2 and 3 clearly show that slightly better results were obtained than by the addition of salt.

Although this process is an English one, it was not worked in England until the year 1874, when works were built at Northwich, and a year later at Sandbach. In 1874 Solvay manufactured upward of 4678 tons and it is evident that the works mentioned above were of the same capacity. They are very clearly described by him in the specifications of patents No. 999 and No. 1904, of 1876. The plant at Sandbach consists of lime kilns, four ammonia stills with heater and absorbers, five carbonating towers, four drying machines, besides other vessels. When working this plant showed a profitable return, but it was almost impossible to keep it continually working because of its extremely complicated parts, chiefly in the ammonia stills, the connecting pipes becoming blocked and the changing valve leaking. Moreover, the lime, being charged unslaked, was very badly burned.

Through these causes it might safely be said that the plant did not work more than two-thirds of its time, yet with all its drawbacks this plant was working in the year 1881. This plant could never have competed favorably with the Leblanc process and the ammonia alkali industry was in a most critical condition when the late Dr. Ludwig Mond, from the capacity of the ammonia distilling apparatus of this plant, designed the vertical still afterward patented by him in the year 1883.

This still is a fine example of chemical engineering, as it shows how the principles of engineering may be applied in the construction of chemical plant; it was such an improvement over the four-still arrangement that the daily output was increased by more than one-third, that is, from 13 tons to more than 21 tons, without increasing the other parts of the plant; in fact, this vertical still made the plant almost continuous, having only to be stopped a day in about six weeks.

This plant was duplicated and worked most successfully for many years. A vertical still had been used in distilling ammoniacal liquors long before this time, and it seems strange that it was not used at first, as it was much cheaper to construct. Solvay describes one in his specifications of patents for 1876.

The carbonating tower patented by Solvay in 1872 is also an apparatus which shows a considerable amount of engineering skill and may be considered his best contribution to this particular process and it still remains a part of the present-day plant with the addition only of an internal cooling coil.

Although the patent protecting this Solvay carbonating tower, which was so ingeniously designed and which produces the most perfect results in working, became void in 1886, another works was established at Runcorn which contained a plant much simpler in construction than that of Solvay; in fact, it consisted chiefly of large cylindrical vessels made of boiler plate to the number of about 30. It was worked as follows:

The ammoniacal gases from the stills, without cooling, passed into vessels called saturators which contained salt-brine and into which solid salt was also charged by means of a cage run-

ning down the length of the saturator, so that brine was made and saturated with ammonia in one operation. The ammoniated brine so produced was similar to that obtained in the Solvay system.

The great drawback of this method was that much salt became deposited in the saturators, sometimes to the depth of 12 ft. or more, which made it difficult to estimate the ammonia in them, a matter of the greatest importance in this process. The stills in this department on several occasions recovered upward of 620 lb. NH_3 in the hour.

From the saturators the ammoniated brine, after cooling and settling, was passed into vessels called absorbers, two in number, so that while one was being filled with ammoniated brine the other was working as a part of the carbonating plant. The carbonating plant consisted of three vessels, each containing about 600 cu. ft. of ammoniated brine. The carbonic acid gas from the lime kiln, after being washed and cooled and at an initial pressure of 38 lb. per square inch, passed through these concentrators, through an absorber, then through a tank containing brine and lastly through a vessel containing acid.

Five carbonators were worked off during the day of 24 hours and produced about 10 tons of soda ash. The washing of the soda bicarbonate was so carried out that about 20 per cent of it was washed away and lost. Analyses of residual liquors from above carbonators have already been given above (Table No. 2).

By the Solvay towers the 600 cu. ft. of ammoniated brine could have been worked off in less than three hours, that is, in the state of carbonation as it passed into these carbonators, and at least 15 tons of soda ash might have been produced instead of 10 tons. The plant was supposed to contain a quantity of ammonia equal to 55,000 lb. NH_3, a quantity quite sufficient for a daily output of 35 tons of finished soda ash. Sixteen tons of limestone were also burnt per day. A particular feature in this plant was that the vessels were so placed that the liquor passed by gravity from vessel to vessel so that pumps were not required. This works was not a success.

At Middlewich, in 1889, a plant very similar to the one already described was erected, but on a much more extensive scale. It contained two large ammonia stills very similar in construction to the Mond still mentioned in the Solvay system, except that the superheater was connected by pipes instead of being built in one column. It also contained four sets of carbonators similarly worked as at Runcorn, and four lime kilns. With a little ingenuity this plant might have produced at least 600 tons of soda ash per week; with the whole plant working only 300 tons were made and with half the plant but 200 tons of soda ash were produced per week.

To illustrate this point two examples will be given. When the whole plant was working and contained a quantity of ammonia equal to 123,000 lb. NH_3, 104 tons of limestone were burnt, 16,000 cu. ft. of residual liquor distilled, 15,000 cu. ft of brine used; 55.8 tons of soda ash were produced in the day of 24 hours. To effect this about 20 carbonators were worked, each containing 840 cu. ft. of ammoniated brine.

Another example when only half the plant was working and contained a quantity of ammonia equal to 57,000 lb. NH_3: 6600 cu. ft. of residual liquor were distilled, 6600 cu. ft. of brine used; 23.68 tons of soda ash were produced; about 10 carbonators were worked off. The loss of ammonia in the working of this plant varied from 3 per cent to 15 per cent, that is, parts of ammonium sulphate per 100 parts of soda ash. This works in the year 1893 with a capital of some £180,000 lost £10,680 on the year's working.

Another works with about the same amount of capital paid only 7½ per cent dividend. In fact, three works of very similar construction were bought up by Messrs. Brunner, Mond & Company, Ltd. If these works had used the Solvay tower or had used more engineering skill in the construction and working of their plants much more favorable returns would have been made.

Similar intermittent plants have also been worked in different parts of Germany about 30 years ago, all of which are said to have failed. None of these intermittent systems could ever succeed against the Solvay system and the price of alkali need not have been lowered to £2.175 6d. per ton, as it must have cost more than £4 per ton to produce soda ash by them.

Islip, Oxford

Cyanidation of Ores.*

By Dr. Wilbur A. Hendryx.

The chemistry of the cyanide process is so well known and universally accepted as commercially correct that I will pass it over, as the object of this paper is to deal with the mechanical handling of ores and the different apparatus employed.

For convenience sake I will divide the ores into four classes:

1. The silicious ores, requiring amalgamation and cyanidation only.

2. Ores containing, besides silica, iron, lead, zinc and copper sulphides, etc., in which amalgamation and cyanidation recover a large percentage of the gold and silver contents. (The precious metals being physically combined.)

3. Ores containing sulphides in which a portion of the gold and silver is chemically combined and at the same time carry a larger percentage of cyanicides, which can be removed by concentration before subjecting them to cyanidation.

4. The sulpho-tellurides, which up to the present time require roasting to put them in a condition to be transferred from the fourth class to the first.

It has been my practice for years to grind all ores and tailings which are to be cyanided to a sufficient degree of fineness to permit the solutions to come readily in contact with the metal which is to be extracted, whether it was ¼ in. or 200 mesh.

I will first refer to the mechanical devices and arrangement of the same for the class of ores arranged under the head of class 1.

I preferably use two rock breakers and deliver to the stamps a product not larger than 1 in. The stamps most commonly used weigh 1050 lb. each and drop 100 to 105 times a minute. A 12-mesh to 16-mesh screen is used, thereby giving from five to eight tons to the stamps per 24 hours, with approximately four tons of solution to one ton of ore. I use the word "solution" instead of "water" because I invariably find where wet crushing is resorted to a weak alkaline solution of cyanide (1/40 of 1 per cent or less) increases the amalgamation of the gold and silver, except where it is prohibitory owing to the presence of a large amount of cyanicides, necessitating concentration, as in the case of ores I have placed in the third class. Not only is amalgamation increased by crushing in cyanide, but considerable gold is put into solution at the battery; consequently, as a general rule, the best practice is to begin cyaniding at the battery.

Dropping the subject of wet crushing for a moment and taking up the ores of class 4, namely, sulpho-tellurides which require roasting, my experience has led me to believe that the most economical apparatus on the market to-day for reducing the ore to the proper mesh dry, after the rock breaker, before it goes to the roasting furnace, is that type of mill known as the Bonnot mill or Kominuter. While they are not adapted to wet grinding they are particularly efficient in dry grinding. Of course, there are many varieties of rolls, some of which are also adapted to dry grinding.

Referring again to wet crushing with stamps I believe up to this point we have reached the greatest capacity and economy. The next step after the stamps is amalgamation if coarse gold is present. After the plates a dewatering sizer is used so that the ore which has been ground fine enough to permit the solutions to come in contact with the gold and silver is classified out and taken to a storage settler, preferably a Hendryx combination agitator and filter, which in that instance acts as a

*Abstract of a paper presented before the Scientific Society of Colorado.

storage settler (up to the time of its having received its proper charge of solids, when it is then used as an agitator and filter) together with three-fourths of the solution from the battery, while the coarser, or over-size, is fed to a tube mill and ground to the proper fineness and discharged into the same storage settler, or agitator, acting as a storage settler, that the fines from the classifier were discharged into, where sands and slimes are brought together for succeeding treatment without further separation, thus permitting the reduction of the amount of solution before the ore is agitated to the proportion of one, two or three to one, by weight if so desired.

I have never found an ore in which the values could be extracted by leaching in two, four, six or eight days that as good or better extraction could not be made by proper and thorough agitation and aeration in the same number of hours. For that reason it is my practice to agitate sands and slimes together.

In placing the tube mill in the position above indicated in the mill I do not wish to be understood that all ores must be tube-milled, or finely ground, but that the great majority do I think is generally conceded.

The tube mill, or pebble mill, has within the past few years been brought to such a state of perfection that, in my opinion, it is far more economical to use a coarse screen in the battery and get the full efficiency of the stamps and finish with the tube mill. I prefer to use the tube mill because it has many advantages in fine grinding, such as dispensing with fine screens, which cause a continuous expense and which become clogged with finely ground wood pulp and other debris mixed with the ore, thereby reducing the capacity of the battery; the silex lining and flint pebbles do not cause a loss of cyanide as does the finely ground metal produced from the wearing parts of the iron or steel grinding machines. The wear of pebbles is much less in number of pounds and at a much less cost per pound than steel.

When sufficient ore and solution (say two to one) has accumulated in the agitator, acting as a storage settler, potassium cyanide is added to bring the solution up to a strength of from ½ lb. to 1 lb. per ton of solution for the average gold ores and not to exceed 1½ lb. per ton of solution for the average silver ore, and ½ lb. to 1 lb. of protective alkalinity (usually procured from calcium hydroxide), then the agitator is started and the ore pulp is agitated from two to eight hours for average gold ores and from eight to twenty-four hours for average silver ores. I have found many more ores that can be treated in two hours than those requiring twenty-four or more hours. During this period of agitation a second combination agitator and filter is used as a storage settler while agitator No. 1 is doing its agitating and filtering.

We have now only treated of that portion of the solutions which have remained in the agitator and are to be used for dissolving of the remainder of the precious metals which were not dissolved while passing through the grinding department. The surplus solution which has been taken off, if found to contain sufficient of the dissolved values and which has accumulated in the precipitation tank is mixed with zinc dust and passed through the zinc dust press, and having left its gold and silver values in the press is now termed "a barren solution." It is either run to the battery supply tank or to a barren solution tank above the agitator, where it is conveniently stored for the first wash of the tailings after the rich solutions have been filtered off from the agitator.

When the dissolving is completed in the agitator 80 per cent of the solution is drawn off through the filter cells and bottom canvas of the agitator and run to the precipitation tank above referred to, then the barren solution is drawn from its above-mentioned tank and introduced as a wash underneath the canvas in the agitator and through the filter cells and the agitator started and the whole mass put in a perfect mix. The agitator is then stopped and the filter cell cocks are opened and another 80 per cent of this solution, which is equal to 16 per cent of the original solution, is taken off. Then the same amount of fresh water that is to be finally lost in the tailings is added to the agitator on top of the settled charge and is drawn downward through the bottom canvas, displacing the remaining barren solution which was mixed with the remaining rich solution, or the charge is again mixed and discharged to the dewaterer.

By either procedure nearly the whole dissolved values have been recovered. This operation is made possible owing to the perfect homogeneous mix procured in this agitator. When the agitator shaft is stopped all the granular matter settles regardless of its mesh and covers the bottom canvas, thus forming a most perfect sand filter next to the canvas which will remain free from slimes sufficiently long to permit an ordinary suction pump to draw out the remaining solutions so that the remaining moisture in the pulp will not contain more than 20 per cent of moisture. (Fig. 1.)

The tailings are then ready to receive sufficient water (say 50 per cent) to allow them to be discharged from the agitator. They may be either discharged to the waste dump, if abundance

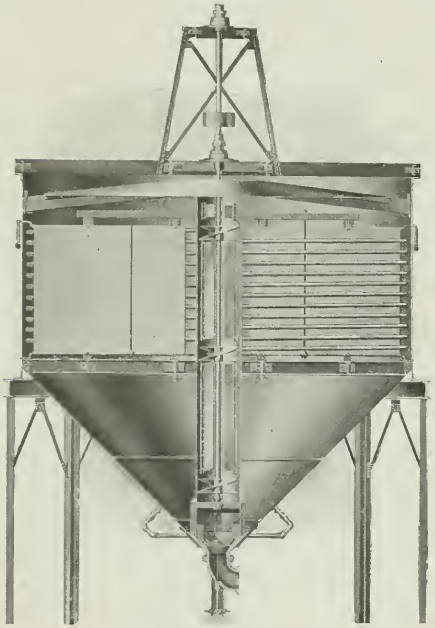


FIG. 1.—COMBINED AGITATOR AND FILTER.

of water is accessible, or if there is no objection to their flowing to the dump. In the event of greater economy being desirable owing to the lack of water, or owing to the fact that the tailings must be stacked in order to keep them from contaminating streams or flowing on to adjacent property, then they should be discharged into the dewatering device. (Fig. 2.)

After the addition of the wash water above mentioned, if it has been mixed with the remaining 4 per cent of the remaining pregnant solutions and barren solution, it should then be discharged to the dewaterer. The gold and silver filtered solutions are passed to the zinc precipitation tanks and pumped through the zinc dust press and returned to the battery storage tank or to the barren solution tank above the agitator. If there is not found enough alkalinity to protect the cyanide remaining in the solutions after passing through the zinc dust, which is often the case in heavy sulphide ores, caustic soda or hydroxide of lime should be added to the battery solution tank from time to time in order to maintain a constant alkaline condition of the solution.

My experience has led me to adopt precipitation by zinc dust because the cost of operation is extremely low and besides almost complete precipitation is possible from a solution containing 1 lb. of cyanide or less per ton, thus making it possible to carry out the general scheme of using weak solutions all through the mill. The use of weak solutions lessens the liability of fouling, which occurs to a greater extent when strong solutions are used.

I do not believe any one who has had considerable experience in electrolytic work will presume to make the claim that a solution containing metals can be rendered barren of its metallic contents economically. On the other hand, there is a class of ore in which the solution becomes so fouled that it is extremely difficult, if not impossible, to successfully and economically precipitate the values on zinc. I have found that the introduction of electrolytic deposition in combination with zinc dust tends not only to regenerate the cyanide, but will deposit out the base metals, together with the gold and silver, leaving the solution in condition so that when passed through the zinc dust complete precipitation of the remaining gold and silver values is effected.

Combination Agitator and Filter

By referring to Fig. 1 it will not be necessary to describe the plain agitator, as the only difference between the two is the addition of the filter cells and the bottom canvas in order to convert the plain agitator to the so-called combination agitator and filter.

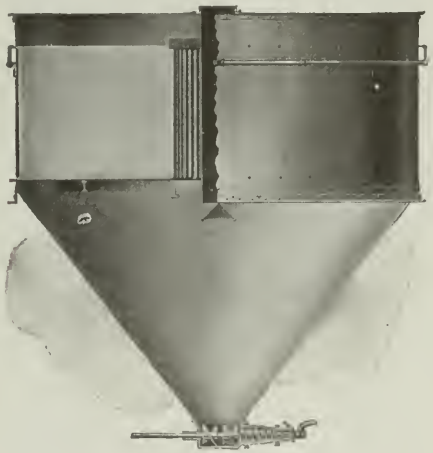


FIG. 2—DEWATERER.

The Hendryx combination agitator and filter consists essentially of four parts: a tank with converging bottom within which are centrally located an agitating pump, a series of filter cells, and a system of coils of pipes for heating the solution.

The tank is made of sheet steel and has a 45° bottom. A quick-acting discharge valve is located at the apex of the sloping bottom. Over the bottom of the tank is fastened a canvas sheet, which serves the purpose of evenly distributing the inflowing wash solution and allows the displacement of the solution remaining in the ground ore either upward or downward.

The agitating pump consists of a large well, within which is located a shaft carrying propellers. At the top of the well is a distributing and aerating apron which extends nearly to the sides of the tank.

The filter cells are located vertically beneath the distributing and aerating apron and consist of a wooden frame with iron tees and elbows. A piece of cocoa matting or wood strips fills the inside of the frame. The entire frame is enveloped in canvas, and the canvas and cocoa matting are quilted together at intervals. The filtered solution flows from a tee at a lower corner of the frame, while a tee at an upper corner serves the purpose of introducing air or wash solution to clean the cell.

The method of operating of the machine is somewhat as follows, although it may be varied to suit individual cases: Ore and solution in the proportion of about two of solution to one of ore, by weight, are accumulated either in the agitator or run in from storage until the tank is filled to nearly the bottom of the distributing and aerating apron. The agitating pump is then started and agitation continued until the soluble gold and silver have been put into solution. At the same time the solution is heated to about 90° Fahr. by means of hot water or steam circulating through the coils of heating pipe.

After the charge has been heated, agitated and aerated for a suitable time agitation is stopped, and the separate outlets to the filter cells are opened, allowing the solution to filter off by means of gravity. Owing to the perfect mixing in the agitator, when the power is shut off, a perfect classification of coarse and fine particles of ore takes place, so that the canvas in the bottom of the agitator has an ideal sand filter covering. This permits of the introduction of wash solution on top of the charge.

After filtering off approximately 70 per cent to 80 per cent of the original solution through the vertical cells and through the filter on the conical bottom, then by use of an ordinary suction pump, a downward displacement can be quickly accomplished. The downward displacement requires much less volume of water or displacing solution than the upward displacement and less time in the great majority of cases.

By the use of violent and rapid agitation and heat the slime is coagulated or curdled, so that it settles rapidly. Vacuum or pressure is not required on the filter cells, the solution filtering off by gravity alone.

Mechanical agitation is the only method of agitation that puts the charge in this condition, and the only method which gives the shortest time for dissolution, as the mechanical agitator with the same horse-power will circulate the ore pulp ten times as rapidly as the air agitator.

Air agitation also causes emulsification of the slime, thereby retarding filtration, prohibiting settling of the charge for decantation, introducing an excess of air containing carbon dioxide, which is a cyanicide, also permits of the introduction of oil from the use of a compressor, which still further emulsifies the solution and forms hydrocarbons, another source of cyanide loss.

A machine of this character capable of agitating and filtering 100 tons of average gold ores per day requires only 6 hp. With this machine the solutions can be readily changed; that is, 80 per cent or less as desired of the first solution can be taken off and a fresh solution added which, it has been found in the treatment of some of the silver ores, decreases the time of treatment as much as 25 per cent.

Storage or Tailings Dewaterer.

This dewaterer is designed to remove the water or solution from concentrates or tailings and deliver the same in such condition that they can be conveyed away on a belt or in a car and stacked. The solution or water is recovered in a clear condition and returned for further use.

It is also used in the case where only one agitator is installed to thicken slimes and to collect and hold the ground ore until such time as a sufficient tonnage has accumulated to make a charge for the agitator, when it is discharged. The excess solution to the dewaterer from the crushing department having been recovered and returned to the grinding department.

Where the water supply is limited, with this dewaterer the mill capacity will be more than doubled besides making a closer saving of the values and preventing the loss of cyanide and chemicals in the discharged tailings.

This machine without the use of vacuum or pressure will de-

water 200-mesh tailings down to from 12 per cent to 20 per cent moisture.

This apparatus consists of a round tank with a conical bottom. Within the tank are placed vertical filter cells for the purpose of clarifying the solution or water. At the bottom of the cone is an outlet within which is a screw connected to a shaft, which is revolved.

The tailings as they settle are removed from the conical bottom and forced through the outlet by the revolving screw in a continuous stream. The screw is run at a very slow speed, hence there is very little wear on any of the working parts.

This machine is used after the combination agitator and filter in order to recover the wash solution and fresh-water wash, thereby making the highest possible saving of chemicals and also shortening the time in the combination agitator and filter, and thereby increasing its capacity.

New York City.

The Analysis of Aluminium Alloys.

By Thorn Smith.

The methods for the analysis of aluminium alloys as given in many textbooks are crude and inapplicable, so much so that one is very apt to conclude that the writers have had no experience and are merely describing methods which they think are applicable. Much annoyance and loss of time follow when their application is attempted by the chemist who but seldom has occasion to use them. Time and again the estimation of aluminium by precipitation with ammonia has been proven worthless in the hands of the ordinary chemist, yet textbook writers persist in giving the method first place; in fact, it is generally the only one given. To suggest that 1 gram of aluminium can be weighed out, dissolved in caustic soda, made acid wth hydrochloric, and precipitated as hydroxide with ammonia, then filtered, washed and ignited, is beyond reason and, to speak plainly, ridiculous. So much has been written on this method of procedure as usually carried out that further comment at this time is unnecessary.

It is sufficient to say that at the present time there is no method for the direct estimation of aluminium in alloys high in this element and that the amount present as obtained by difference is the best that can be done. Therefore, if even the approximate percentage be desired, it is essential that care be taken in estimating the other elements in the alloy and that the chemist have a full understanding of their solubilities and interferences. Considerable latitude must be allowed as the various alloys are radically different and the procedure may be greatly shortened in many cases. New methods are not common, but the older methods can in many instances be shortened and presented in a new light. The present article is written to help those who but seldom have use for such methods and at the same time to furnish thinking points for those who have occasion to analyze these alloys frequently.

Analysis of Commercial Aluminiums.

The two impurities present are silicon and iron. Neither of these present any difficulty if ordinary precautions are taken.

Silicon.—Dissolve 1 gram of the drillings in 30 cc of the following mixture: Nitric, 1 part; hydrochloric, 3 parts; sulphuric, 2 parts, and water, 4 parts. A 4-in. porcelain casserole well covered is the best for dissolving the drillings. When violent action ceases, add an additional 5 cc of sulphuric acid and take rapidly down to white fumes. This is quickly done when the solution is nearly evaporated by rapidly rotating the casserole over an open burner. Taking down to complete dryness or evaporating off the excess sulphuric acid is unnecessary. Allow to cool and add 50 cc of cold water, stir, and boil until the sulphates are in solution, a point easily noted. Filter *at once,* washing with hot water, containing a little HCl, several times and finally with hot water alone until the filtrate is free from chlorides. Ignite the moist precipitate in an unweighed platinum crucible and after the paper is destroyed blast for at least five minutes. Cool and weigh. To the crucible add 1 drop of sulphuric acid and 10 drops of hydrofluoric acid. Evaporate on the hot plate to the drop of sulphuric acid, then heat cautiously over a burner until the sulphuric is expelled. Blast five minutes, cool and weigh. Factor 0.4693 equals silicon. There should be no appreciable residue left in the crucible.

Iron.—Contrary to the usual manner of estimating the iron by reducing the silicon filtrate and titrating with permanganate a separate portion of from 2 grams to 5 grams is weighed into a No. 3 beaker and from 10 cc to 20 cc of 1 to 1 hydrochloric acid cautiously added. When in solution boil a few minutes and add a drop or two of stannous chloride to reduce any iron that may have been oxidized. From this point titrate with normal twentieth, or weaker, permanganate solution, using a titrating mixture and following some definite method of procedure, preferably that used by the United States Steel Corporation, as described in the February, 1909, number of this journal (Vol. VII, page 65). It is important that the permanganate be standardized in the same manner. Another point of no less importance is the titration of an iron-free blank and deducting this figure from all titrations even in standardizing. With these precautions the results are accurate, providing titanium is absent.

Aluminium Alloys.

The one most commonly in use contains from 5 per cent to 10 per cent of copper in addition to the impurities mentioned above.

The silicon is estimated in the same manner. For the iron proceed as above until solution is effected, dilute slightly with water, filter and wash two or three times with hot water. Heat to boiling, reduce with tin solution and titrate as above. The little copper that goes into solution does no harm. The iron usually present in the alloy is so small and the solution so strongly acid that the amount held up by the copper is negligible.

Copper.—A 1-gram or 2-gram weight is taken, depending on the amount of copper present. Place in a No. 3 beaker, add 5 grams or more of sodium hydroxide and enough water to cover. When action ceases add 200 cc of hot water and boil five minutes or more. Filter and wash once with hot water. Dissolve the precipitate on the filter with 1 to 1 nitric acid, allowing the filtrate to run into the beaker in which the alloy was dissolved. Wash the paper three or four times, then pour on ammonia until the solution is deep blue. If the filter paper is colored blue, due to copper, wash again. To the blue solution, which should not exceed 125 cc, add 2 cc each of sulphuric and nitric acids. This amount should make the solution acid. Electrolyze.

Columns might be written on the electrolytic method of determining copper, so confusing are the textbook methods. Suffice to say that given a copper solution which may contain any or all of iron, aluminium, zinc and manganese the plating process is simple. Whatever form of cathode is employed the main precaution to be observed is the avoidance of too heavy a current, which will cause sponging or blackening of the deposited metal. The voltage may be ignored providing it is not too low; in other words, a 125-volt, direct-current lighting circuit may be used successfully. The rate at which the electrolyte gases is the best measure of the amount of current to use and a little experience is sufficient to guide one.

While for first-class work the electrolytic method is to be preferred, there are times when the chemist may prefer to use the cyanide method. To this there is no objection providing a strict method of procedure be followed. In dissolving the copper precipitate use 10 cc of 1 to 1 nitric acid in every case. Wash the paper thoroughly and boil off the nitrous fumes. Cool and add 10 cc of ammonia. Make up to a definite volume, preferably 100 cc and titrate with KCN, 1 cc of which equals 2/10 per cent of copper. In standardizing the cyanide the metallic copper should be dissolved in 5 cc of nitric acid and

made alkaline with 10 cc of ammonia. The volume should be the same as before. It is best to titrate to a faint pink color rather than to the complete discharge of the blue color. The addition of the cyanide should be very slow as the end point is approached. The presence of iron, etc., in the solution does no harm and it is a waste of time to remove them. The main precaution is that the operator is sure that he has all of the copper in the solution which he is titrating that originally existed in the alloy. The addition of hydrogen sulphide to the solution before separating the metallic copper by filtration is advisable. If much sulphide is thrown down it is well to use a little bromine water with the 1 to 1 nitric acid. The same amounts of acid and ammonia should be used in every titration and in titrating the operator should not go too far. Much time may be saved by having a standard solution of copper on hand made by dissolving 2 grams of copper foil in as little nitric acid as possible and diluting to 1 liter. Determine the copper in 25 cc of this solution by the electrolytic method. In using this solution as a standard do not omit the addition of 5 cc of nitric acid.

When the alloy contains *zinc* in addition to the iron and silicon proceed as before for these elements. For the zinc dissolve 0.5 gram to 1 gram, or more, of the alloy in dilute hydrochloric acid, boil a few minutes and cool. Make up to a volume of about 100 cc, add three drops of methyl orange, then sodium bicarbonate, until the solution is just straw color. Now add 20 per cent formic acid, drop by drop, until the solution again shows acid, then add five drops in excess. Dilute to 200 cc or 300 cc with hot water and heat to very near boiling. To this hot solution run in hydrogen sulphide until precipitation is complete. Allow the precipitate of zinc sulphide to settle, filter and wash by decantation three times, using hot water. Wash the paper to remove aluminium, then throw back into the beaker in which the precipitation was made; add 50 cc of hot water and 10 cc of hydrochloric acid. Boil until the zinc sulphide is completely dissolved. Filter, washing with hot water containing a few drops of hydrochloric acid. Boil off the hydrogen sulphide and add a few drops of bromine water to oxidize any possible iron. Cool slightly and add ammonia in excess. Boil and filter. If the precipitate is more than a trace reprecipitate. Now make the solution slightly acid with hydrochloric and add a *large* excess of sodium or ammonium phosphate. Make just alkaline to litmus paper with ammonia, then add 1 cc of strong acetic acid. Warm until the zinc phosphate is crystalline in appearance, which requires but a few minutes. Allow to settle, filter and wash by decantation several times with hot water before adding the precipitate to the filter. Dry the paper and precipitate, detach the latter and burn the paper in a porcelain crucible. Then add the precipitate and ignite very gently at first, increasing the heat gradually and finally heat to bright redness for five minutes. Factor 0.4291 equals zinc

This is at present the most accurate method we have for the determination of zinc. Too much emphasis cannot be laid on the use of an excess of the phosphate salt.

Where strict accuracy is not desired titrating with ferrocyanide is very satisfactory, the results depending on the experience of the analyst with the method and the strict observance of uniformity in all operations.

When copper is present with the zinc it is better to dissolve a weighed portion in sodium hydroxide, dilute to 200 cc and heat to boiling. Precipitate the copper and zinc together as sulphides. The presence of the copper permits the filtering of the mixed sulphides without the usual running through the filter paper. Wash once with water containing hydrogen sulphide to remove the greater part of the aluminum. Place filter and contents back into the beaker in which the precipitation was made; add 50 cc of hot water and 10 cc of hydrochloric acid. Boil to dissolve the zinc sulphide and filter. Wash with water containing a few drops of hydrochloric acid and a little hydrogen sulphide. Boil off the hydrogen sulphide, add bromine water and proceed as before, making a double separation with ammonia if the aluminium is in much evidence. From this point proceed as above.

The estimation of *manganese* is frequently desirable and there is no method to equal the "Bismuthate." A weighed portion up to 5 grams is dissolved in sodium hydroxide, diluted, boiled and filtered. The manganese is on the filter. Dissolve by pouring on 50 cc of nitric acid, sp. gr. 1.135, containing a few drops of sulphurous acid. From this point proceed with the bismuthate method.

Tin, antimony, lead, nickel, titanium and even other elements may be found at times, depending on the source of the alloy or the use to which it is to be put. None of these present material difficulty and can be estimated by well-known methods.

In conclusion, the chief causes of poor results are: The impossibility of securing an even approximate result by estimating aluminium in the usual manner. This is increased by the use of glassware in precipitating.

The attempt to estimate iron on too small a weight of the material. The attempt to precipitate and filter zinc from an alkaline solution in place of the newer and better method using formic acid.

The failure to adhere strictly to methods as given and making no departure until convinced by repeated experiment that harm will not be done.

Note.—The reader is referred to the following publications for more detailed descriptions of methods mentioned in this article: *Bulletin* 304 U. S. Geolog. Survey; *Journal* of the American Chemical Society, volume 26, page 4, and volume 29, page 262. Blair's "Chemical Analysis of Iron" for the bismuthate method.

Detroit, Mich.

Formulas for Pinch Phenomenon.

By Carl Hering.

The internal electromagnetic force, which acts to contract the cross-section of a conductor, due to the current, and which is now known as the pinch phenomenon, occasionally enters into calculations, especially in connection with electric furnaces in which there is a liquid resister in which such relatively small forces can act and must therefore be considered. In such cases it is desirable to have the formulas for calculating it in convenient form for direct use, as errors often arise in reducing a formula from one kind of units to another. The purpose of the present note is to give these formulas in their different forms.

The original formula deduced by Dr. Northrup[1] was intended for a particular purpose and is not in the most convenient form for direct application to furnaces. It is in pounds per square centimeter, a unit of pressure seldom used, and it contains as two factors the cross-section and the current density, hence the latter depends on the former.

Omitting the factor n, which is unity in the general case, this original formula is as follows:

$$P = \frac{S\,C^2}{44{,}479{,}100} \qquad (1)$$

P = pressure in pounds per square centimeter.
S = section square centimeter.
C = current density in amperes per square centimeter.

It applies strictly only to cylindrical conductors. P is the pressure at the center. The constant is a purely mathematical one and involves no empirical determinations.

In the form thus given the formula brings out the interesting fact that the pressure increases as the *square* of the current density. This, however, is somewhat deceptive in that when the current density is increased by diminishing the section then

[1] "Some Newly Observed Manifestations of Forces in the Interior of an Electric Conductor," *Physical Review* Vol. XXIV, No. 6, June, 1907, p. 474.

the pressure will not increase in that proportion because the other factor S will be decreased also. It is for this reason that the writer does not favor the use of such interdependent quantities in formulas when it is just as easy to use entirely independent ones.

By reducing it to English units and eliminating the current density, hence using only independent factors, the formula reduces to

$$P = 0.000,000,022,482,5 \frac{C^2}{S} \qquad (2)$$

P = pounds per square inch.
S = square inches.
C = current in amperes.

To reduce this awkward constant to a simpler one the cross-section may be represented in square mils and the formula will then reduce to

$$P = 0.022,482,5 \frac{C^2}{S} \qquad (3)$$

P = pounds per square inch.
S = square mils.
C = amperes.

To reduce it to the metric system it is preferable to express the cross-section in square millimeters instead of square centimeters, as the constant is then less unwieldy. The formula then becomes

$$P = 0.001,019,79 \frac{C^2}{S} \qquad (4)$$

P = grams per square centimeter.
S = square millimeters.
C = amperes.

The reason for the awkwardly small constant is that the formula involves the square of the current, which is generally a numerically very large quantity, and as the pressures are not very large numerically the constant must necessarily be very small. Calculations are therefore simplified and the constants more easily remembered by expressing the large currents C in kilo-amperes (1000 amp) instead of amperes. Formula (2) in inches and pounds then becomes

$$P = 0.022,482,5 \frac{C^2}{S^2} = \text{approximately } \frac{9\,C^2}{400\,S} \qquad (5)$$

P = pounds per square inch.
S = square inches.
C = kilo-amperes (1000 amp)

and the one in the metric system, S now being in square centimeters instead of square millimeters, becomes

$$P = 10.197,9 \frac{C^2}{S^2} = \text{approximately } \frac{10\,C^2}{S} \qquad (6)$$

P = grams per square centimeter.
S = square centimeters
C = kilo-amperes (1000 amp).

The two approximate formulas (the errors are less than 2 per cent) are now quite simple and easily remembered.

In the c.g.s. system the formula is simply

$$P = \frac{C^2}{S} \qquad (7)$$

P = dynes per square centimeter: S = square centimeters: C = current in c.g.s. units (10 amp).

Attention is here called to the fact pointed out by the writer some years ago[2] that complete rupture of a conductor can take place long before this pinch pressure has reached the hydraulic pressure at the bottom of a trough. This is due to the fact that after a certain amount of shrinking of a conductor an unstable state is reached after which further shrinkage takes place automatically and by rapidly increasing amounts, thereby causing complete rupture. This should be taken into consideration in applying the above formulas to such cases.

Philadelphia, Pa.

[2] "The Working Limit in Electrical Furnaces Due to the 'Pinch' Phenomena." *Trans.* Am. Electrochem. Soc., Vol. XV, 1909, p. 255.

Annealing of Copper and Diseases of Copper.

By F. Johnson, M.Sc.

The annealing of copper is designed to remove hardness and brittleness conferred on the metal either by cold working or by the repetition of minute stresses. Softness and toughness, which enable the metal to undergo further stresses without being dangerously embrittled, are regained.

There are many conditions to be observed during annealing, chief of which are as follows:

(1) *Correct temperature.*
(2) *Neutral or oxidizing atmosphere. An atmosphere of a reducing nature must be strictly avoided.*
(3) *Absence of foreign material from annealing furnace.*
(4) *Uniformity of heating.*
(5) *Duration of annealing.*
(6) *Method of Cooling.*

The observance of all these conditions is profoundly affected by the size, thickness and form of material which is to be annealed. Generally speaking, the thinner the material the more susceptible it is to injury by abnormal treatment.

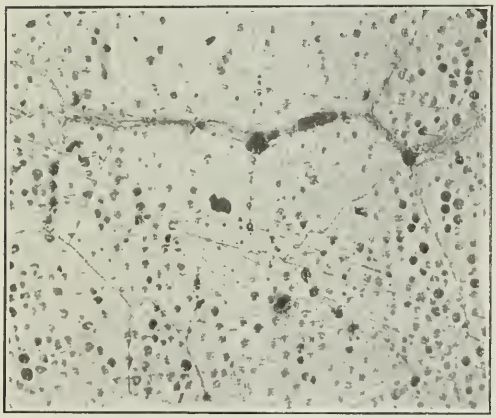


FIG. 1—WROUGHT COPPER. "BURNT" IN ANNEALING. MAGNIFICATION 100 DIAM.

(1) *Temperature.*—A bright red heat, which corresponds to about 900° C., is not too high for the purest copper. Impure copper of the refined type, if capable of being satisfactorily worked, should also easily stand this temperature.

The danger in raising copper to too high a temperature during annealing (1) lies in the occurrence of incipient fusion, but (2) is much intensified if gases of a reducing nature be present If the annealing is in the nature of a reheating preparatory to hot working, a bright-red heat may not be exceeded, for to work copper at a temperature above 1000° C. is suicidal. In the neighborhood of this temperature copper is extremely brittle and rotten, possessing neither strength nor toughness, the reason being, possibly, that the cuprous oxide eutectic assumes a pasty condition. At any rate it becomes brittle and unmalleable with an alarming abruptness, quite incommensurate with the small rise above the forging temperature.

If the annealing temperature be too low then crystalline readjustment is not complete, i. e., the hardness and brittleness due to stressing are not sufficiently removed and the material is liable to rupture if again submitted to cold work. The presence of cuprous oxide ("oxidules")* tends probably to promote recrystallization at lower temperatures than in the case of oxygen-free (deoxidized) copper.

Micrograph No. 1 shows the structure of a piece of cold-

* The author: *Journal Inst. of Metals*, Vol. IV, No. 2.

wrought copper which has been allowed to attain to too high a temperature in the annealing process, with the result that incipient fusion has been accompanied by intracrystalline oxidation causing oxidized boundaries and intracrystalline pitting.

2. *Atmosphere of Furnace.*—This should be carefully watched, as, if gases of a reducing nature, such as H, CO, CH_4, etc., all of which may be present in the products of imperfect combustion of carbonaceous fuel, are allowed to pass over copper at a red heat, or even at a lower temperature, they remove the all-important oxygen, rendering the metal porous, brittle, weak, destitute of all the properties which good copper possesses and,

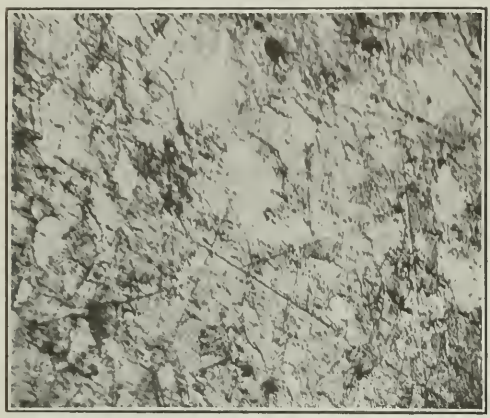


FIG. 2—WROUGHT COPPER (ARSENICAL) NORMALLY ANNEALED.

finally, amenable to no remedy short of remelting. This is commonly known as "gassing," the copper being the more sensitive to the action the higher the temperature.

3. *Foreign Material in Furnace.*—There should be no sharp grit, nor other foreign material at all on the floor of the annealing furnace, which must be kept perfectly clean. Apart from the mechanical action of such material in scratching or grooving copper, causing initial flaws which may engender failure in final tests, there is the possibility of the copper absorbing zinc from articles of brass. Brass articles should never be annealed in the same furnace and at the same time as copper articles.

Just as mercury will amalgamate with copper at ordinary temperatures, on account of the remarkable molecular fluidity and absence of viscosity of the former, so will zinc suffer a transference of its molecules to take place from the brass which contains it to copper in contact with it at a high temperature.*

So it is never advisable to calcine, in a muffle used for annealing copper, any refuse or sweepings, especially such as contain particles of brass. Otherwise the copper may take up defects, which, causing subsequent erratic behavior, may help to still further thicken that shroud of mystery which all too completely envelops the metal even now.

4. *Uniformity of Heating.*—This is of extreme importance. There are, perhaps, few works where wrough-copper material of large dimensions is annealed uniformly all over. The practice, for instance, of annealing large copper pipes over a smith's fire, bit by bit, is quite common. Where intricate shapes and sharp bends are being dealt with it is by this method impossible to insure the desirable uniformity of heating. Where furnaces are used the closed type is preferable to the open type, for, in addition to insuring uniformity of heating, all injurious gaseous products of combustion of the fuel are kept from contact with the heated metal. But whatever type of furnace is used it should be the invariable rule of the furnace man to see that every inch of metal has come to a red heat, even if it is necessary to reverse the material before withdrawal.

If this is not done the metal will vary considerably in its properties, and if it is to be put into service where, as in the case of a steam pipe, it will have to undergo the inevitable stresses due to alternate expansions and contractions, and to vibrations, it is quite certain that those parts which have been heated to the highest temperature and are, in consequence, softest, will yield most to the stresses. This localization of strains results in premature cracking and rupture of an otherwise satisfactory pipe.

5. *Duration of Annealing.*—There is no advantage in prolonging the heating beyond the attainment of a *uniform* red heat. The great danger is that the heating may be sufficient at one part and, while waiting for the remaining parts to come to the same temperature, the first part may be overheated or even "burnt."

Overheated metal, or metal which has been kept too long in the furnace, is softer than metal which has been just annealed and then withdrawn, as shown by the following figures:

Just Annealed.	*Over Annealed.*
Tensile Str. tons per sq. in.	*Tensile Str. tons per sq. in.*
15.12	14.24
See Micrograph No. 2.	See Micrograph No. 3.

Prolonged annealing induces crystalline growth and has the invariable effect of lowering the tensile strength and possibly the percentage elongation, although the latter may not be altered at all if the temperature has not exceeded a dull-red heat. Under the microscope it will be seen that the crystals have increased in size, small crystals being absorbed by larger ones.

6. *Method of Cooling.*—Copper may be quenched from a bright-red heat or cooled slowly. Quenching (1) has the effect of removing scale and retaining it in the bosh until cleaning

FIG. 3—WROUGHT COPPER (ARSENICAL) OVER-ANNEALED.

out becomes necessary. This prevents the scale, a material of considerable value, from being trodden about on the floor and lost. (2) Softens the metal and toughens it against alternating stresses. The following figures are from a paper read by Mr. J. M. Allen, January, 1909, before the N. E. C. Inst. of Engineers and Shipbuilders. The test pieces were all taken from the same sheet and were bent backward and forward by hand through 90° and over a curve of 1¼-in. radius until fractured. The figures given denote the mean number of alternations endured in two such tests:

* *The Analyst*, Vol. XXX, No. 357, Dec., 1905, L. Archbutt; also, *Journal Inst. of Metals*, Vol. III, No. 1, Bengough & Hill, p. 45.

	Sheet No.	Mean No. of times bent.	Tube No.	Mean No. of times bent.
Untreated Strips ...	S 1	36	T 1	61.5
Heated to dull red and slowly cooled.	S 10	38	T 2	92
Heated to dull red and quenched.....	S 3	39.5	T 3	99.5
Heated to bright red and slowly cooled.	S 4	43	T 4	93
Heated to bright red and quenched.....	S 11	57.5	T 11	113

Judging from these figures it would seem to be the best practice in the technique of annealing to heat to a bright-red heat and quench rapidly, thereby gaining a maximum toughness

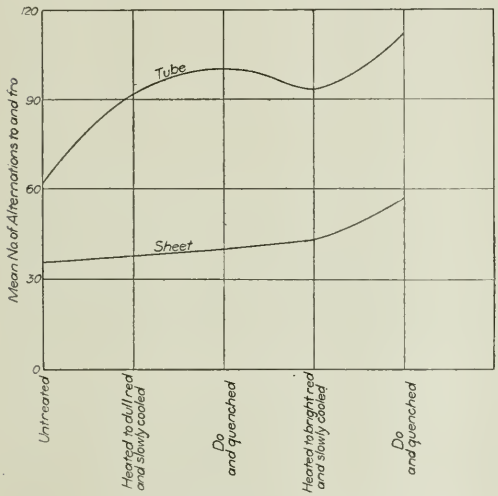


FIG. 4—EFFECT OF HEAT TREATMENT ON BEHAVIOR UNDER ALTERNATING STRESSES.

with no lowering of tensile strength or percentage elongation. The advantage of quenching seems to be the more marked (relative to slow cooling) where the copper has more lead in its composition. (As in the case of the sheet [S], see curves Fig. 4.)

Diseases of Copper.

Corrosion.

Chemical Action.—This may occur in many ways and its continuance or retardation will depend on (a) whether the product of the solvent action is protective or contributory to further wasting, (b) whether it is continuously removed so soon as formed or allowed to remain undisturbed in its protective or in its corrosive action.

In case (a) the sheet copper used for covering domes of buildings may be instanced. After exposure for some time to a marine atmosphere a coating of cupric chloride or oxychloride is formed which confers a distinct ornamental effect and is at once a protection against further atmospheric attack and an embellishment. Again, articles of copper exposed to similar action, either of sea water or other corrosive liquid, may, owing to the fact that they are in constant or intermittent motion, be continuously exposing clean surfaces to further corrosion and thus wasting gradually away.

Articles of copper exposed to a corrosive liquid and permanently removed therefrom, if not very carefully cleaned and dried, will harbor the corrosive agent in minute superficial pits or grooves and corrosion will proceed, more slowly than before and more erratically, but quite perceptibly.

Rhead ("The Corrosion of Copper and Brass," J. Inst, of Metals, Vol. 11, No. 2, p. 87) has shown that copper hardened by mechanical work is more susceptible to corrosion in acid and saline solutions than copper softened (by annealing) under similar conditions.

Sir G. Muntz (discussion on "The Failure of Non-Ferrous Metals," T. V. Hughes, Inst. of Metals, January, 1910) instanced alkaline water in boilers as the cause of intense pitting and riddling of boiler tubes.

Bengough (ibid.) stated that salt water in boilers causes corrosion on the water side of the tubes, whereas fresh water is inert, corrosion in the latter case occurring only on the fire side of the tubes. He thinks that particles of slag in the metal may form "loci" where corrosion will be first fostered, but it has yet to be proved that copper or brass tubes contain particles of slag. Even admitting the possibility of their presence in copper billets or brass castings—a possibility which is extremely remote—they would prevent the forged billet or the casting from surviving the drawing process.

Handscomb ("Characteristics of Copper under Various Conditions," Inst. Marine Engrs., September, 1909) attributes pitting in steam pipes to the ammoniacal nature of patent compounds used as heat-non-conducting coverings. In all cases of chemical corrosion the action is hastened and intensified in proportion to the temperature; the higher the temperature the more rapid the corrosion.

Galvanic Action and *Decay* have been very fully dealt with by Milton and Larke in Proc. Inst. Civil Engrs., 1903.

Gaseous Action in Course of Manufacture.—The effects of "burning," overheating and "gassing" have already been dealt with under annealing and it is extremely unlikely that wrought copper which has been seriously injured during the processes of manufacture will be allowed to issue from the works of any reputable firm of manufacturers.

Careless treatment of copper steam pipes over the coppersmith's fire may cause "gassing" of the exterior of the pipe. If only the merest superficial layer is deoxidized trouble is being brewed, for incipient cracking, even if well-nigh imperceptible, is unspeakably dangerous, since the propagation of larger cracks throughout the thickness of the pipe is thus freely encouraged and the life of the pipe assuredly shortened.

Gaseous Action (B) in Service.—Hot gases from the combustion of fuel may consist of inert gases, such as N and CO_2 and corrosive gases, such as Cl, SH_2, SO_2, COH, etc. Chlorine, sulphuretted hydrogen and sulphur dioxide must, if present in the furnace gases of a locomotive boiler, cause wastage of a serious character. They would act in the formation of a superficial layer of scale, free oxygen being probably necessary to complete the chemical equation of which an oxychloride or sulphate would be the probable product. Such a salt would be friable and non-adherent, easily shaken off by vibration or by the abrading action of cinders and dust, thus constantly exposing a fresh surface to further corrosion.

Carbon monoxide would only act on copper at a high temperature (say, incipient red) and then it would have a more than superficial effect; it would penetrate into the metal, reduce cuprous oxide present, and leave in its wake a weakened structure. It is almost certain, however, that boiler *tubes* at all events, even at the firebox end, never reach so high a temperature as to admit of the insidious inroads of reducing gases.

The abrading action of *cinders* and the "burning" action of *flame* may be noted under this head, since they are inseparable from the onward sweep of furnace gases caused by the fierce draught. Webb (Inst. C. E., 1903) showed how the red-hot cinders and ash were caused to impinge on the lower half of the boiler tube, where the latter were held into the firebox tubeplate at the ferrule, and were deflected thence in an upward direction, meeting the top half of the tube a few inches further on. The great majority of failures occurred in these two places. Cinders and flame are responsible for much of the wastage of stay-heads and "scoring" of firebox plates.

Here, too, may be mentioned the theory of local overheating (due to oil in the feed water adhering to parts of the firebox and causing a retardation of the conduction of heat to the water). Such spots would, it is thought, be subject to intense local heating, causing, on the fire side of the plate, incipient or partial fusion and subsequent weakness.

The greatest trouble experienced with circulating *water pipes* of copper is internal pitting of a local character. This has been traced in some instances to the liberation of air from the circulating water. Sea water is known to contain air in solution and if the conditions are such that the supply of water to the pipes is inconstant, they will not always be quite full, with the result that free air will be liberated, and rising to the "roof," as it were, of pipes in a horizontal position, will attack the copper under the necessarily moist, saline conditions obtaining, often in an erratic manner, causing the pipe to scale. Scale thus formed will be shaken off by vibration, and "pitting" will proceed until at length a hole is formed right through.

Ensuring that the pipes are always quite full of water has been the means of preventing further corrosion of this nature.

The presence of impurities in wrought copper may be considered as helpful, if anything, in enabling the copper to resist corrosion. Certainly the purest electrolytic copper is just as susceptible to corrosive influences as commercial "tough" copper, and possibly more so.

Swansea Technical College,
Swansea, England.

The Work of the Chemical Laboratories of the Bureau of Mines.

Dr. J. K. Clement, the physicist of the Bureau of Mines, presented an interesting paper at the recent Minneapolis meeting of the American Chemical Society on the chemical laboratory work of the Bureau of Mines.

This work is divided among a number of separate laboratories, each carrying on its own lines of work under the direction of its own chief, the whole forming a group of more or less independent units. The relation of the work of the several chemical laboratories to that of the other departments of the bureau varies with individual cases. In general, however, the problems of the chemists are closely connected with those of the mining and mechanical engineers.

The fuel testing laboratory is occupied mainly with the analysis and calorimetric testing of fuels, including coal, coke, lignite and peat. In addition to analyzing samples of all fuels used in the boiler and gas producer tests of the bureau, ultimate analyses and calorific value determinations are made on mine samples of coal collected at various places. The data on these latter tests are of value in establishing the composition and heating value of the coals in connection with the classification of the coal fields of the United States.

In addition to the laboratory at Pittsburgh, there is located in Washington, D. C., a laboratory in which are tested samples of coal, delivered to the various buildings, arsenals, navy yards and military posts within the District of Columbia and in various parts of the country, and of the coal purchased by the Panama Railroad.

Fusibilty and Clinkering of Coal Ash.—In the use of coal under steam boilers, the property next in importance to its calorific value is perhaps the fusibility of its ash. Indeed, some coals, which have a high heating value, are worthless for making steam on account of their tendency to clinker and adhere to the grate bars. The relation between the fusibility and clinkering properties of coal ash and its chemical and mineralogical composition is now being investigated.

It is interesting to note that TiO_2 was found in all the clinkers examined in amounts varying from 1 per cent to 3 per cent. Determinations of the fusion point of various ashes give values ranging from 1150° C. to 1400° C.

Chemistry of Petroleum Technology.—The bureau is making a study of the commercial bodies contained in the crude petroleums of the United States; of the methods for their separation and purification and of their economic uses. The California fields, because of their showing at this time the greater promise of a large and continued production, their promixity to naval stations and the peculiar adaptability of their product as a maritime and a locomotive fuel, have been selected for first study. Besides the determination of the properties and uses of the various products of the petroleums of the country, an investigation is being made of the processes of distillation and of the methods of refining.

Combustion Investigations.—The processes of combustion in the boiler furnaces are being investigated in a furnace specially designed for the purpose. By taking simultaneous samples of the combustion gases, the progress of the reactions may be followed and the time or space necessary for the complete combustion of various coals and under varying conditions of operation may be determined.

The process of producer gas formation is being studied from a physico-chemical standpoint, and an attempt will be made to apply on a commercial scale the results of laboratory experiments on the rate of formation of carbon monoxide and water gas.

The Composition of Coal.—Our scientific knowledge of the chemical character of coal is limited almost entirely to its chemical analysis and its adaptation to certain industrial operations. The object of one of the investigations of the bureau is the isolation and identification of some of the constituents of coal. By the use of inert solvents it has been found possible to extract as much as 35 per cent of the original coal. A number of different substances have been isolated and the analysis and molecular weights of some of these substances have been determined. In a few cases it is believed that the materials are practically pure substances.

The Volatile Matter of Coal.—The quantity and composition of the gases evolved from various coals, when heated to temperatures of from 400° to 1000° C., have been determined. In the experiments which are now in progress particular attention will be given to the influence of the rate of heating on the character of the gases produced; to the initial composition of the gases at the instant of liberation and to the thermal decomposition of these gases during passage over heated surfaces.

Weathering and Deterioration of Coal.—In co-operation with the Navy Department, the Panama Railroad Company and the University of Michigan, the Bureau is conducting an extensive series of tests on the deterioration of various coals in storage both in the open air and when submerged in fresh water and sea water.

The Accumulation of Gas from Coal.—The quantity and rate of formation of inflammable gas from freshly mined coal, at ordinary temperatures, and the rate of absorption of oxygen by the coal have been determined.

The Spontaneous Combustion of Coal is being investigated by the Bureau. Statistical information will be combined with the results obtained in the laboratory.

The Burning of Coal in Mines under a Diminished Supply of Oxygen.—The factors governing the propagation or extinguishing of fires in mines are being investigated. Chief among these are variations in temperature and in the oxygen content of the surrounding atmosphere.

Examination of Mine Gases.—Examination is made of samples from normal mine air, from the after-damp following explosions, from stagnant areas and from burning areas during mine fires. Particular attention has been given to the detection of small amounts of carbon monoxide. By analyzing samples during the progress of mine fires, the chemist has assisted in combating fires. The effect of variations of barometric pressure on the exudation of methane and the influence of carbon dioxide on the explosibility of mine gases are being investigated.

The Chemistry of Explosives.—Chemical analyses are made of all explosives submitted to the Bureau for test, of the products of combustion of explosives and of electric detonators, blasting caps and fuses. All explosives, blasting caps, electric detonators and fuses purchased by the Isthmian Canal Commission are inspected by representatives of the Bureau, and all shipments of such explosives are sampled and analyzed.

Coal Dust Explosions.—The two greatest sources of danger encountered in mining operations are the explosive gases given off by the coal, and the finely divided coal dust which exists throughout most coal mines. The first danger can be overcome by increasing the ventilation in the mines. Unfortunately, this increases the danger from the coal dust by the removal of its moisture.

A laboratory method has been devised to test the inflammable character of samples of coal dust, and to classify them according to their inflammability. This method is based on determining the amount of combustion which takes place when clouds of dust of the same density are ignited under the same conditions, the amount of combustion being determined by the pressure developed within the explosion vessel. In this way it is possible to obtain results on any one sample of coal dust which agree to 3-5 per cent of the total pressure developed.

One of the proposed means of lessening the inflammable character of coal dust is to add a non-inflammable dust. The laboratory method used to investigate the inflammability of coal dust has been extended to various percentages of coal dust and finely ground shale in order to determine to what extent the combustion is limited by the presence of the inert dust. The experiments indicate that a marked diminution of pressure is not obtained until about 25 per cent of inert dust is added, the pressure then falling off rapidly with a further increase in the amount of shale dust added.

Western Chemists and Metallurgists Disband.

The Western Association of Technical Chemists and Metallurgists, which was organized in Denver, Col., six years ago, has taken the necessary steps to dissolve the incorporation and disband. The association was formed to promote the interests of Western men engaged in chemistry and metallurgy of the non-ferrous metals and has unquestionably supplied a need that was not met by any other society in the United States.

The conduct of the affairs of the association has never been an easy task, for the Western country is large, its centers of industry few and far between, and the men in position to give time and attention to the work are widely scattered. Further, concerted action often has been difficult by reason of the itinerant character of a large proportion of the membership.

The association has published six volumes of a creditable journal, which has been the most difficult feature to maintain, and without which it would have been impossible to have attracted or held a membership. It is with considerable regret that the board of control has taken steps toward disbandment, but the action was taken only after securing an expression of opinion from the members, whereby it was concluded unwise to enter on a new year with poor prospects of a successful finish.

The Utah Society of Engineers conducted an excursion over the lines of the interurban roads recently put into commission between Salt Lake City and Brigam City, Utah, on Saturday, Nov. 26. Stops were made to inspect power plants at Willard, Ogden Canyon and Lagoon; to see the concrete-lined Weber-Davis canal and to view other engineering works of interest. This is the second regular inspection trip made this season, the first having been to the new smelter of the International Smelting & Refining Company at Tooele, Utah, on Oct. 29.

Zinc Ore-Dressing in Colorado—IV.

By H. C. Parmelee.

Magnetic Separation.

The presence of much iron sulphide in zinciferous ores necessitates the adoption of different treatment than in the case of ores practically free from such mineral. Wet dressing alone is insufficient for the separation of the iron from the zinc on account of the slight difference in specific gravity, that of blende being about 4, and of pyrite 5. If galena also is present it may be separated by the usual wet methods, its specific gravity being 7.5.

In case all three of these minerals are present two courses are open. By preliminary wet dressing the lead may be removed as a clean concentrate and a zinc-iron middling made for further treatment by magnetic or electrostatic processes; or the treatment may be reversed, the iron being removed by magnetic separation and the zinc-lead product finally separated on tables. Electrostatic separation of iron and zinc is effected without preliminary roasting, and for the details and principles of this process the reader is referred to our issue of November, 1910 (Vol. VIII, No. 11, page 636).

Magnetic separation of blende and pyrite may be accomplished without preliminary roasting by the use of high intensity

WILSON MILL, ROBINSON, COL.

magnetic separators of the Wetherill type, the zinc mineral being lifted out of the mass. But the more usual procedure is to give the ore a magnetic or "flash" roast, rendering the iron magnetic, after which it is separated from the mass by separators of the low-intensity type, such as the Dings. Manifestly the choice of machines and method of procedure depends largely on local conditions and must be determined by experiment. In Colorado both types of machines and methods of practice are employed, although the prevalent method is roasting and magnetic separation, followed by wet dressing.

The object of the roast is to convert certain non-magnetic minerals into magnetic substances so that they may be separated by means of a magnet properly applied. The conditions required for successful work are very precise and need considerable care. Either magnetic sulphide, Fe_3S_4, or magnetic oxide, Fe_3O_4, may be produced, but some experiments have shown[1] that in an ore containing pyrite, blende and galena the magnetic concentrate runs lower in lead and zinc when magnetic sulphide is produced than when magnetic oxide is the

[1] G. M. Gouyard. *Proceedings Colorado Scientific Society.*

product of the roast. Furthermore, the production of magnetic sulphide is a simpler procedure than roasting to magnetic oxide, the latter usually requiring the addition of carbonaceous material to reduce ferric oxide to magnetic oxide.

Siderite often occurs in zinc-lead-iron ores, and in the calcination process it is converted into magnetic oxide under much the same conditions which convert pyrite into magnetic sulphide. Less time and slightly higher temperature are required for the conversion of the siderite than of pyrite, but an

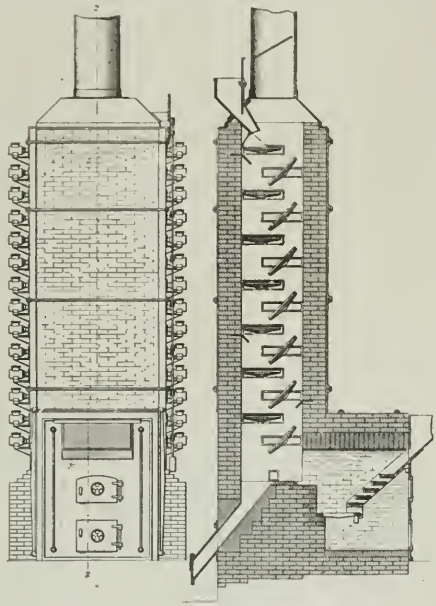


FIG. 1.—FAIRCHILD ORE DRYER, FRONT ELEVATION AND SECTION ON LINE 2-2 OF FRONT ELEVATION.

ore containing both may be treated successfully, although the same degree of conversion of both minerals can hardly be obtained. Ore properly roasted for magnetic separation will be recognized by its uniform black color and the absence of the reddish tinge which indicates oxidation. Blende will be coated with a thin layer of oxide. When these conditions are attained a magnet of low m.m.f. will remove the iron readily.

The Wilson Mining Company.

The mines and mill of the Wilson Mining Company are situated near Robinson, Summit County, Colorado. The ore is a mixed zinc-lead-iron sulphide, with some siderite, in a silicious gangue. The average composition is: SiO_2, 11 per cent; Fe, 23.6 per cent; Pb, 10.4 per cent; Zn, 14.8 per cent; CaO, 4 per cent; S, 35.6 per cent; Ag, 7 oz. per ton, and Au, 0.07 oz.

The mill has a daily capacity of 125 tons according to its present roaster equipment. The principal steps in the process are crushing and grinding, drying, roasting, magnetic separating and wet dressing. The mill is conveniently divided into units, each with ample bin room in which to store its finished product ready for treatment in the succeeding unit, thus making the operation of each unit independent of every other.

Crushing Unit.

There are no crude ore bins at the head of the mill, but ore is trammed from the mine directly to the crushing unit, where it is dumped over a grizzly (1¼-in. openings) to a Symons No. 5C gyratory crusher, reducing to about 2½-in. size. This machine is much more compact than the usual type of gyratory, and differs further in having a stationary central shaft around which revolves an eccentric imparting the crushing force to the head. Automatic circulation of lubricating oil is provided by a pump, chain-driven from the pinion shaft. About 5 gal. of oil per minute are pumped to the top of the central shaft, where the stream is divided, half passing downward between the stationary shaft and rotating eccentric sleeve and half between the sleeve and crushing head. The oil collects in a basin in which the gear is housed and is returned by gravity to the pump.

The crushed ore and undersize of the grizzly are combined and discharged by belt conveyor over a revolving electromagnet contained in the pulley of the conveyor, for the purpose of removing pieces of iron and steel which accidentally find their way into the mill feed. The ore is then sized for a second crushing by passing over a shaking launder, 12 ft. x 3 ft., having a perforated area of 1-in. holes near its discharge end. The oversize is crushed to about 1-in. size in a Symons gyratory No. 3C.

A 14-in. B. and B. elevator then raises the crushed ore to a ¼-in. mesh trommel screen, the oversize of which is returned

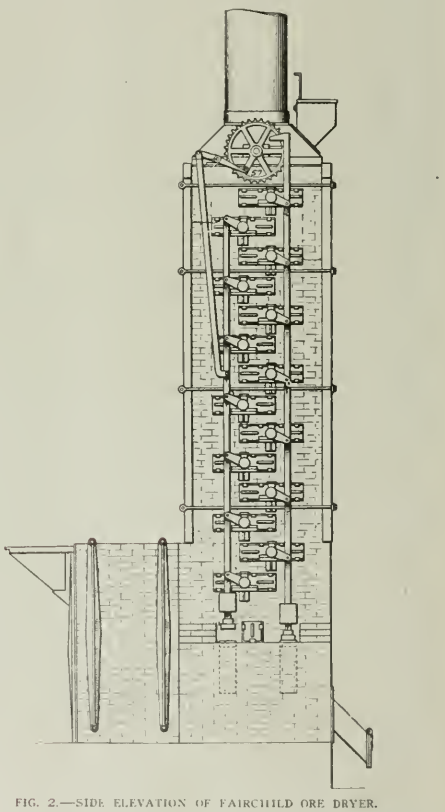


FIG. 2.—SIDE ELEVATION OF FAIRCHILD ORE DRYER.

to a set of 14 x 27 rolls set ⅜ in. apart. The roll product is returned to elevator and screen. The screen undersize, which is the finished product of this unit, flows to a point over the center of a rectangular ore bin of 1000 tons capacity, to which it is distributed by a belt running lengthwise of the bin. The belt and frame on which it moves is half the length of the bin and is operated by a reversible motor, so that ore falling onto the distributor may be dropped into the bin from

either end of the belt. The entire distributing mechanism also may be shifted by means of the same motor to occupy any position on a track extending the length of the bin. In this way ore may be conveniently distributed to any desired point.

Drying and Grinding Unit.

The crushed ore bin is provided with eight discharge gates through which ore is drawn for treatment in this unit. The gates are of the rack and pinion type, provided with a belt-driven plunger feeder. Ore thus discharged from any number of gates falls onto a belt conveyor running the length of the bin just below the gates. This manner of drawing the crushed ore from the bin insures a uniform feed for the succeeding unit, as regards size and contents.

The ore is elevated to the dryer, which is of novel construction, designed by Mr. O. H. Fairchild, superintendent of the mill. It is illustrated by drawings in Figs. 1, 2 and 3. It consists of a vertical brick chamber, about 20 ft. high and 5 ft. x 8 ft. in its other dimensions. Within the chamber are two sets of

FIG. 3—ORE DRYER, WILSON MILL.

pans so arranged and operated that each set alternately dumps its contents onto the other. As the pans of one set are staggered vertically with reference to those of the other it will be seen that ore fed onto the uppermost pan makes its progress through the dryer by a series of intermittent discharges from pan to pan. This exposes new surfaces of ore to the hot gases ascending from the fire box in the base of the chamber and thus effects the drying in a short time. There are seven pans in each set. Ore is discharged from one set to the other every 15 seconds, so that it requires 3½ minutes for ore to pass through the dryer. Ore is delivered by elevator to the hopper shown in the top of the chamber. The feed is continuous, but the pans dump when a charge of about 75 lb. has been received, giving the dryer a capacity of about nine tons per hour. The fuel consumption is about 7 lb. to 8 lb. of coal per ton of ore dried. Cost of fuel, $4.75 per ton.

From the foot of the dryer the dry ore is conveyed by traveling plow conveyor through a metal launder to the elevator which feeds a 6-mesh trommel screen. The oversize is returned to a set of 14 x 27 rolls and thence again to the elevator and screen. The undersize passes to a second trommel fitted with 10-mesh screen, the oversize of which is returned to a second set of rolls similar to the first and thence to the elevator and 10-mesh screen. The 10-mesh undersize is the finished product of this unit and is sent by belt conveyor to the dry ore bins of 275 tons capacity.

Roasting Unit.

From the dry ore bins the ore is elevated to an oscillating plow conveyor placed above the roasters, which are arranged in two rows, one on either side of the conveyor. The ore is discharged into feed hoppers on each furnace, to which it is finally fed by plunger feeders.

There are five roasting furnaces of the circular multiple-

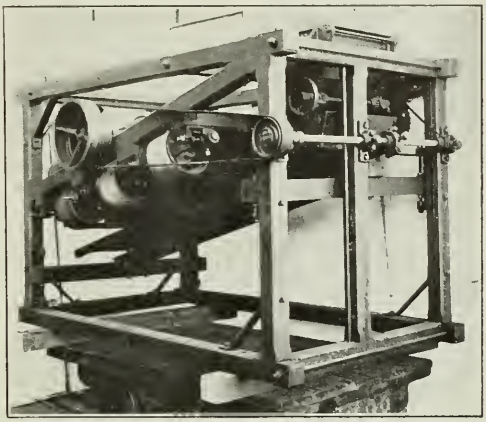


FIG. 4—FAIRCHILD MAGNETIC SEPARATOR.

hearth type, locally known as the Dewey roaster. Each furnace has six hearths, 14 ft. in diameter. A vertical central shaft driven from beneath the furnace carries the stirring arms, of which there are two, diametrically placed in each hearth. Ore is fed at the periphery of the upper hearth and plowed to the center, where it falls onto the next lower hearth and is plowed to the periphery, falling to the third, and so on through the furnace. The rabble arms make one revolution in 20 seconds, and the time required for the passage of ore through the

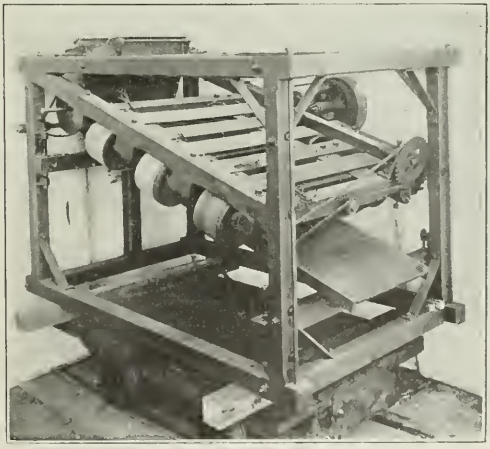


FIG. 5—REAR VIEW OF FAIRCHILD MAGNETIC SEPARATOR.

furnace is about two hours. The capacity of the furnace is 24 tons in 24 hours, making a total capacity of about 125 tons per day, which determines the present capacity of the mill.

The temperature of the three lower hearths of each furnace is indicated by Bristol electric pyrometers. An average tem-

perature of 1100° Fahr. to 1150° Fahr. is maintained, although the furnace attendant is not wholly guided by the pyrometer indication. The appearance of the ore on the lower hearth is a good guide to an experienced man, and this is often accepted in preference to the pyrometer reading and regulation made accordingly. The temperature is usually a good guide to successful roasting for magnetic separation, but changes in the physical condition or chemical composition of the ore are such that two furnaces apparently receiving the same feed may have to be run at different temperatures in order to get equally good results in the following magnetic treatment.

Furnaces of the type described are usually preferred for magnetic roasting on account of their low cost of installation, maintenance and operation, and the fact that they require less space than revolving cylinders. Considerable dust is made, and at the Wilson mill the roasters are provided with dust catchers. The roaster gas passes from the furnace through a down-take into a cylindrical iron chamber with conical top and bottom. The gas enters the settler practically tangent to its periphery so that a downward swirling motion is imparted to it. The dust particles settle into the conical bottom and the gas rises and is discharged into the air. As the roasting of the dust particles is practically completed they are returned to the lower hearth of the furnace and mixed with the discharging ore.

The roasted ore is cooled in a revolving drum, 39 ft. x 3 ft., set at a pitch of 1 in. to the foot, and making about 4 r.p.m. Cooling water is discharged over the drum from a launder placed above it. The cooled ore is elevated to an Edison 10-mesh screen for the purpose of removing the small quantity of oversize which results from the fritting together of particles in the furnaces. The oversize of the screen is reground in a set of small rolls and returned to the elevator and screen. The screened ore is then conveyed by a traveling plow conveyor to bins of 500 tons capacity.

Magnetic Separating and Wet Dressing Unit.

The magnetic separators, which also are designed by Mr. Fairchild, are placed just below the roasted ore bins, and receive a continuous feed through spouts leading to their feed hoppers. The machine is illustrated in Figs. 4 and 5. There are three sets of electromagnets, the second and third being of higher intensity than the one immediately preceding it. The voltage is 110, and the amperage 1.5, 3 and 6 for the first, second and third magnets respectively.

Each set is composed of 10 electromagnets of alternately reversed polarity, this being the principal feature of the machine. The magnets are arranged transversely to the shaking tray, and a belt travels around each set for the purpose of removing the magnetic concentrate from the machine. Ore is brought into the magnetic fields on an inclined shaking brass tray, on the under side of which are three bars of iron so placed as to be beneath the magnets. Passing under the first set of magnets the strongly magnetic particles of ore are attracted to the cross belt and removed from the machine. The same operation is performed by the second and third sets, where the weaker magnetic particles are attracted by the stronger magnets.

The object of alternating the polarity of adjacent magnets is to obtain a cleaner separation. In the process of magnetic separation it happens that more or less non-magnetic material, mineral and gangue is removed with the magnetic concentrate by being mechanically entangled with it. By alternating the polarity of the adjacent magnets in each set the particles attracted to the cross belt are caused to change position as they pass underneath the successive magnets, thereby offering the greatest opportunity for mechanically entrained matter to drop out of the mass.

At the Wilson mill there are five of these separators, each having capacity to handle the ore from one roaster. The shaking tray oscillates 325 times per minute. The cross belts have a speed of 350 ft. per minute. The power required for the mechanical operation of a machine is about ¼ hp.

The magnetic concentrate is removed by belt conveyors running beneath the floor of the separator room and finally delivered to storage bins. This product is useful as fluxing material for lead smelters, and valuable also for the small quantity of lead, silver and gold contained.

The zinc-lead-silica mixture is also conveyed from the magnetic machines by a system of belts to a central elevator which raises it to a point above the concentrating tables, where water is added and the pulp prepared for concentration. The pulp is mixed by a screw revolving in a launder, water being introduced at about the middle point of the latter. From the mixer the pulp flows into a box which provides for the immediate elimination of some slime by overflow. The heavy pulp is then classified in a pocket classifier, the pockets being conveniently arranged at points in a launder extending above the tables. Hydraulic water is introduced by means of a pipe entering the side of the pocket and standing vertically in it. The pipe is capped at the top and water is discharged through three annular series of holes drilled in the pipe. The first series is 1 in. below the cap and each of the others is arranged 3 in. below the one above it. Just above each set of holes an annular apron is attached to the pipe to spread the flow and provide a uniform rising current of water in the sorting column. The last two pockets are dewatering boxes with clear overflow and slime underflow, which is combined with the first slime for concentration.

The table equipment comprises nine Wilfley and two Card tables, one of the latter being a special slime table. Eight of the tables are used for direct treatment, and three for retreatment of middling. The two Card tables are used for the finest sizes, one on direct concentration and the other on retreatment of middling. The products are lead concentrate, zinc concentrate and tailing; the concentrates are laundered to filter bins of the usual type and the tailing to waste.

Products and Sampling.

The following analyses of concentrates represent average shipments:

Magnetic Iron: SiO_2, 6.5 per cent; Fe, 46 per cent; Zn, 3 per cent; Pb, 2.2 per cent; Au, 0.05 oz. per ton, and Ag, 4.5 oz.

Zinc: SiO_2, 7 per cent; Zn, 41 per cent to 42 per cent; Fe, 11 per cent; Pb, 6 per cent; Ag, 8 oz. per ton.

Lead: Pb, 67 per cent; Zn, 6 per cent; Ag, 34 oz. to 38 oz. per ton.

The tailing carries about 9 per cent zinc and 1.2 per cent lead and amounts to 15 per cent to 18 per cent of the ore treated.

Provision is made for automatic sampling of the crude ore and tailing. The former is sampled just above the crude ore bins after it has been screened through the ¼-in. mesh trommel. The sample represents 5 per cent of the ore. The tailing sample represents 1/230 of the pulp, and provision is made for collecting a separate sample for each eight-hour shift. The tailing sampler is a horizontal wheel with vanes on its periphery, against which the stream of pulp flows, causing the wheel to revolve. At one point on the periphery of the wheel is a ½-in. slot which receives a portion of the stream and conducts it to the sample boxes. The slot bear a ratio to the periphery of the wheel of 1:230.

Power, Water and Labor.

The mill is electrically operated throughout, power being received from the Central Colorado Power Company at 10,000 volts and transformed to 440 volts. The distribution of motors is as follows:

Crushing Unit: For crushers, rolls, elevator and screen, 75-hp motor; maximum demand, 43 hp. For distributing ore to bins, 2-hp motor.

Drying and Grinding Unit: For dryer, rolls, elevators, screens and conveyor, 75-hp motor; maximum demand, 32 hp.

Roasting Units: For roasters, cooler and elevator, 25-hp motor; maximum demand, 14.5 hp; on cooler only, 6 hp. For conveyor and elevator, 25-hp motor; maximum demand, 9.6 hp.

Concentrating Unit: Motor generator, 10 hp; maximum demand, 11.2 hp. For tables, conveyors, etc., 25-hp motor; maximum demand, 17.7 hp; or with centrifugal pump, 23.7 hp.

The water supply is from Ten Mile Creek, and is pumped by an electric driven duplex piston pump to a 50,000-gal. storage tank 180 ft. above the creek. This pump is operated by a 15-hp motor. The pumping station is also equipped with a centrifugal pump operated by a 50-hp motor. Some of the mill water is reused, viz., that from the concentrate bins. The bins are constructed so that the filtered water is laundered to a sump, from which it is returned to tanks in the mill by a centrifugal pump having a capacity of 180 gal. per minute and operated by the motor running the tables.

The labor required in the operation of the mill is five men per eight-hour shift when crushing is done, which is usually but one shift per day. On the other two shifts only four men are needed; one for the dryer and rolls, one for roasters and separators, one for tables and a shift boss.

The Wilson mill is the latest in Colorado to apply successfully the process of magnetic separation to complex sulphide ore. The foregoing analyses of the products attest the quality of the work, and it will not be long until the plant is working at full capacity. Acknowledgment is due Mr. O. H. Fairchild, general manager of the company, for the data and information.

Magnetic Specimen-Holder for Microscope.

A specimen-holder for microscopes, particularly a magnetic holder for metallic specimens used under a high-power instrument, has recently been patented by Prof. ALBERT SAUVEUR, of Harvard University. In order to properly examine a piece of metal it is necessary that the surface be held at a plane accurately perpendicular to the optical axis of the instrument. While it is possible to prepare a specimen with two parallel sides the operation is a difficult one, and this difficulty is overcome by Dr. Sauveur by the arrangement shown in the adjoining illustrations, reproduced from his patent (977,842, Dec. 6, 1910).

Fig. 1 is a plan view of the mechanical stage of a high-power microscope equipped with the magnetic holder and showing a specimen in place, Fig. 2 is a plan view of Dr. Sauveur's magnetic specimen slide with a specimen slightly removed therefrom, and Fig. 3 is a central sectional view showing the specimen in place beneath the slide.

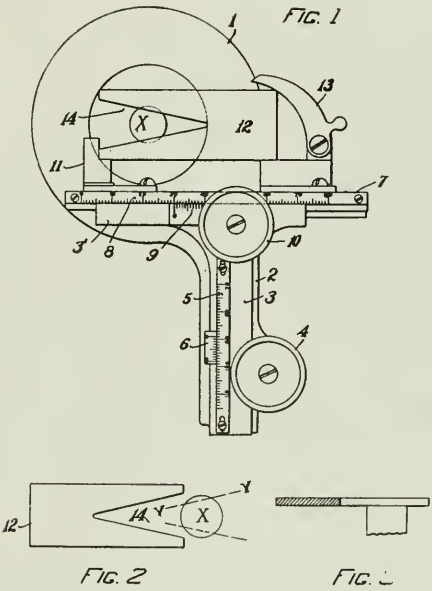
The circular stage 1 is provided with a lateral bracket 2 upon which is mounted a slide 3, being controlled by the milled wheel 4. The scale 5 reads on the vernier 6. The slide 3 carries a cross slide 7 having a scale 8 reading on a vernier 9 and controlled by a milled knob 10. On the cross slide 7 is a stop 11, having a right-angled notch, within which one edge of the magnetic specimen slide 12 may rest. The spring arm 13 bears against the corner of the slide diagonally opposite from that first mentioned, so that the slide is always clamped in exactly the same position.

The slide 12 is cut with a V-shaped opening 14, so that specimens of varying sizes may be held under it. As it is frequently necessary to re-examine a specimen at exactly the point first observed, Dr. Sauveur finds that this V-shaped opening affords the possibility for the following practice. A specimen, as indicated by X, is placed under the slide 12 in any convenient position where it has sufficient bearing to be held firmly by the magnetism of the slide and the operator then scratches on the plain surface of the specimen a pair of converging lines Y—Y, which will indicate the exact position on the V-notch 14 which the specimen is to occupy. The position will, therefore, be readily replaced at any time upon the slide and may be brought by the scale readings back to its proper position under the objective.

The Adhesion of Electrodeposited Silver in Relation to the Nature of the German Silver Basis Metal.*

BY ANDREW MCWILLIAM AND W. R. BARCLAY.

A few years ago the authors had their attention specially drawn to a rather serious trouble in connection with the behavior of high-quality electroplated spoons and forks under very severe conditions of wear. It is well known that at the present day a much larger proportion of such goods made are used in hotels, in steamboats and by caterers than formerly. Owing chiefly to the necessities of the case, but partly to the



FIGS. 1 TO 3.—MAGNETIC SPECIMEN-HOLDER FOR MICROSCOPE.

destructive instincts of virile humanity, these goods are subject to very much severer trials than they would have undergone had they found their way into use in the service of private families. For in hotels spoons and forks may have to be gathered up in large baskets and thrown into the receptacle in which they are to be washed, a kind of treatment that must be remembered in considering what follows.

A large quantity of spoons and forks, manufactured presumably under the best commercial conditions, examined and passed by experienced observers before being sent out from the factory, and apparently only differing from ordinary products in having an exceptionally heavy deposit of silver, were, after actual usage for eighteen months or so, returned to the makers with the complaint that the deposited silver coating was "peeling" or "stripping" along the edges and from parts of the surfaces of the articles.

As the trouble was not confined to a few isolated pieces, but extended to the majority of the articles in a consignment, and was also in time found to occur in connection with goods from different firms, it was felt that the matter warranted a thorough investigation, particularly as no specific fault could be traced in any of the processes through which these goods had passed, where the details of the processes could be known.

In order to correct a misapprehension which might arise, it may be necessary to point out that no attempt is here made

*A paper presented before the Institute of Metals at the January, 1911, meeting.

to discuss the ordinary causes of the "stripping" or "peeling" or "blistering" (as it is sometimes called) of electrodeposited silver from the basis metal during processes of manufacture, such, for instance, as that due to inefficient cleaning from grease or from oxide, careless treatment in scratch-brushing, or in the general preparatory stages of treatment, unsuitable plating liquid, etc., because articles so treated rarely pass from the hands of reputable manufacturers, the faults being revealed during the subsequent finishing and burnishing processes, it being well-known that if the slightest trace of grease or oxide remain on the surface of the basis metal the electrodeposited silver will blister under the pressure of the steel burnishing tools used in finishing.

The investigations were conducted from an entirely different point of view, the authors being convinced that the stripping of silver from articles which had passed all the ordinary tests of the manufacturer, and had indeed been in the hands of the users for periods varying from six months to two years, could hardly be ascribed to any of the ordinary causes mentioned.

From general experience and from the examination of many plated articles that had been a generation in wear the authors had formed the opinion that the adhesion had something to do with the nature of the basis metal, hence the investigations described in the present paper were undertaken with the view of determining not only the nature of the adhesion of electrodeposited silver to the alloys known commercially as German silver or nickel silver as a basis metal, but also to find out what difference, if any, exists between the various grades of German silver in general use in the trade, as to their suitability to form a basis metal for heavily electroplated articles which are liable to receive rough usage.

(A) Samples Tested.

The earlier experiments were made on a number of spoons and forks of various sizes supplied by manufacturers, and of the grades known in trade circles as firsts, seconds, thirds and fifths. Analyses of these were not made, but the makers reported them as having approximately the following compositions as to nickel: Firsts, 20 per cent; seconds, 16 per cent; thirds, 12 per cent; fifths, 7 per cent. The content of copper appeared to vary between 56 and 59 per cent, according to the few assays made.

In addition to these spoons and forks several test strips were obtained of each of the grades described, and of a similar character to spoon and fork blanks. All the pieces were treated so far as possible in exactly the same manner, and cleaned, prepared and electro-silverplated, according to the best methods in general use, with the one exception that a few of the samples were given a preliminary coating of copper before silverplating. The silver deposits given to these were equal to the highest qualities for high-class work, and the electroplating was done by experienced workers, under the ordinary conditions of works practice. The experiments on the above pieces were carried out several years ago, and samples were electroplated in different works. Those which will now be described have been made under the personal supervision of the authors during the last twelve months, as, in order further to test the matter, in view of the possible publication of results, they determined to prepare a series of nickel-silver alloys corresponding approximately to those in ordinary use among electroplate manufacturers.

The following table gives the compositions of the alloys accordingly made

Table I.

Marks.	Nickel. Per Cent.	Copper Per Cent.	Zinc. Per Cent.
A	20	57	23
B	18	57	25
C	16	57	27
D	14	57	29
E	12	57	31
F	10	57	33
G	8	57	35
BZ	18	50	32
BC	18	64	18

The choice of the above alloys is largely empirical, but they are fairly representative of the usual grades of copper-zinc-nickel alloys in commercial use. An endeavor has been made to systematize the series by fixing the proportion of copper on the uniform basis of 57 per cent throughout, with the exception of the last two members, BZ, in which 7 per cent of the copper is replaced by zinc, and BC, in which 7 per cent of zinc is replaced by copper.

Each alloy has been made under the ordinary conditions of works practice, manganese being used as a deoxidizing agent. The metal was cast into ingots of about 18 lbs. weight. These were rolled into sheets of a thickness and width suitable for spoon and fork blanks, and cut up into strips each 8 inches long and varying from ½ in. to 1 in. in width.

In every respect these strips correspond to spoons or forks with the exception that they have not undergone the process of stamping. A few odd strips from some members of the series have been made into forks, and these have been submitted to tests in the same way as the blank strips.

The electro-silverplating of these strips has been done under the best commercial conditions of manufacture, in an electroplating shop through which large quantities of spoons and forks are daily passing. They have been prepared for the depositing vat with due care and with all the precautions which would be taken by any careful electroplater to ensure the best adhesion of the silver deposit, but they were not subjected to exceptional treatment.

The electroplating solution used was the usual standard one of the double cyanide of silver and potassium, and the weight of silver deposited on each strip was at least equal to that given to the highest qualities of spoons and forks and to those previously described.

Table II gives the weight of deposit on the strips of each alloy, calculated per square inch of the surface.

Table II.

Marks	Deposit per Square Inch in Ounces Troy.
A	0.0196
B	0.016
C	0.0215
D	0.021
E	0.017
F	0.024
G	0.024
BZ	0.0198

The slight variation in the weight of deposit is explained by the fact that it was considered advisable to give each strip a minimum deposit at least of the quality indicated above, without in any way interrupting the course of deposition, such as by taking the specimens out of the vat for intermediate scratch-brushing, a practice often followed in the making of very heavy deposits in order to keep the coating as smooth as possible. The strips were consequently in the vat during the greater part of the working day, and naturally the rate of deposition varied as more or less work passed through the vat in the course of the day's operations.

(B) Tests Applied.

The guiding principle in designing tests for the purpose of the investigation was to subject the samples to strains such that the basis metal and its silver coating would be given, if possible, a certain amount of movement relative to one another, and in addition that the coating itself should be fractured. In all the cases of stripping of which mention has been made in the introduction the stages of development of the fault have appeared to be, first, that along the edges of the spoons or

forks concerned the silver deposit has in course of severe wear become broken or cut through, then these broken edges have gradually curled or peeled up, and if the deposited silver is thick, and therefore of sufficient substance, it may readily be pulled away from the basis metal surfaces. In some cases it has been possible by care and patience to peel the silver coating away from the fork almost from one end to the other.

With these points in view several preliminary tests were made with the apparatus available, and finally it was decided to adopt the test which had first appealed to the authors.

FIG. 1. PHOTOMICROGRAPH OF FIRST-GRADE GERMAN SILVER TEST-PIECE, COPPERED AND SILVER-PLATED. MAGN. 140 DIAM.

namely, a simple cold bending test. One part of the sample is gripped in a vise and the free part is bent to and fro until it breaks off. It is advisable that the angle of the bend should be about a right angle, in order to insure that the deposit and the basis metal get sufficient movement relative to one another.

It will be evident that this test is an extremely severe one, and that unless the metals are holding together almost as one the silver must be parted from its base, for the silver on the outside of the bend must move a greater distance than the basis metal, and on the inside of the return bend a less distance.

In view of the results it should be noted also that the order of increasing ductility of the metals being as a rule from the high-percentage nickel alloys to the low ones, the latter bearing more bending before fracture than the former, the test is more severe for metals of the G, F and E types than for A and B. In practically every case the samples were tested to destruction.

In addition to the mechanical tests applied representative sections were cut from each series for microscopical examination.

(C) Results of the Tests.

In the case of all the samples of the third and fifth grades and of the G, F and E specimens of the special alloys in Table I, it has been exceedingly difficult to peel up the silver coating in any way. In some few cases just at the point of fracture it has been possible to raise the silver coating very slightly at the top, but, in endeavoring to strip it further, the silver, though very tenacious and ductile, has broken rather than leave its base. It has, indeed, repeatedly surprised the authors to find how extremely adherent the coating is even under the severest tests.

Test pieces of first and second grades and of A and B of Table I, however, behave in a very different manner. In practically every instance the silver coating left the base at the point of fracture, and in a large majority of instances it could be pulled away quite readily from a considerable part of the surface without breaking the silver. In the cases of C and D evidence of imperfect adhesion was not so pronounced as in A and B, still the authors do not regard these as so good as E, F and G.

In the earlier experiments described a few samples of first and second grades were given rather a different treatment; they were prepared for plating in the ordinary manner, but before silver was deposited on them they were given a thin coating of copper from an alkaline copper solution, then transferred to the silver vat and given a heavy deposit of silver. These samples were then put through the same tests as the previous ones, and again there was obtained evidence of imperfect adhesion, though not quite to the same extent as formerly. The adhesion even in these cases was not to be compared, however, with that obtained with the lower-percentage nickel alloys G, F and E.

The electrodeposited silver in all the cases was tough and strong.

In order to carry the matter still further the authors quite recently obtained from another firm of German silver manufacturers a few spoons and forks of first and second grades, and the results given by these pieces when plated and tested are in every way confirmatory of the foregoing results.

A considerable number of test-pieces after plating and before being submitted to the bending tests were burnished by experienced silver finishers. This process is considered in the trade to be, in addition to a help in finishing, a reliable means of testing the adhesion of the silver. The operation is usually done by clamping the article firmly on a bench and then passing a smooth, dry steel burnishing tool over the entire surface and particularly on the edges. Unless the silver is in closest contact with its base at every point, the deposit will not pass this test without blistering. Every one of the test-pieces, spoons, forks and blank strips, used in the investigation emerged successfully from this test.

Small divergencies in results were noticed between test-pieces made from nickel silver that had been deoxidized and those made from alloy that had not been deoxidized, the former giving better results than the latter for the same grades of alloys, but the number of the latter used were so few that attention is only drawn tentatively to the result, as the authors intend to make further comparative tests in the future.

Results of Microscopical Examinations.—Samples of every type were carefully examined under the microscope at about 40 and 400 diameters, and in special cases at 1000 diameters.

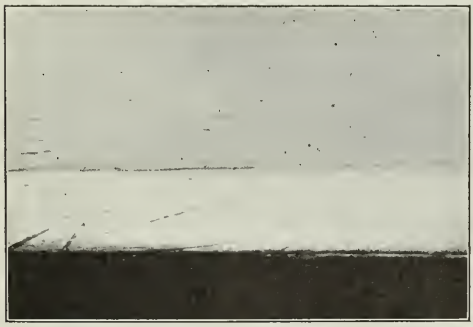


FIG. 2. PHOTOMICROGRAPH OF FIRST-GRADE GERMAN SILVER TEST-PIECE, SILVER PLATED. MAGN. 140 DIAM.

It was found that in every case, when not submitted to strain, the silver deposit appeared closely adherent, and the microscope did not reveal any marked difference between the nature of the junctions of the deposits with the various alloys. Those pieces from which the silver had lifted after being subjected to severe strain showed that the whole of the silver deposit had left the base metal, but in the case of the samples of firsts metal coppered before silverplating part of the copper had remained adhering to the silver and part to the alloy.

The photomicrograph shown in Fig. 1 represents a transverse

section of first-grade German silver coppered before silver-plating. The photograph includes a portion of the place where the mark had been struck, and shows the consequent movement of the two metals. The thin deposit of copper between the nickel silver and the silver is clearly discernible as a narrow dark band. This piece shows the three metals in closest contact throughout.

Fig. 2 shows a transverse section silver deposited on first-grade metal direct. In one place there is a slight indication of parting, evidently caused by the comparatively rough treatment by sawing and filing given during the preparation of the microsections.

Fig. 3 shows in transverse section of the third-grade metal dessert fork with a heavy deposit of silver. In this specimen, even at the highest powers, the two metals seemed to be in closest contact. The line of demarcation seen in the photograph is due to the difference in color between the metal and the alloy.

The scratches are more prominent in the photographs than the authors would have liked, but as the sections are quite sufficiently free for the purposes of their own examinations and records, they did not, when these were originally made, think of wasting any more time on them. When, however, it was decided that they should lay the matter before the members of this Institute, they repaired to the box where all the sections were stored together in order to prepare them more perfectly and a strange thing had happened.

The four most typical specimens which had been photographed had been left on the plasticene that had been used for temporary mounting purposes, and the sulphur that this useful material evidently contains had strongly attacked the samples during the three years or so they were laid away. Not only had the surfaces of all four sections been attacked, but the action extended along the junctions of the silver with the alloys so that the idea of re-preparing the samples until they were practically free from scratches was abandoned. This would really have been an unprofitable work, for the sections show all that is necessary. In the case of such a soft metal it is also a tedious one, for, in contradistinction to a recently published opinion, the authors find such materials as the comparatively hard tempered steels much more easy to polish than soft metals.

Conclusions.

The investigations clearly indicate considerable differences in the adhesion of deposited silver to the various grades of German silver alloys, even when all are treated under the same conditions of practice, and the authors are of opinion that this fact should have full consideration in choosing a basis metal for articles to receive the thickest silver deposits given, which will also be required to bear rough usage.

They are not prepared to state dogmatically that any one particular alloy is best for this purpose, but they consider they have fully proved that ordinary commercial alloys of copper, zinc and nickel containing more than 14 per cent of nickel are of doubtful utility in this respect. On the other hand, alloys of the types of G, F and E used for carrying heavy silver coatings electrodeposited with due care, will endure exceptionally rough usage without giving any cause of complaint on the ground of the silver stripping.

It should be noted that the tendency to strip increases with the thickness of the silver deposit.

The properties considered in deciding upon the grade of German silver to be used as a basis metal for electroplating are strength, color and malleability. As a general rule, passing from G to A in the series of alloys the strength increases, the color approaches more nearly the white of silver and the malleability decreases.

With regard to whiteness, however, where thick coatings of silver are concerned this should not be considered as a matter of such primary importance as the perfect adhesion of the silver under the conditions of wear. Strength is more important than color, and the lower members of the series are deficient in strength; but articles having quite sufficient strength and stiffness can be made of the type of alloy recommended by suitable mechanical and heat treatment.

With regard to the nature of the adhesion, the authors do not at present care to advance any theory to account for the differences described, but as they consider that the quicking process forms a superficial amalgam with the basis metal in the first place, and afterwards on the inner surface of the silver de-

FIG. 3. PHOTOMICROGRAPH OF DESSERT FORK OF THIRD-GRADE GERMAN SILVER WITH THICK DEPOSIT. MAGN. 140 DIAMETERS.

posit, the two amalgams may form a kind of cement between the plating and the basis alloy.

It is possible that with the more suitable alloys this very thin film of amalgam is more efficient than with the alloys of higher nickel content.

Hence, although the investigations at present in train are undertaken with regard to the general properties and structures of the series of nickel silver alloys given in Table I, it is intended to note specially any changes in microstructure among the members of the series, with the view of any possible change in the nature of the amalgam that would be produced. They also, for the same reason, intend, with their new series, to examine, with the highest powers of the microscope, a fresh set of specimens after the most careful preparation. The above idea of the nature of the adhesion is supported by the manner of the tearing of the intermediate film of copper, part of which, as already stated, adhered to the silver and part to the basis metal.

A strongly worded appeal has reached one of us to support a demand for foundation alloys of higher nickel content to be used for the thickest coatings, which, if carried into effect, as in many other cases of undue interference of the buyer with the practice of the best makers, would defeat its own purpose of obtaining the best article for the required use.

It is interesting to note how the metallurgist has to reconsider the properties of his materials and his methods of manufacture in order to be able to make articles that will do him credit in the world's work when new conditions of life arise requiring these articles to bear a different type of usage. It is no use for him to claim that the usage to which his products are subject is unreasonable, if the new conditions of life require them to bear rougher treatment or some other special strain. He has no say in regard to sociological changes, and if he be wise he will just accept the conditions and endeavor to solve the problem. This was the task placed before the authors, and the practical results of their work have stood the severest trials since their earliest investigations were completed.

The Industrial Fixation of Atmospheric Nitrogen.

From a French Chemical Society Paper by E. LAMY.

Translated in abstract by MARK R. LAMB.

Before discussing the fixation of nitrogen it will be well to recall what nitrogen is, its importance and the rôle which it plays in nature.

Nitrogen is a part of all living matter, and plants as well as animals could not do without it while continuing to live and repair their tissues.

Nitrogen appears in nature in compounds, of which the most important is the nitrate of sodium, and in the free state in the atmosphere, of which it constitutes 80 per cent by volume, the balance of 20 per cent being oxygen. The atmosphere which surrounds the earth constitutes, then, an almost unlimited supply of nitrogen placed freely by nature at the disposition of plants and animals. It is true that the latter cannot directly assimilate the nitrogen of the air, but they borrow it from the plants upon which they feed. Now, though all plants require nitrogen, only some of them, such as the leguminous, thanks to certain bacilli, called "bactériocécidées" by Prof. Mattei, can directly fix atmospheric nitrogen. Very interesting experiments made during the last score of years have clearly demonstrated this.

Certain sea weeds have this same ability to fix nitrogen.

But the above classes constitute but a small part of the vegetable kingdom. The great majority of plants, such as wheat, barley, corn, oats and potatoes, upon which man and other animals are nourished, have not this faculty. According to recent experiments a small amount of assimilable nitrogen occurs in the atmosphere in the form of nitric acid and ammonia, but in infinitely small and altogether insufficient quantities for the needs of those plants.

In order therefore that they may grow it is necessary that they draw the required nitrogen from the ground. Now the nitrogen compounds which exist in the soil in its natural state are not sufficient to induce the best growth of plant life. So it is that it has become necessary to add nitrogen to the soil in the form of fertilizer such as manure and other natural products, of which the principal ones are the guano of Peru and the niter of Chile. As farming methods are improved and as farming becomes more intensive it becomes more and more necessary to enrich the land with nitrogen fertilizers to compensate for the elements removed from the soil by vegetable life.

Guano, which is an excellent fertilizer, exists only in small quantities, the deposits being about exhausted.

Fortunately this is not true of the niter deposits discovered in Chile about 1825. The first developments were made near the ports of Iquique and Patillos and afterward on the celebrated and inhospitable desert of Atacama. Now the plants are numerous and occupy an area of 500,000 acres.

The consumption of niter over the entire world is increasing at a rapid rate. The following figures will make this clear. The exportation of nitrates from South America was 800 tons in 1830; 7000 tons in 1835; 20,000 tons in 1850; 70,000 in 1860; 150,000 in 1870; 300,000 in 1880; 800,000 in 1890; 1,300,000 in 1900; 1,500,000 in 1905, and 2,000,000 tons in 1909.

At present the principal consumers of nitrate are as follows:

France	250,000 tons
Great Britain	100,000 "
Germany	500,000 "
Belgium	200,000 "
United States	400,000 "

The price of nitrate varies between 200 and 260 francs ($40 and $50) per metric ton (2204 lb.), and at present the price is 210 francs ($42) at Dunkirk.

The larger part of the nitrate goes to agriculture, but an important part is used in various chemical industries, among which the manufacture of HNO_3 should be given first place, since in France alone this industry absorbs ten to twelve thousand tons annually. Nitric acid is obtained from niter by decomposing the latter with heat by H_2SO_4. The uses for that acid are numerous and we may cite only the manufacture of explosives, of celluloid, artificial silk, dyes, etc.

The available reserves of Chile have been the object of many and widely varying estimates. Nevertheless the deposits are considerable in spite of the statements—perhaps intentionally pessimistic—which have been given out for some time.

According to a report given me by Mr. Paul Lemétayer, professor of applied chemistry at Santiago, a technical commission of the Chilean government has recently made a study of all the nitrate fields now being worked in order to determine the quantity of nitrate of soda which can be recovered by the processes now in use. After an exhaustive study of the fields the commission has arrived at the conclusion that even after prudently reducing the estimate the quantity of nitrate yet to be extracted amounts to 240 millions of tons. Therefore, even after taking into account the increase in the consumption, the supply is sufficient for agriculture and the industries for more than a century.

It should be added that, contrary to what is generally known, the deposits now being worked represent but a small part of the total area of the nitrate region, which covers about 25,000 square kilometers, or 10,000 square miles. This immense zone, which, not including unexplored areas, is over 700 kilometers, or over 420 miles long, will supply all demands for many years.

It seems possible to improve the present methods of working and particularly of treatment and separation of the nitrate from the foreign matter which accompanies it. The government of Chile has established an international competition, with important prizes for any one improving on the more or less primitive processes still in use.

Other deposits of nitrate have been found in California, Colombia and in the Caucasias, but up to the present none is important nor can the deposits be worked as easily as those of Chile.

There are, however, still other sources of nitrogen. For example, certain animal refuse, such as hides, wool, etc., spoiled fish and food, all of which are used as fertilizer after treatment with sulphuric acid.

Coal contains from 1 to 2 per cent of nitrogen, and as 800 to 900 millions of tons of coal are consumed annually in the world a considerable quantity of nitrogen is thus released. However, as at most but 1-20th of that coal is distilled in retorts of producers or coke ovens, and further as only a part of the nitrogen contained is recovered as ammonia, the amount of nitrogen compounds thus actually produced is small.

The recent development of new forms of coke ovens has appreciably increased the world's production of ammonium sulphate, so that this total may be given as 800,000 tons, of which more than 50,000 tons are produced in France. This is therefore an important source of nitrogen for agriculture, which will exist as long as the coal deposits of the globe endure, but it will not make up for the shortage when the natural nitrate deposits are exhausted.

Is, then, humanity, and the white races in particular, destined, after some centuries, to be unable to feed itself on account of the impoverishment of the soil and the failure of the crops? This is a melancholy view of the situation which the illustrious savant, Sir Wm. Crookes, took in 1899 and which he opposed with the statement: "Before we suffer a real dearth the chemist will have intervened and he will have put off the day of famine to a date so distant that we and our children and our great-grandchildren may live without worrying about the future."

These prophetic words of Crookes have been realized. The scientist has turned to the enormous supply of nitrogen contained in our atmosphere, and, thanks to electricity, that benevolent fairy which has again come to the assistance of humanity, they are beginning to exploit this "deposit" and to extract

nitrogen in a form which is useful to the industries and in agriculture.

We will now see how the problem of the fixation of nitrogen has been solved.

The nitrogen compounds obtained are in two forms:

First.—Cyanamide, or "lime nitrogen."

Second.—Nitric acid and the nitrates.

(*A*) *Cyanamide.*

This name is applied to a product composed of carbon, nitrogen and calcium, according to the formula CN_2Ca.

In organic chemistry "amide" is the name for a dehydrated salt of ammonia. Cyanamide CN_2H_2 results in this way from the dehydration of ammonium cyanate, $CNO.NH_4$, according to the formula

$$CNO.NH_4 = CN_2H_2 + H_2O.$$

Calcium cyanamide is a calcium salt of cyanamide CN_2H_2, in that the two atoms of hydrogen are replaced by one of calcium, as is shown in the above formula, CN_2Ca.

Franck, a scientist of Charlottenburg, discovered cyanamide by a lucky chance and without seeking it. With Caro he was studying the preparation of metallic cyanides, such as barium cyanide, when he discovered that the carbides of the alkaline earths absorb nitrogen under the influence of heat—that is, if nitrogen is passed through calcium carbide which is kept at a high temperature, the gas is absorbed; the carbide looses an atom of carbon and is changed to calcium cyanamide,

$$CaC_2 + 2N = CaCN_2 + C$$

In the presence of water (as, for instance, in moist soil) cyanamide breaks up into ammonia and calcium carbonate as follows:

$$CaCN_2 + 3H_2O = CaO.CO_2 + 2NH_3$$

The industrial manufacture of cyanamide was accomplished first in Italy at the Piano d'Orte plant near Pescara, where there are available 25,000 hp in hydroelectric power, then in Switzerland, in Norway[1] and in France (at Notre-Dame de Briançon).

The Société des Produits Azotés has installed an important plant near the carbide factory.

The carbide of calcium used is the ordinary commercial product, 80 per cent pure, yielding 300 liters of acetylene gas per kilogram (4.8 cubic feet per pound). It is crushed, then ground in a closed ball mill in which circulates a current of pure, dry nitrogen to prevent any alteration due to the humidity of the air. The fine product is charged into retorts similar to those used in German gas plants and is heated with coal, or, as is more frequently done, is charged into iron crucibles heated electrically, as at Notre-Dame de Briançon, the charge per crucible being 400 to 500 kg (900 to 1100 lb).

When the temperature is sufficiently high (cherry red), say, 1200° to 1300° C. (about 2200° to 2400° Fahr.), the current of nitrogen is introduced.[2] The absorption of the gas continues regularly and during 30 to 60 hours. The completion of the reaction is indicated by a manometer in the nitrogen pipe line.

As the combination which takes place is exothermic, the quantity of coal or of current required for heating the furnaces is relatively small.

The product obtained is a blackish, porous mass which is immediately crushed and powdered for delivery in 100-kg (200-lb.) sacks.

The percentage of nitrogen varies greatly, from 15 to 20 and even more. It depends on the quality of the carbide employed and also upon the purity of the nitrogen. It is important that the nitrogen should contain no oxygen and should be very dry. For these reasons the Linde and Claude apparatus is usually employed,[3] as this gives a product 99 per cent pure nitrogen.

As this apparatus has been described before let us now only consider the principle involved.

The air passes through towers sprinkled with a solution of caustic soda which fixes the carbonic acid. It is then subjected to a pressure of 4 to 6 kg per square centimeter (57 to 85 lb. per square inch), cooled with water at ordinary temperature and dried by circulating over calcium chloride.

It is then filtered to remove dust together with entrained particles of the reagents used.

The air thus purified is cooled with ammonia to at least 20° to 25° Cent., at which temperature it is put in the counter-current circuit which reduces it 180° to 190° in the coils surrounded by the gas coming from the distilling apparatus, and liquefaction at the valve results.

As in all processes of rectification (fractional distillation), it is necessary to obtain a very low temperature above the distilling apparatus in order to hold back the oxygen and separate the nitrogen, which is the more volatile.

This result is obtained by the use of a refrigerating machine, which consists of nitrogen pumps drawing gas from the upper part of the still, a circulating system parallel to the air circuit and the release of liquid nitrogen on the upper part of the still.

This nitrogen as it descends from one level to the next holds back the less volatile oxygen and thus pure nitrogen is obtained in the upper part of the apparatus with 75 to 80 per cent oxygen at the bottom.

The net cost of cyanamide depends first of all upon the cost of carbide, since 80 lb. of the latter are required for every 100 lb. of cyanamide.

The use of cyanamide as a fertilizer in Europe has formerly disclosed several inconveniences. It could not be used as a soil covering. It had to be scattered at just the right time, either when the ground was being cultivated or when the seed was sown, as the very fine powder was blown about by the slightest breeze. It was also caustic and burned the hands of the farm laborers.

It seems now that a remedy for these disadvantages has been found in slightly moistening the cyanamide without allowing the temperature to rise, so as to prevent any heating which would transform part of the combined nitrogen into ammonia gas. This ammonia would not remain in the highly alkaline and hot mass and it would be released with what would represent a considerable loss.

After this moistening, which eliminates all traces of undecomposed carbide, the product is oiled with cheap hydrocarbons—petroleum in Germany, and coal oil in France—which gives a commercial fertilizer which is not dusty nor caustic.

A patent has recently been applied for mixing oxide of iron with cyanamide powder, which gives a granular product of good appearance and which it is easy to use.

(B) Nitric Acid and Nitrates from Air.[4]

We have now reached the second method, which is also the more interesting, that is, the production of nitric acid and nitrates from the nitrogen of the air.

Many processes have been designed during the last few years. We will not discuss all of these but will deal only with the three processes which are really practical and which are now in actual operation, and which consequently will interest engineers. These are the processes of Birkeland and Eyde, of the Badische Anilin und Soda Fabrik and of Pauling.

1. *The Birkeland and Eyde Process.*

In the apparatus[5] used by Birkeland and Eyde the principal and most important item is the furnace (Fig. 4 on page 74).

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[1] A large Norwegian calcium carbide and cyanamide works was described in some detail in our volume VII, pages 212, 309, 360 (May, July and August, 1900).—Editor

[2] The temperature at which cyanamide begins to form is not more than 800° to 1000° C. (about 1500° to 1800° Fahr.). Since the reaction is endothermic, it evolves so much heat that no large supply of electrical energy from the outside is necessary.—Editor.

[3] In some cyanamide plants, instead of the liquid air process, a simple chemical method of burning out the oxygen from the air by means of copper is employed. The copper is mixed with asbestos to give a large surface and the oxide is reduced either with methane or with producer gas. The process is exceedingly simple on paper, but has given a good deal of trouble in practice on account of the excessive heat produced.—Editor.

[4] This part of Lamy's paper has been considerably abstracted to avoid repetition of what has been discussed at length formerly in this journal. Compare also the article on page 73 in this issue.—Editor.

[5] See also page 73 of this issue; further our vol. II, page 399; vol. IV, pages 126, 295 and 360; vol. VII, pages 304 and 305; vol. VIII, page 451.—Editor.

The arc is given a very great surface by means of the deviation produced by an electromagnet. The arc is thus caused to take the form of a flat disk with a diameter of 6 ft. in commercial furnaces. The general form of the furnace resembles a thick lens. It consists of two thick cast-iron shells which surround and support the refractory material. These are cored for air and gas circulation, and between them is a space 3 to 4 in. wide and 6 ft. long, which is the oxidizing chamber in which the electric discharge is maintained.

The electrodes consist simply of copper tubes bent in U-shape and cooled by a circulation of water so that their temperatures are never high. Fig. 4 on page 74 shows a vertical section of this furnace. The oxidizing chamber is represented by the restricted space between *a* and *a*. The electrodes, only indicated in the figure by the little circle in the center, are perpendicular to the plane of the drawing. The poles of the electromagnet are shown at the right and left, carrying the magnet coils. The current of air under pressure from a blower circulates in the furnace in the direction shown by the arrows. It enters through the two ducts at the right and left, shown at the bottom of the diagram, passing through two distributing chambers and through a large number of small tubes to the oxidizing chamber in the center. It is conducted from this through the annular canal *a a*, which surrounds the oxidizing chamber, finally escaping at about 1000° through the central exit duct to the apparatus which we will take up later.

It seems that the Birkeland furnace is remarkably durable, as it can run a year without excessive maintenance and repair charges. The refractories need replacing every five or six months and the electrodes every three or four weeks.

A number of furnaces of this type have been built at Notodden which vary in the amount of power required from 250 to 4000 hp.

The necessary electric current is supplied by the Svaelgos plant in the form of three-phase, 50-cycle, 10,000-volt currents.

2. Badische Company Process.

The engineers of this company, Dr. Schönherr in particular, in 1905 invented a different furnace system which is now being tested in the Norwegian plants, where it seems to be giving good results.[6]

The furnace of the Badische Company seems simpler than that of the Norwegian company. Its construction is cheaper, while all the costly and delicate mechanism necessary for creating the magnetic field is avoided, but it is more difficult to operate and requires more attention. The quantity of NO produced is practically the same, but the percentage of NO in the gas mixture is higher, that is, the effluent air contains 2 per cent of NO corresponding to 0.050 gram HNO_3 per liter (0.05 oz. per cubic foot), while the product from the Norwegian furnace contains from 0.03 to 0.04 gram per liter (0.03 to 0.04 oz. per cubic foot).

Several Badische Company furnaces have been built, but at the present writing they cannot use more than 700 to 800 kw, as already with this power the arc has reached a length of 7 m. (22 ft.). The two largest Birkeland and Eyde furnaces have a capacity of 4000 hp (2940 kw) each.

The average productions of the furnaces, which, as we have said, are practically equal, vary from 570 to 600 kg of HNO_3 (or from 12,500 lb. to 13 500 lb.) per kw-year. It does not seem possible to easily improve upon the latter figure. For a 750-kw furnace this represents 440 to 500 tons of HNO_3 per year, which is equivalent to 800 tons of commercial acid of 36° Baumé.

The Norwegian furnaces, which have been in use for several years at Notodden, have proved themselves eminently practicable. The Badische furnaces, which have been under test for some months, have behaved well up to the present, and it is difficult to choose between the two types, so much so that the Norwegian company intends to use both in the new works which it proposes to build.

But let us return to the gas which flows from the furnace of either system. This gas, consisting of a mixture of NO with a large excess of air, is, as we have said, at a temperature of 700° to 800° C. and even of 1000° to 1100° C. with the new 4000-hp furnaces. This temperature is much too high to permit of the gas being immediately conducted to the absorption towers. It is, therefore, cooled to 40° C. by being conducted through large boilers (the steam from which is used in evaporating pans which we will describe later), then through air-cooled aluminium pipes, the air thus heated being conducted to the furnace. Aluminium pipes are used because this metal is not attacked. The cooled gas is conducted to a sheet-iron tower lined with refractory material where the oxidation is continued and completed at the expense of the oxygen of the air. Then it is conducted to the high absorption towers, which are of two kinds. One is of Drammensfjord granite, not affected by acid, and with joints of asbestos and silicate of soda; the others are simply of wood.

The three granite towers are 34 m. (100 ft.) in diameter and 20 m. (65 ft.) high. They are filled with pieces of quartz which are continuously sprinkled with acid solution. The gas as it traverses successively the three towers is brought into contact, following a definite course, with more and more dilute acid.

In this way the strongest acid, which is taken from the bottom of the first tower, contains 500 grams of HNO_3 per liter, while pure water enters the top of the third tower. If the manufacture of calcium nitrate is the only object in view, it is useless to push the concentration too far, and in practice the acid is drawn off containing only 300 to 400 grams of nitric acid per liter.

The handling of this liquid is accomplished automatically by the use of montejus made of pottery which is not affected by the acid, and operated with compressed air.

At the gas outlet of the last tower about one-quarter of the NO is still undissolved. To catch this it is necessary to use an absorbent which is more energetic than water, such as a solution of soda or lime. This absorption is carried on in two towers filled with pebbles or pieces of granite. The absorption is so nearly complete that the gas discharged to the atmosphere contains but 4 per cent of the combined nitrogen, that is, 96 per cent of the NO has been recovered in the towers in the form of nitrogen compounds.

The liquid flowing from the wooden towers is a mixture of solutions of nitrite and nitrate of calcium if milk of lime has been used. This mixture is treated with nitric acid produced in the plant itself. The nitrous acid which is freed is conducted to the absorption towers.

If instead of milk of lime a solution of carbonate of soda is used as a spray in the wood towers, as is the general practice, the resulting solution is concentrated and the nitrate of soda is separated by crystallization and is a product in demand for the manufacture of dyes.

The nitric acid coming from the granite absorption tower is too weak to be put on the market. It can be concentrated to a commercial standard strength of 36° to 40° Baumé, and at present deliveries of this strength are being made in Norway. Eyde claims to have secured a 96 per cent concentration of nitric acid, but I have not yet been able to find that the Norwegian works have produced and sold acid so concentrated. They generally prefer to use the tower acid for the manufacture of calcium nitrate.

To do this, the acid is conducted to granite vats filled with coarsely broken limestone, which is rapidly attacked. When the solution is neutral—determined by adding a small amount of milk of lime—it is concentrated by evaporation in a Kestner apparatus, using the steam from the above-mentioned boilers, which are heated by the gas from the furnace. The material,

[6] This furnace is illustrated in Fig. 6 on page 74. For further details see also our vol. VII, page 245.—EDITOR.

kept in fused state to a temperature of 145° C. may be poured into thin-sheet iron cylinders where it solidifies upon cooling. It may be then sold in this way to chemical manufacturers, but it is generally solidified in cast-iron molds, crushed and pulverized and packed in 220-lb. barrels for delivery to the farmer.

The adjoining diagram is a flow-sheet of the above-described operations. 1 is the blower, 2 the electric furnace, 3 the boiler, 4 the cooler, 5 the oxidation tower, 6 the three absorption towers of granite, 7 and 8 the two absorption towers of wood, 9 the final nitric acid, 10 the nitrite solution. *A* is the apparatus for the production of saltpeter, *B* the apparatus for the production of nitrite, 11 solution tank, 12 a neutralization tank, 13 and 18 are Kestner evaporators, 14 solidifying tank, 15 crushing, 16 montejus, 17 receiver, 19 crystallization, 20 filter, 21 mother lye, 22 centrifuge, 23 montejus and 24 receiver.

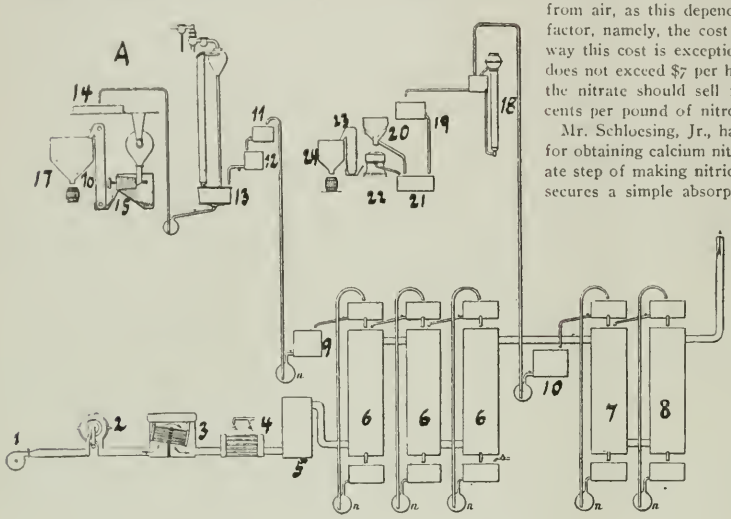


FIG. 1.—FLOW SHEET OF BIRKELAND-EYDE PLANT.

The calcium nitrate thus obtained contains an average of 13 per cent of N and is an excellent fertilizer. The tests made both in France and abroad are conclusive in this respect.

Mr. Malpeaux, director of the Agricultural College of Berthonval, has begun a comparative study of sodium nitrate and the Norwegian calcium nitrate. During several years he has experimented with various plants, such as wheat, barley, sugar beet, potato, poppy and onion.

"We may conclude," Mr. Malpeaux remarked, "from our experiments that the calcium nitrate such as is furnished by the factory at Notodden at least equals sodium nitrate. While the former furnishes the plant with the calcium nitrate already formed, providing nitrogen and calcium at the same time, the sodium nitrate is changed in the soil to calcium nitrate and the remaining sodium salt is of no use to the plant. It is then in the form of calcium nitrate that the crop utilizes the nitrogen, and it is, therefore, not astonishing that under certain conditions the action of the Norwegian nitrate should be superior to that of sodium nitrate."

The opinions of scientists and farmers in other countries concerning the qualities of calcium nitrate seem to be the same as those of their French colleagues.

Using equal weights of calcium and sodium nitrates—from 100 to 300 lb. per acre, according to the crop—identical results were obtained, even though sodium nitrate contains 2 to 3 per cent more nitrogen, or 15 to 16 instead of 13 per cent nitrogen. In certain countries, especially in Germany, the prices of the two are the same. This is not true in France, where the nitrogen content is taken into consideration. If, for example, nitrate of soda is worth $1.90 per 100 lb. at Dunkerque, calcium nitrate sells for $1.70 per 100 lb. at Rouen.

The one slight disadvantage of the calcium nitrate is that it is hydroscopic, which makes it difficult to spread in the wet season, especially by the use of machinery. It is always best not to open the barrels until the moment when the material is to be used. Means are now being sought to diminish the hydroscopic nature of the product.

It is very difficult to state the net cost of producing nitrate from air, as this depends principally upon a very variable factor, namely, the cost of an electrical hp-year. In Norway this cost is exceptionally low, it being claimed that it does not exceed $7 per hp-year. Under such circumstances the nitrate should sell for $1.15 per 100 lb., this being 9 cents per pound of nitrogen.

Mr. Schloesing, Jr., has invented and patented a process for obtaining calcium nitrate directly without the intermediate step of making nitric acid as we have described it. He secures a simple absorption of the nitrogen oxide gas by lime. After having first dried the air entering the furnace, he conducts the gas leaving the furnace (cooled to 300° or 400°) to large sheet-iron chambers containing lime prepared in the following manner: Ordinary limestone is crushed and molded into briquettes and heated to the temperature of dehydration. These briquettes have the property of very easily absorbing the nitrogen oxide vapors at 300° to 400° C. He obtains a complete absorption of the NO and with an apparatus much smaller than is required by the other process. Moreover, the nitrate obtained contains 14.5 to 15 per cent of nitrogen. The process requires comparatively very simple apparatus and therefore entails a considerable economy in a given industrial application.

A small-scale test of the process has been made at Notodden which confirms all that Schloesing has claimed. There is therefore, reason to believe that a large-scale application will soon be attempted.

3. *Pauling Process.*

We will now describe our third practical process for making nitrates from air, that of Pauling.[7]

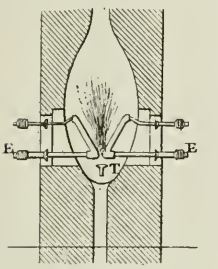
After numerous and long laboratory tests Pauling make his first installation in 1904 in the plant of the Saltpetersaure company, at Patsche, near Innsbruck, in Austria. Since then he has made a series of improvements on his process so that he now has made it a truly practical industrial process. In 1908 a French company, La Nitrogene, acquired the French rights to the Pauling process and is now exploiting it in a recently built plant at Roche-de-Rame, near Briançon.

We have visited this plant and we will now proceed to describe it from our own observations and from data kindly supplied us by the manager, Mr. Mollard, and the engineer, Mr. Gros.

Let us first go into the fine, large furnace room. The furnaces appear to be nearly square (3 ft. 3 in. by 3 ft. 8 in.), they are 10 ft. high and built in groups of three. Built as they are of simple masonry of refractory brick and stayed with iron, they show to a first glance two important qualities, simplicity and strength. The essential part is a vertical slot of refractory brick in which are placed the hollow cast-steel electrodes ar-

[7] See Fig. 3, page 73, of this issue; also our Vol. VII, page 430, where the Innsbruck plant is described.

ranged as shown in Fig. 2. They are reversible and interchangeable. Just below the space between these two electrodes and in the same vertical plane with them is the opening of a flattened tuyere *T* through which the hot compressed air reaches the furnace, the force of which serves to blow the arc. There is here a difficulty to overcome. It is necessary that the space between the two electrodes should be sufficient to pass the necessary quantity of air, but at the same time it should be constant and not extinguish the arc as it lengthens. Pauling has ingeniously supplied the main electrodes with auxiliary, movable copper knife-blade electrodes, the extremities of which can be brought near enough together to lure and start the arc as soon as it goes out, thus requiring a relatively low voltage.



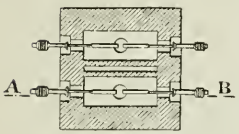


FIG. 2. PAULING FURNACE.

The positions of the main electrodes are adjustable by means of screws, and of the knife-blades by a rack and pinion. These maneuvers are very easily made from without the furnace, the electrodes and their adjustments being protected by movable screens.

Under the influence of the jet of air from the tuyere the arc, rising between the copper blades, spreads to the tops of the horns of the electrodes and produces a flame 3 ft. to 4.5 ft. high. The combination of O and N in form of NO and nitrous vapors is made at the contact of the flame and the incoming air. In order to obtain a rapid cooling of the mixed gases, a suitable current of air is admitted at the bottom of the furnace. The furnace is provided with permanent peep-holes for examining the knife-blades, electrodes and tuyeres, to assist in maintaining the furnace in certain and constant operation.

The copper blades last for 20 hours and can be replaced in two minutes. The electrodes are replaced with equal facility. They are good for 500 to 600 hours, thanks to the cooling effect of the circulation of cold water and also to the interesting peculiarity that the hottest part of the flame is the upper part, that is, the point at greatest distance from the electrodes.

An opening in the gas discharge channel permits the introduction of a pyrometer to measure the temperatures.

Each furnace includes two arcs in series using a three-phase 50-cycle 4000-volt current. The electrical equipment consists of two small coils, a small transformer and a starting box. All this is placed on the wall of the room in front of the furnace. The operation of the apparatus is simple and without danger. I have tried it myself. As soon as the switch is closed the furnace begins normal operation and one man can care for two batteries of three furnaces. Nine furnaces are now in operation. Each absorbs 500 hp, but it is expected to raise this figure to 600 and then to 700. The production will be proportionately improved, as was demonstrated in the Pratsche plant with furnaces consuming 1000 hp and even 1500 hp.

But, let us suppose that the furnaces are in operation. We have said that the air is brought to them through large conduits underneath, and that the air blows through the tuyeres, coming in contact with the long electric flame where the combination already explained takes place. The mixture of gases containing 1.15 per cent to 1.20 per cent of NO goes out at a temperature of 700° to 750° and immediately enters a cooling chamber, where it gives up part of its heat to the air which is on its way to the furnace. The gas is then conducted through two masonry flues alternately. While one is cooling the gas the other is connected to the atmosphere through a chimney, which cools it ready for use as soon as the other becomes too hot, which is usually after a period of six hours.

After being drawn through an exhauster which creates a suction as far as the furnace, the gas at a temperature of 35° C. enters a large oxidizing tower, made of reinforced concrete 35 ft. in diameter by 70 ft. high. It is in this tower, on account of its large volume, that a complete mixture is obtained and a prolonged contact provided and a complete oxidation of the lower nitrous vapors secured.

An earthenware conduit then carries the acid gases to the absorption towers, five in number. These towers are built of refractory brick and contain a material unaffected by the acid (similar to what is employed in the glover towers of sulphuric acid plants) over which flows pure water or water already acidulated in order to obtain a regular absorption.

A series of automatic earthenware montejus are used in handling these liquids.

The strong acid comes from the first tower. The acid reaches 5° Baumé in the fifth tower, 8° in the fourth, 12° in the third, 15° in the second and finally 25° to 30° in the first, which means 35 per cent to 40 per cent of nitric acid.

This strong acid is put in porcelain capsules, which are placed in a kind of Benker furnace, where it is concentrated by the hot gases which come directly from the furnace. The entrained water vapor is recovered from the cold gas in one of the absorption towers. The concentrated acid goes through a cooling coil of earthenware to large reservoirs, from which it is drawn for shipment.

This acid contains from 60 per cent to 62 per cent nitric acid and, therefore, measures from 38° to 40° Baumé, which is a good degree of commercial concentration.

The Innsbruck works, with special apparatus, is able, it seems, to push the concentration to 97 per cent nitric acid, which corresponds to more than 48° Baumé. It is also proposed to produce an acid of similar concentration at Roche-de-Rame.

Upon leaving the absorption towers the remaining gases are conducted through wooden towers containing soda solution and are then allowed to escape. A certain amount of nitrite of sodium is thus obtained, which, after concentration, crystallization and treatment in a centrifugal drier, is marketed in a crystal form, 97 per cent to 98 per cent pure.

Ninety-six per cent of the NO is recovered during normal operation. The yield amounts to from 65 grams to 70 grams (or 2.3 oz. to 2.5 oz.) of nitric acid per kw-hour consumed, or 1100 lb. to 1300 lb. per kw-year, or, again, about 2200 lb. of 36° commercial nitric acid, as in the Norwegian furnaces.

As regards net cost of production we must repeat that it depends largely upon the cost of power. The rather vague data on this subject which we possess does not permit our giving accurate information.

Taken as a whole, the Pauling furnace appears simple, compact, to require inexpensive repairs and the whole process should be practical; in fact, it appears that the Pauling plant particularly has reached a point where it runs regularly and with commercial success.

The Roche-de-Rame plant gives the visitor a curious impression, as do other similar plants. There are few men, no noise outside of the building of furnaces and, above all, no visible raw material, these being restricted to water and air, the entrances of which are not noticed.

But there is one very important element which enters quietly on copper and aluminium wires. That it plays an important part in the net cost can easily be seen in the above-quoted figures of production per kw-hour. It requires a kw-year to produce a single ton of calcium nitrate of 13 per cent nitrogen. Power must, therefore, be obtained in large quantities at a low price. Thus recourse must be had to "white coal" (the French name for waterfalls as producers of electrical energy).

At Roche-de-Rame water-power is obtained from the

Biaisse River, an affluent of the Durance. Four sets of 2000 hp have been erected and considerably more power is being sought.

But, as regards natural power, Norway is a privileged country. It is on this account that it has been considered feasible to create the new industry there, and that the powerful Norwegian Nitrogen Company, with a capital of $8,000,000, has built its plants in that country.

But Norway is not the only European country in which important available waterfalls can be found. In France, particularly, there are still good hydraulic powers available in the Alps and in the Pyrénées, and, even if the cost per horse-power of installation be appreciably higher than in Norway, it should be possible to reach a reasonably low net cost of production if the right to use processes can be obtained at a reasonable figure. It is, therefore, probable that other companies will follow the example of the Nitrogen Company and construct more plants in France to use one of the described fixation processes.

The fixation of nitrogen, due to the fruitful collaboration of scientist and engineer, constitutes one of the great discoveries of modern times—one of the most useful, undoubtedly, to agriculture.

That industry has no reason to fear the lack, even at a remote period, of the nitrogen fertilizers which it must have to support humanity. If even neither the niter deposits nor any of the other sources of nitrogen become exhausted, the farmer will find in calcium nitrate an inexhaustible supply of fertilizing material, because it will be possible to produce it so long as our earth is surrounded by its atmosphere and supply limestone—so long as water evaporates from the sea and, precipitating as snow and rain, forms torrents which descend from the mountains.

Thus will be realized a wish expressed by Renmaux, who said: "We should make the conquest of Nature by means of Nature herself."

Paris, France, and Lima, Peru.

Synopsis of Metallurgical and Chemical Literature

Iron and Steel.

Crystalline Structure of Iron at High Temperatures.—Walter Rosenhain describes in the *Physikalische Zeitschrift* of December 1, 1910, tests with iron and iron alloys to determine the nature of the change going on in iron and steel at high temperatures. As it was necessary to keep the original polished surface of test piece free from all oxides, even at temperatures above 1000° C., the iron was heated by an electric arc in a very high vacuum and tests were made in a specially constructed apparatus with a spring to test the effect of tension. Soft steels, electrolytic irons of great purity and almost chemically pure iron were tested. The conclusions may be summed up as follows: Iron up to 1100° C. acts like a crystalline aggregate and a plastic change of form takes place along the plane of cleavage of crystals of which it is composed. Between ordinary temperatures and 1000° C. iron exists in three modifications which have different mechanical properties. The temperatures within which these modifications take place can be identified with the three forms, alpha, beta, gamma, of iron according to the Osmond and Roberts-Austen theory of allotropic forms. The beta-iron forms at higher temperatures but is stronger and harder than the alpha-iron. It changes from the alpha-state into beta-state with an accompanying change in volume. Almost pure iron at high temperatures has the characteristic structure and a few characteristics of gamma-iron, as is also the case in several steel alloys. The existence of a hard beta-iron modification is also proven at high temperatures; failure of iron quenching is due to difficulty in preventing change from beta-state to alpha-state by this method. Evidence points to the fact that beta-iron if it could be kept unchanged at ordinary temperature would be very hard.

Direct Reduction of Iron Ores in the Electric Furnace.—Mr. G. Arnou in *La Revue de Metallurgie* December, 1910, discusses the new electric furnace of the La Neo-Metallurgic Company for the direct reduction of iron ores. The Chaplet-Neo-Metallurgic furnace is used, the design of which is shown in Fig. 1. It is an arc furnace; the current enters through one or more vertical electrodes and leaves through one or more side channels *B*. The latter are connected with the main bath *A* by horizontal channels *C* below the level of the bath. The slag is charged first, then the ore to be reduced mixed with carbon in proper proportions. Then the reduction begins under the action of the arc and the reduced metal filters through the slag and collects on the bottom. The furnace works similar to a small blast furnace where there is also a separate reducing and fusion zone. More than 15 tons of metal have so far been produced. The slag contains in normal operation less than 8 per cent iron oxide. The charge was a mixture of hematite ore and dried, powdered charcoal. Dried hematite briquets and wood charcoal and later mixtures of carbon and hematite in form of powdered dust were also tried.

With ore and carbon the furnace produced 77 lb. iron per hour.

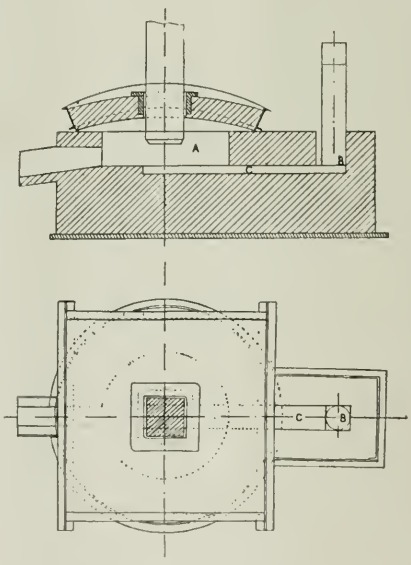


FIG. 1.—ELECTRIC FURNACE FOR STEEL FROM ORE

With briquets the furnace produced 70 lb. iron per hour.

With powder (dust) the furnace produced 68 lb. iron per hour.

The consumption of carbon was different in the three cases, being 660 lb. in the first and 726 lb. and 792 lb. per ton in the second and third cases. These differences are due to the hygroscopic nature of wood charcoal, the evaporation of the water consuming considerable heat. The differences are less if coke or anthracite is used instead of charcoal. In all cases it is possible to treat pulverized ores in this electric furnace, and even a mixture of hematite and retort coke in the powdered state without difficulty. Comparative tests were made, smelting hematite, magnetite, and siderite with coke, charcoal, and

anthracite. It does not make much difference which of the three forms of carbon is used. With charcoal the voltage tends to remain higher than with coke or anthracite, but this is not a serious defect. As regards thermal efficiency anthracite gives the best result. Reductions of hematite with charcoal produced 68 lb. per hour, while anthracite gave 84 lb. an hour with the same electric energy consumption. In the first case 792 lb. carbon per ton of iron, in the second case 596 lb. were used. Magnetite gives better returns than hematite with charcoal. Reduction of magnetite briquets produced 90 lb. per hour, whereas hematite gave only 70 lb. With anthracite both ores gave about the same result. The siderite was easily reduced, but with a large consumption of energy, yielding only 64 lb. per hour for the same energy consumption. Products obtained when analyzed gave the following results:

Carbon	Manganese	Silicon	Sulphur	Phosphorus
0.08%	0.10%	0.02%	0.02%	traces
0.09%	0.09%	0.04%	0.02%	"
0.11%	0.46%	0.06%	0.02%	"

In the last case a large addition of ferromanganese had been made for comparison. These results were obtained by direct reduction from the ores. The iron is perfectly malleable and can be hammered cold into thin sheets. After tempering it bends at 180° even with 0.20-0.25 per cent carbon, which fact is due to its small percentage of manganese. It has great tenacity. As to the cost, it is stated that most of these tests were made with a 120-kw furnace. In the reduction of hematite with charcoal for the production of soft iron the consumption was 3,430 kw-hours with a 120-kw furnace and 2,600 kw-hours with a 200-kw furnace. Studies in regard to the influence of the forms of ores and carbon compounds used as charge on the amount of energy used for the production of 1 ton of soft steel gave the following results:

Magnetite and charcoal	3,150 kw-hours
Magnetite and anthracite	3,050 kw-hours
Hematite and charcoal	3,430 kw-hours
Hematite and anthracite	3,100 kw-hours
Siderite and anthracite	4,000 kw-hours

These tests were made in a 120-kw furnace. For a 200-kw furnace the efficiency is as high as 80 per cent.

As to the cost of maintenance it is pointed out that the arc in this furnace is entirely enclosed by the charge, which absorbs all the heat of radiation, so that the heat losses are small. The wear and tear are small. During the entire time of the tests the furnace used was not repaired and was as good at the end of operations as at the beginning. The amount of carbon in the charge varies with the different mixtures. Reduction of powdered hematite required 792 lb. charcoal and 594 lb. anthracite per ton of iron. Magnetite under same conditions required only 682 lb. charcoal and 572 lb. anthracite. As a rule 55-77 lb. of electrodes are consumed for the production of a ton of soft steel.

Gold and Silver.

Milling at Cobalt.—An interesting feature of ore dressing at Cobalt, Canada, is that the methods in use had to be devised especially for the ores of the district. No precedent could be referred to by the metallurgist and it is greatly to his credit that he has succeeded so well. In the *Mining Journal* (London) for Dec. 17 and 24, 1910, Dr. JOSEPH T. MANDY has given the results of his observations on the work at Cobalt. At present thirteen mills are in operation by electric power, and three more are in course of erection. Wet methods of dressing are adopted, and in some cases hydrometallurgy is practiced to the extent of the final recovery of the metals at the mill.

No two mills are exactly alike, but are the same in general principle. For illustration, the equipment and practice of the Coniagas mill may be taken as typical. The ore is crushed in Blake and gyratory crushers to 1½ in., then rolled to ¾ in. and screened in trommels. The coarse is concentrated in Harz jigs and the fine on Wilfley tables. The residue from these is crushed in 30 stamps, following which the pulp is thickened and treated on Deister and Wilfley tables, followed by a canvas slime plant.

Considerable difficulty was experienced in the early work in saving values from the tailings, owing to the escape of small scales of metallic silver and smaltite which seemed to have a great tendency to float away. By paying more attention to the slime recovery this loss has been largely averted. The average extraction in the district is probably 82 per cent. During 1909 the ratio of concentration was 39 to 1. The cost of milling was higher in 1909 than in 1910 owing to the use of cheaper electric power in the latter year. The cost of the work at the McKinley-Darragh mill was $1.98 in 1909, and is reported to be about $1.30 in 1910. Power formerly cost in the neighborhood of $150 per hp-year, and is now obtainable at about $50.

At three plants—the O'Brien, Buffalo and Nova Scotia—cyanide plants are operated in connection with the concentration. Dr. Mandy considers the success of this process as yet problematical. The Nova Scotia plant uses pan amalgamation and cyanidation, but no figures are available on the success of this system.

The concentrating plants all treat what is known as the low grade ore, containing 10 to 40 oz. silver.

The high grade ore, varying from 100 to 3000 oz. silver, is treated in smelters. The smelting ore is further divided into low and high grade; the first carrying from 100 to 600 oz. silver and the last from 800 to 3000 oz. The purchasers pay a sliding scale according to the silver content, and have various schedules for the arsenic content. United States smelters treated 2/3 of the tonnage in 1910, and Canadian plants treated about 32 per cent, the balance going to England and Germany.

The ores are silicious and make a good fluxing ore in silver lead smelting. They also may be used for converter lining, but for this purpose there are several objections, viz., the arsenic fails to volatilize completely, and practically not at all after the first 10 min. of blowing, causing about 1.5 per cent of arsenic to appear in the copper; the slag is inclined to be thick; the charge effervesces owing to carbonates in the ore, causing high copper slags, sometimes as high as 15 per cent; alumina frequently has to be added to the ore for binding; and finally, the lime and iron in the ore tend to rob the lining of active silica, leaving less to do the work of slag production.

Milling Native Silver Ore in Mexico.—A combination of amalgamation and cyanidation for silver ore is described by WALTER M. BRODIE in the *Mexican Mining Journal* for January, 1911. The ores of the Batopilas silver mines are peculiar in that the clean base minerals, galena, blende and pyrite, seldom carry more than 4 oz. silver per ton. The values occur mostly as native silver, argentite and sometimes arsenical silver.

The ore is sorted into two classes, the high grade native silver ore which is treated without concentration, and the low grade milling ore which is concentrated.

The high grade ore is stamped through a fine screen. Some coarse native silver accumulates in the mortar and is removed and washed by hand prior to smelting and refining. The screened material is amalgamated in pans by the following process: The charge for a 4-ft. pan is 1000 lb. ore, 15 to 17 lb. mercury, 4.5 lb. cyanide of sodium, about 4 oz. lead acetate, 3.5 lb. lime, and 1 to 1 of stock solution. The solution in the pan then runs from 40 to 48 lb. cyanide per ton, and never below 33 lb. The ore is first ground for six hours without mercury, when it is added and grinding continued for 10 hours. The charge is then run into a settler where it is leached and agitated, mechanically and by air, for 12 hours. The settler is then washed four hours, and the amalgam and residues removed and washed by hand. The former is prop-

erly retorted, and the latter is sent to percolating tanks, referred to later.

The slime washings from the settler are thickened in a Dorr machine, the clear overflow of which goes to strong solution sumps, and the thickened underflow to Pachuca tanks. One of the latter is filling while the other is being agitated and drawn off. Blaisdell filters are used, four charges being put through in 8 hours. The tailings from the filter run about 9 oz. silver per ton.

The low grade ore is concentrated 50 into 1. The concentrate consists of pyrite, galena and blende, with native silver and argentite. It is reconcentrated into two classes, high and low grade. The high grade is amalgamated in a pan, adding mercury pound for pound of silver contained, and a slight excess. Grinding proceeds for 14 hours before amalgamation is started, this being continued for 2 to 5 days. Cyanide, lime and lead acetate also form a part of the charge, and the pan is decanted twice daily into the rich solution sump, the solution being made up again with stock solution holding 10 lb. KCN per ton. After 6 days the pan is discharged into a settler, where it is agitated mechanically and by air for six days, the solution being decanted once a day. The tailings from the settler go to the Dorr thickener, and thence to the Pachuca tanks.

The low grade concentrate, which forms the bulk of the product treated, is leached in the percolating tanks referred to above. The charge consists of from 35 to 60 tons of ore; the chemicals added being lead acetate, lime and cyanide. The latter is added in sufficient quantity to make 50 lb. KCN per ton. The solution is left on 24 hours, then drawn off and the tank allowed to aerate for 48 hours. Stock solution of 10 lb. KCN per ton is then run on and allowed to stand for 12 hours, after which it is withdrawn and the tank allowed to stand dry for 24 hours. This process is continued with 12 hour periods of percolation and aeration for two months, when 95 to 96 per cent of the value has been leached from the concentrate. The precipitated silver assays about 800 fine, and the refined product about 990 fine.

Classification of Tailing Pulp Prior to Cyaniding.—For the clean separation of sand and slime a number of schemes have been devised with greater or less degree of success in operation. In the October, 1910, issue of the *Journal* of the Chem. Met. & Min. Soc. of S. Africa, E. H. Johnson gives some results obtained with a plant of the type invented by W. A. Caldecott some time previous, and described by him in the same publication, August, 1909. The idea consists in classifying the pulp in primary diaphragm cone classifiers, washing the underflow in secondary diaphragm cones, finally delivering the separated sand to a horizontal revolving filter table where excess moisture is removed by vacuum, and the dried sand plowed off and transferred to cyanide tanks.

Mr. Johnson gives results obtained at the East Rand plant, where four 20-ft. sand filter tables are in use, with their complement of four 9-ft. primary cones and two 5 ft. washing cones. This plant has for some months handled the sand from approximately 4000 tons of tailing per day. A little over 50 per cent of the pulp is separated as a slime with less than 2 per cent remaining on a 200-mesh screen. The sand contains about 1 per cent slime as determined by elutriation, while 88 per cent is 200 mesh. Not only is the size separation efficient, but the bulk of the pyrite is retained in the sand.

The cubic space occupied by the sand and slime produced by this process is greater for the sand and less for the slime than under former conditions. This indicates that the interstitial spaces of the sand are more nearly true voids than under the old system which left more or less slime in the sand. Tests on 20,000 tons of sand indicate 23.5 cu. ft. to the ton. The extraction is accordingly more rapid, as shown graphically in the original paper. The essential difference between this system of diaphragm cone classification and all previous efforts in the same direction, is the maintenance of a large underflow of minimum moisture, the slime elimination being proportionate to the reduction of moisture in the underflow.

Gold from Clay Deposits.—*Metallurgie* of Dec. 8, 1910, brings a note on a new method of winning gold from clay deposits, due to J. J. W. H. van der Toorn (German patent 225,810). The processes now employed treat clay with water whereby a good deal of the gold is lost, as the separation of gold from clay is very difficult on account of the great cohesion of the clay. The new process tries to remove this cohesion. The gold-containing clay is spread over a large area, dried and kneaded. Then it is heated in a furnace similar to a brick kiln, till it assumes the hardness of stone. The rocklike masses are then broken in a stamp mill and the gold separated in the usual manner.

Copper.

Early Pyritic Smelting at Leadville.—An interesting article on the operation of the old Bi-Metallic Smelter at Leadville, is contributed to the December, 1910, *Bulletin* of the Amer. Inst. of Min. Eng., by Charles H. Doolittle and Royal P. Jarvis, respectively former superintendent and chief chemist at the plant.

Pyritic smelting of raw silver and copper bearing ores was commenced at Leadville by the Bi-Metallic company about 1892 in a remodeled lead smelter, known as the La Plata. After the success of the process was established a new plant was erected in Sept., 1893, and operated until the spring of 1900. Its success was based on the excellent facilities for obtaining suitable ores; heavy sulphides for the production of a preliminary low grade matte, and rich silicious ores for the second smelting and production of a high grade matte.

The plant consisted of three blast furnaces of different dimensions, but averaging about 36 by 163 in. at the tuyeres, and 6 ft. 6 in. from tuyere-center to feed floor. The jackets were supported on a bed plate, and the tuyeres were placed about 13 in. above the level of bottom brick lining, giving a crucible of the depth mentioned. The furnaces were provided with sumps lined with fire brick, attached to the middle of their long side. Matte and slag discharged continuously through the sump into settlers, from which slag flowed continuously and matte was tapped periodically. The superstructure of the furnace was built unusually high, and each had a down-comer to a flue in which most of the dust was settled.

A system of wet recovery was in use also. The hot dry gases were drawn from the flue and treated in a scrubbing tower, where an additional recovery was made. The flue dust proper comprised 3.5 to 7 per cent of the ore treated and consisted of Ag, 14 oz. per ton; Au, 0.12 oz. per ton; Cu, 2.3; Pb, 3.5, and Zn, 7.1 per cent. The slime from the scrubbing towers comprised 2 to 4 per cent of the ore smelted and consisted principally of lead, gold and silver, and the object of recovery was its sale to lead smelters. It carried Ag, 16 oz. per ton; Au, 0.11 oz. per ton; Cu, 2.4 and lead (dry), 6.3 per cent.

Sampling was entirely by hand, a tenth-shovel portion usually being reserved when unloading shipments. The portion cut out for the sample was reduced in quantity and size, and finally the usual buyer's, seller's and umpire samples prepared. The balance of the sample was reserved until settlement had been made.

Smelting practice comprised two steps, preliminary smelting of the raw ore into a low grade matte, and further smelting of this product with silicious ores to form a matte carrying about 300 oz. silver per ton. Formerly mattes had been made carrying several thousand oz. per ton, but better saving was made with a lower concentration ratio. The final product contained 14 to 23 per cent copper.

The raw ore-charge contained about 14 to 20 oz. silver per ton, 2 to 3 per cent copper, and an excess of iron over silica or insoluble matter of from 8 to 15 per cent. Thus a charge consisted of 2600 lb. ore, 225 lb. lime rock, 1400 lb. slag from the concentrating furnace, 200 to 300 lb. flue dust, and 325 lb.

coke. About 65 per cent of the sulphur was eliminated on the first smelting. The concentration, based on net ore smelted and matte produced, was only 2.44 per cent. The slag from this operation was clean to be thrown over the dump.

The matte produced from the green ore-charge was re-smelted with silicious ores. The ore from the Commodore mine at Creede gave the best results, although some Leadville silicious ores were used even if leady. Cripple Creek ores were unsuited and gave sticky slags. The fuel consumption, based on net ore smelted, was about 14 per cent.

The capacity of the furnaces smelting green ore varied from 125 to 175 tons of net ore per 24 hours. On a concentrating charge from 225 to 300 tons of charge were smelted in the same period.

Electrolysis of Iron-Containing Copper Solutions.—*Metallurgie* of December 8, 1910 contains a short note on the use of revolving cathodes in the electrolysis of impure copper solutions, according to a process of F. Lacroix (German patent 225,652). Impure solutions containing only 1 per cent copper can be treated. The electrolyte containing ferrous salts is stirred up by rapid rotation of the cathode and is driven at considerable speed through a perforated anode by centrifugal force. The ferrous salts are changed at the anode to ferric salts and passed outside. This prevents the ferric salts from reaching the cathode where they would otherwise react with the copper deposited there. The electrolyte enters the electrolytic tank at a point btween the revolving cylinder cathode and the surrounding perforated anode. The cathode makes about 40 revolutions per minute. The anode solution is rich in ferric salts and these are passed into coke towers or similar apparatus where they are treated with the sulphurous vapors from the roasting process. The ferric salts are again reduced to ferrous salts with formation of sulphuric acid. The solution of ferrous salts is then taken again to the electrolyzer.

Tin.

Concentration and Reduction Works.—The production of ingot tin, 99.8 per cent pure, is reported from Mount Franklin, near El Paso, Tex., by Chas. A. Dinsmore, who contributes a description of the plant to the Dec. 31, 1910, issue of the *Mining Journal* (London).

Concentration is done by a combination of wet and dry methods. The ore is crushed and screened by the usual methods. The first concentration is by Harz jigs, the tailing from which is dried, reground, sized and treated on dry tables. Sizing for dry concentration is very important, and in this plant 10 sizes are made, from 20 mesh to 200 mesh, by means of the Keedy dry sizer. The work of the tables is reported to be excellent.

The tin concentrate is reduced in a reverberatory furnace, oil being used as fuel. The charge consists of one part pulverized anthracite coal and eight parts concentrate. Reduction requires from six to eight hours, and about one ton of concentrate can be smelted per day. Temperature attained, 2700° Fahr. Loss of tin from concentrate to ingot, about 0.5 per cent. The concentrate represents about 60 per cent of the original tin in the ore.

A Cornwall Plant.—In the *Journal* of the Chem. Met. & Min. Soc., of South Africa for October, 1910, C. F. Thomas has published the details of tin concentration at a plant in South Crofty, Cornwall. The crude ore is hand sorted and finally crushed through jaw crushers and stamps. The latter weigh 1065 lb., drop 7¾ in., 102 times per minute. Discharge 1¾ in. to 2¼ in., through 20-mesh gunmetal screen. This screen costs twice as much as steel screen, but lasts 5 days as against 36 to 48 hours for steel. Stamp duty, 2.8 tons per 24 hours.

The pulp is classified by Luhrig classifier into four products; the first three classes are treated on tables and vanners, the fourth, or slime, being settled and then treated on vanners. The concentrates are calcined in Brunton furnaces, 15 ft. in diameter. Capacity, 4 tons to 5 tons of concentrate to a sweet roast in 24 hours. Fuel consumption, 1000 lb. to 1200 lb. coal. The arsenic recovered from the calcination is sold crude.

Wetherill magnetic separators are used to clean the concentrate. The first separator has six magnets, the first two being used to remove iron and the other four to remove wolfram. The iron product contains about 3 per cent to 8 per cent cassiterite, very little of which is free. The wolfram product averages 40 per cent to 55 per cent tungstic acid, and is treated with concentrated sulphuric acid for from five to seven days, after which the residue is dried and again magnetically treated with the recovery of from 10 per cent to 25 per cent cassiterite. The non-magnetic cassiterite is easily cleaned for market. The current used on the separators is 110 volts, with 1.5 amp to 3 amp for the weak magnets and 10 amp to 14 amp for the strong.

Zinc.

Separation of Zinc from Vapors Containing Zinc Oxide.—According to *Metallurgie* of December 8 Paul Schmidt and Desgraz (German patent 225,956) proceed as follows: The vapor containing zinc oxide is passed through scrubbers of the usual type where the gases come in contact with the purifying liquid. The combination of water and soap solution did not give satisfactory results since the sulphurous acid generally contained in the vapors is absorbed by the water, soluble zinc salts being formed, and the zinc oxide dissolved. This new process therefore uses oil (crude petroleum, tar oil, etc.), as any zinc sulphate in the gases is insoluble in these solutions. The zinc oxide is separated and afterward freed from the oil by means of presses.

Aluminium.

Alloys and Impurities.—The *Giesserei Zeitung* of January 1, 1911, brings an article on aluminium alloys and impurities. The natural impurities of aluminium are silicon, iron, titanium, and the alkaline metals, sodium, potassium. Calcium and titanium are found very rarely and are easily extracted. Titanium has a good influence on aluminium favorably if it does not exceed 3 per cent. An aluminium alloy with 2 per cent titanium offers tremendous resistance to tensile stresses. Iron and silicon give luster to aluminium and make it hard. Sodium and all other alkaline metals are bad impurities in alloys poor in copper. They offer no resistance to the destructive influences of dampness and air. Aluminium bronze has from 7½ per cent to 10 per cent copper. Above 10 per cent alloys become brittle. Aluminium with less than 50 per cent zinc gives a non-crystallizing mixture of metals which is so homogeneous that the two metals cannot be recognized under the microscope. Nickel increases its elasticity. Magnesium-aluminium alloys do not resist the destructive influences as well as other alloys. They are expensive but very light. An alloy of this type contains Al 95.5 per cent; Mg. 1.50 per cent; Cu. 1.50 per cent; Ni, 1.50 per cent. Manganese and chromium increase hardness so that with 50 per cent chrome the aluminium cannot be worked. Tin in small quantities makes it fragile. The alloys of aluminium flow more easily than aluminium and it was tried to use them in soldering aluminium. But the tests were failures due to the great heat conductivity and oxidation of aluminium. Tungsten hardens aluminium and increases its resistance to any destructive influences.

Grinding.

Chilian Mills.—In Russia the Chilian mill is almost universally used to crush ores preliminary to amalgamation. Its use represents a slow development from the practice of crushing ores by spreading them on the main thoroughfares for carts and other vehicles to pass over. In *Bulletin* No. 75 of the Inst. of Min. and Met., London, Mr. H. C. Baylon traces the development of crushing in Russia and gives the results of present practice. The following tabulation gives a summary of the type of mills used in five typical plants.

	Plant No. 1	Plant No. 2	Plant No. 3	Plant No. 4	Plant No. 5
No. of Chilian mills in plant.	6	4	6	4	6
No. of runners per mill....	2	2	2	2	2
Diam. of pan across bottom.	7'6"	8'9"	8'2"	10'6"	9'4"
Distance between centers of runners	6'	6'9"	6'6"	8'3"	7'6"
Width of face of runner, in	10	10	10.5	10	12
Diameter of runner..	5'6"	5'6"	6'6"	5'6"	6'6"
Weight of runner, lb.....	5400	5400	5400	5400	108,000
Revolutions of mill per min.	12	14	13.5	12	11
Mean travel of runner per min	226'	297'	275'	311'	260'
Height of discharge, in.....	10	10	9.75	9.5	8
	Screen Used.				
	1 mm slot.	1 mm slot.	¼ to ⅛ in. punched.	1 mm slot.	1 mm slot.
Water to ore ratio, by weight.	10:1	6.5:1	7:1	13:1	..
Duty per 24 hours, tons....	16	26	20	19	24
Amalgam collected—					
from mills.....	86%	81%	83%	85%	..
from plates.....	13%	18%	16%	15%	..
from sundry sources....	1%	1%	1%	....	..

No accurate data on power is available, but it is estimated at from 7 hp to 12 hp per mill. The following sizing tests show the work of the mills.

1 mm screen	Plant No. 1 %	Plant No. 2 %	Plant No. 3 %	Plant No. 4 %
+30	0.2	0.4	0.2	0.1
+60	2.41	14.84	3.2	1.3
+100	14.72	22.06	12.80	11.0
—100	82.67	62.70	84.80	87.6

As the main object in view is to crush the ore very fine and to obtain the highest possible percentage of gold by amalgamation in the pan itself the results are excellent. The low capacity of the mill is due to the purpose for which it is used, and little change will be made as long as this factor remains the guiding principle in the milling practice of the country.

A pair of tires will serve for the crushing of about 9000 tons of ore and will weigh about 7800 lb. per pair when new and 1000 lb. per pair when discarded. Dies weigh 5000 lb. per set when new and about 2200 lb. when discarded.

Inside amalgamation is standard practice. Clean-up is made once a day, and 6 lb. to 8 lb. of mercury are added to the mill during the 24 hours. Outside hard plates are used and kept under locked covers. They are cleaned once a month.

Hand feeding is usually practiced, but the author describes a new mill for which he has provided an automatic belt feeder. The belt receives ore from a chute and is moved toward the mill a certain distance with each revolution of the runners. The device is automatic and increases or decreases the quantity of ore as needed.

The new Chilian mill, also devised by the author, which is used in connection with the automatic feeder, has runners 6 ft. in diameter, 1 ft. face and weighing 10,000 lb. It makes 14 r.p.m. to 16 r.p.m., the runners having a rate of travel of 340 ft. and 427 ft. In the mills running at 14 r.p.m. it has been found that a height of discharge of 9 in. to 10 in. and a water ratio of 4.5:1 to 5.5:1, with 1 mm screens, gives the best product and results. The wear of tires is from 0.62 lb. to 0.65 lb. per ton crushed, and of dies, 0.28 lb. The life of the slotted screen is about 1200 tons of ore crushed. The capacity of the machine is 42 tons per 24 hours.

The author believes that if the same amount of thought and attention were devoted to the development of the capacity of this type of Chilian mill as has been devoted to the heavy stamp-tube mill combination in South Africa it would prove a serious rival and give a product nearer to the ideal aimed at in that goldfield.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Blast Furnace Charging Apparatus.—With the object of regulating and evenly distributing the charge, and avoiding the abrasion of the inner wall of the furnace by the falling charge, Mr. J. B. McKennan, of Pueblo, Col., has devised the charging apparatus shown in the illustration, Fig. 1. A metal stock guide 1 with stock bell 3 and apron 3 form annular walls which guide the descending charge, causing it to fall in rings and preventing it from striking the wall of the furnace. The main stock hopper is shown at 5 and the receiving hopper at 8, the latter being closed by the bell 9. The distributing hopper 10 is provided with a spout 12, eccentric to its axis, and is designed to rotate above the receiving hopper. This movement

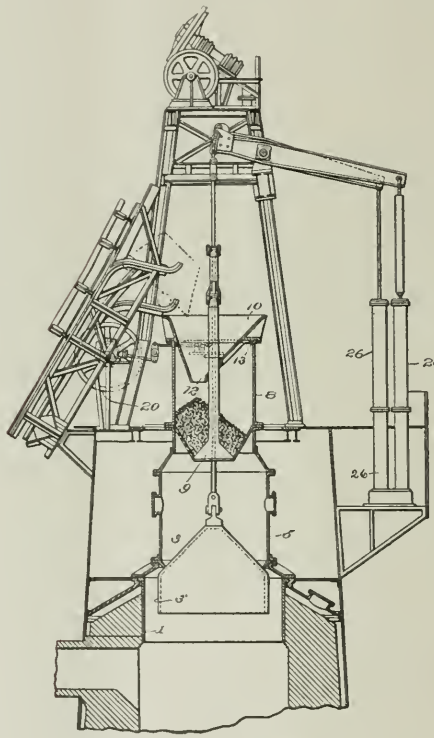


FIG. 1.—BLAST FURNACE CHARGING APPARATUS

is accomplished by means of rack and pawl mechanism connected with one of the sheaves 20 of the cable hoists.

In operation each skip load of material is delivered through the distributing funnel to the receiving hopper, as shown in the cut, the bulk of the charge being deposited eccentric to the axis of the hopper. The bell 9 is then lowered by the power cylinder 23, and the charge is deposited on the bell of the main hopper. The bell 9 is then closed again. During this interval another skip load is being hoisted and the funnel 10 has been shifted to another position, so that the eccentricity of the next load is different in position. Each succeeding load is advanced in position in a similar manner. In practice Mr. McKennan proposes to deposit nine skip loads of material in each furnace charge, six of coke, two of ore and one of limestone, preferably in the order mentioned. The distributing funnel will have seven positions and be moved two-seventh of a

revolution after each skip load has been deposited. Thus in seven furnace charges each skip load of the charge will have been put through the funnel while in each of its seven positions. After a full charge has been deposited in the hopper 5, the bell 3 is lowered by the power cylinder 26, and the charge falls through the annular opening formed by the stock guide and apron of the bell. (980,811, Jan. 3, 1911.)

Gold and Silver.

Agitator.—In the agitation of ore pulp it is desirable to have air valves which permit of flow in one direction only, so that should the air pressure be cut off for any reason, pulp cannot return into the nozzle. Such a nozzle is described in specifications of a patent granted to LOUIS T. SICKA, of Milwaukee, and assigned to the Allis-Chalmers Co. The details of the nozzles and their application to the bottom of an agitation tank, are shown in Fig. 2. There may be as many nozzles as desired, arranged around the bottom of a tank; one set projecting through the tank wall into the central well, and the other flush with the wall. In operation the set projecting into the central well cause the pulp to flow upward through it, and cause the agitation. The other nozzles are useful only in breaking up accumulations of slime or pulp when starting the agitator.

The pipe forming the nozzle 10 is capped at the end and perforated with a series of openings 11, which are larger at

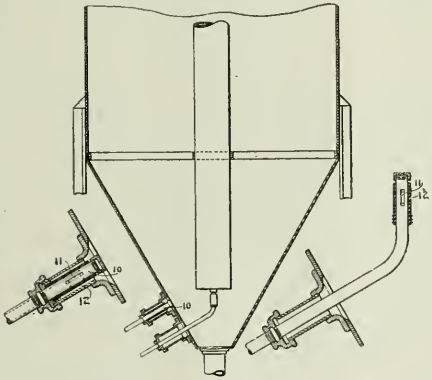


FIG. 2.—AGITATOR

the outer surface of the pipe. A piece of rubber tubing 12, is placed on the pipe over the openings, and fastened to the pipe at its lower end by cords. When air is forced through the pipes, it will press the rubber tubing outward and escape into the tank. If the pressure of air is cut off, the material above and outside of the nozzles will close the same and prevent the entrance of any foreign material. (978,667, Jan. 3, 1911.)

Suction Filter.—A continuous filter for cyanide pulp, which is also automatic in its action has been devised by HARRY E. KIER, of Colorado Springs, Col. The device is a circular rotary structure, upon which are mounted horizontal filter trays to receive and filter the pulp. In one revolution of the structure each tray is in turn brought into connection with suction for removing the cyanide solution, then wash water and suction to remove the weak wash solution, and finally with compressed air to loosen the cake and assist in discharging it when the tray is automatically dumped.

The filter is shown in vertical and horizontal sections in Figs. 3 and 4 respectively. As shown, there are eight filter trays 24, each connected by flexible hose with a set of pipes 51, 52 and 56, these being connected by means of valves 54, 55 and 56, with the circle tubes shown in the center of Fig. 4. The circle tubes have connections through the central hollow column with suitable pumps for creating vacuum or pressure. The pipe 51 is the suction pipe and is connected with suction pump 95; through it the first cyanide solution is withdrawn. The pipe 53 is a second suction pipe, connected with a suction pump 94, by means of which the weak wash solution is drawn off. The pipe 52 is the air pipe which delivers compressed air to the filter tray from a compressor, such as 96.

In operation the trays pass under a source of flowing pulp, as the pipe 47. After receiving a charge of pulp the valve

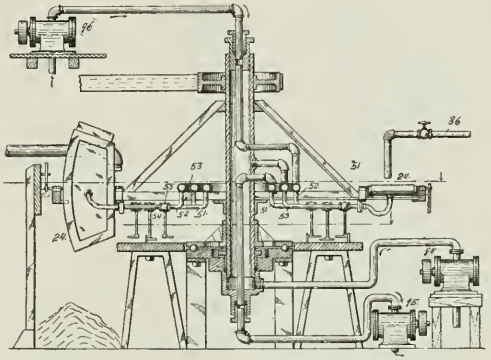


FIG. 3.—VERTICAL SECTION OF SUCTION FILTER.

connecting with the first suction pump is opened and cyanide solution drawn off. This suction continues until the tray reaches the wash water pipe 86, when it is cut off, and the cake subjected to a wash of water to remove the remaining traces of cyanide solution. From this point the valve connecting with the second pump is opened and the wash solution drawn off into a suitable receiver. Finally, the tray reaches

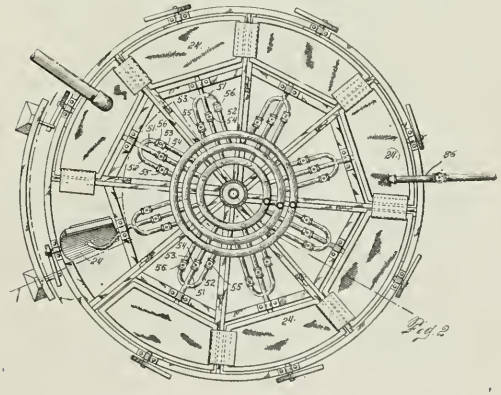


FIG. 4.—HORIZONTAL SECTION OF SUCTION FILTER.

a point where the suction is cut off and compressed air delivered into the filter tray under the cake, thus loosening it ready for discharge when the tray is dumped as shown in the illustrations. The action is automatic, the various valves being opened and closed by means of triggers mounted on the stationary platform 7, and engaging the stems of the valves as shown in Fig. 3. (978,381, Dec. 13, 1910.)

Copper.

Reverberatory Furnace.—In a patent recently granted to Messrs. W. J. MURPHY and A. E. CULLEY, of Butte, Mont., a new form of reverberatory furnace is suggested, the objects

being to provide a different method of feeding the charge, and a more efficient method of applying the heat. As shown in Fig. 5, which is a plan view, it is proposed to feed the charge at two constricted portions of the hearth area, through a system of hoppers and screw feeders, delivering the material at a point slightly above the level of the molten mass on the hearth. The inventors believe that this is preferable to dropping the charge in a mass from the top of the furnace. In

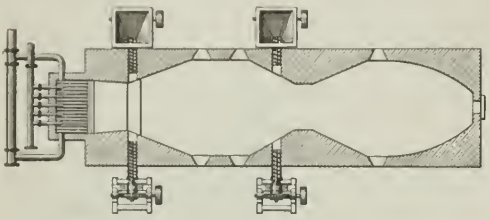


FIG. 5.—REVERBERATORY FURNACE.

applying the heat to the charge it is proposed to do away with the bridge, and direct the flame from the elevated fire-box, downward on the hearth of the furnace against the charge, by means of a series of jets of air or steam. (978,621, Dec. 13, 1910.)

Baggaley Process of Smelting.—Metallurgists will recall the experimental work of RALPH BAGGALEY, of Pittsburg, Pa., as carried on at the smelting works of the Pittsburg and Montana company at Butte, Mont. It was his intention to provide an inexpensive means for the direct treatment of gold, silver and copper ores, without preliminary concentration or other treatment, and thus avoid the losses incident thereto. Where such ores contained sufficient fuel to provide heat for smelting he added no carbonaceous fuel. His furnace was a converter-smelter, for the production of a low grade matte, which he afterwards enriched and converted by treatment with silicious ores, after the manner of the present practice of basic converting. The process was in operation for several months, but later abandoned. Metallurgists interested in knowing the details of Mr. Baggaley's ideas may find a very complete description in his latest patent specifications, numbers of which are given herewith. (977,922, 977,996, Dec. 6, 1910.)

Zinc.

Volatilization Process and Furnace.—An improvement in the metallurgy of zinc is the basis of a patent granted to HARRY H. HUGHES, of Springfield, Mo. In our issue for August, 1910, page 502, we gave some details of the furnace, which consists of a revolving cylindrical retort and condenser, with provision for keeping air from entering the apparatus. The feed is continuous, and spelter and residue are discharged separately. In a more recent patent granted to the same inventor, he provides a larger and stationary condenser in place of the small rotating one indicated in his previous patents. (980,640, Jan. 3, 1911.)

Furnaces.

A stirring device for mechanically rabbled furnaces has been patented by UTLEY WEDGE, of Ardmore, Pa. The illustration Fig. 6 represents a vertical cross section of the improved rabble arm and rabble. The stirrer arm is represented by 1', the rabble head by 3' and the rabble blade by 2. The rabble head rests on the stirrer arm, and is held in position while moving by means of the bent portion 4. The rabble head may rise to ride over any obstruction on the furnace hearth, the pivotal axis being near the point 4. Rabble arms may be interlocked on the arm by means of a tongue projecting from one into a recess of the other, as at 6. The rabble blades are made independent of the head, so that they may be removed when worn. They fit onto the head with a dove-tailed joint as shown. In order to prevent the corrosion of the stirrer arm it is provided with sleeves of refractory material as shown at 11. The advantages claimed by the inventor are the ease of removing any rabble head or blade, and the reduction in cost of the parts and of the roasting operation. Alternate forms of the interlocking rabble head, and of the method of applying the refractory sleeve, are shown in the specifications. (980,717, Jan. 3, 1911.)

Dust prevention in the discharge of roasted ore from a furnace, is the object of a device patented by THOMAS EDWARDS, of Ballarat, Australia. The machine consists of two or more cylindrical chambers mounted one above the other. A central rotary spindle extends vertically through these chambers, and carries a number of apertured discs which rotate in the bottom of each chamber. As the discs revolve they periodically register with openings in the bottom of each chamber and allow ore to pass to the chamber below; but the discs are so arranged on the spindle that the apertures in all the chambers are never open at the same time, so that the ore is discharged from the furnace in small quantities by a series of short drops. This prevents the dusting which is inevitable when large quantities of hot are discharged from a furnace. If desired the chambers may be water-cooled, and the ore thereby cooled for a subsequent operation. (979,937, Dec. 27, 1910.)

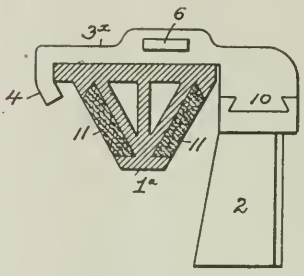


FIG. 6.—STIRRING DEVICE.

Granulating Slag.

A process for granulating molten slag recently has been patented in the United States by HEINRICH COLLOSEUS, of Berlin, Germany. It already has been proposed to granulate slag by allowing it to flow over a rotating drum having nozzles in its outer circumference through which liquid could be ejected for the purpose of assisting the granulating process, or for adding a desirable substance to the slag. The present inventor finds that the process is greatly facilitated by directing horizontal streams of fluid just beneath the flowing stream of slag and just as it falls on the revolving drum. (980,606, Jan. 3, 1911.)

Electrolytic Processes.

Chlorates and Hypochlorites.—In the production of chlorates and hypochlorites by chloride electrolysis without a diaphragm (as well as in other electrolytic processes) it is of importance to prevent the hydrogen bubbles set free at the cathode from sticking to it. For this reason Mr. FRANK H. BRIGGS of Elyria, Ohio, does not use a straight plate for cathode, but forms it with slanting sides so that the hydrogen bubbles detach and rise. For chlorate and hypochlorite production he uses a lead cathode coated with platinum. (981,371, Jan. 10, 1911.)

Persulphuric Acid.—In the past the production of persulphuric acid by electrolysis of sulphuric acid has been carried out by cooling the electrolyte below 10° C. and adding some hydrochloric acid to increase the yield. Dr. G. TEICHNER of Nürnberg, Germany, has discovered that no cooling of the whole electrolyte is necessary and that the addition of hydrochloric acid may be omitted, if only the anode is maintained at about 15° C. The electrolyte may be at a temperature of more than 20° C. He makes the anode hollow and passes water of 12° C. to 15° C. through it. He uses sulphuric acid of specific gravity 1.35, an anode of platinum, a cathode of lead and an anodic current density of 0.5 amp. per square centimeter. The electrolysis is carried out as rapidly as possible to avoid

the formation of Caro's acid which has a depolarizing action. This is effected by making the volume of the electrolyzed liquid as small as possible, for instance, 5 cu. cm to 10 cu. cm electrolyte for 1 amp. (981,900, Jan. 17, 1911.)

Cleaning Metallic Articles.—To clean articles of gold, silver and other metals from the tarnish of oxide and the like, Mr. H. E. GENET of New York City immerses the articles in a solution of hot sodium hydroxide and brings them in contact with a piece of aluminium, for instance, in form of a perforated aluminium dish. This represents a short-circuited galvanic couple and the oxide tarnish is reduced to metal. (980,627, Jan. 3, 1911.)

Mercury from Poor Ores.—To extract mercury from poor ores, Mr. BELA SZILARD of Paris, France, uses as lixiviating agent a mixture consisting of a hot solution of sodium chloride and of calcium sulphide obtained as a by-product in the manufacture of soda by the Leblanc process, or of other calcium sulphides rendered alkaline by oxide of calcium. The tepid mixture is added to the finely crushed ores, and after standing for two days a current of hot air is passed through the mixture from below. The calcium sulphide is converted into calcium thiosulphate which is deposited with the ores. The liquid discharged contains mercury not in the form of a sulphide but in the form of a double salt solution of mercury and sodium chloride which is submitted to electrolysis. Any silver contained in the ore is also dissolved and precipitated by electrolysis. The lixiviation is best carried out in an autoclave boiler at a pressure of four atmospheres. (980,364, Jan. 3, 1911.)

Electric Furnaces.

Ferroboron.—To produce carbon-free ferroboron for use in the steel industry, Mr. C. A. HANSEN of the General Electric Company, Schenectady, N. Y., uses silicon as reducing agent. The mineral colemanite $Ca_2B_6O_{11}.5H_2O$ is first dehydrated by heating to a temperature of 450° C. It is then powdered and mixed with powdered ferrosilicon (since this is cheaper than pure metallic silicon). Sufficient silicon must be added to reduce all of the boron. The powdered mass is mixed with a suitable binder, like waterglass, and pressed into pellets. The pellets are heated in an electric furnace to a temperature above 2000° C. The silicon reduces the boron and is itself oxidized to silica which combines with the calcium to form a slag of calcium silicate which separates perfectly from the metal matrix. (982,135, Jan. 17, 1911.)

Pure Metals.—Pure tungsten, tantalum, vanadium, etc., are produced by Mr. WM. C. ARSEM, of the General Electric Company, by reduction in his vacuum tube furnace. The reducing agent is very finely divided carbon, produced from lampblack by heating to a very high temperature in a vacuum or by graphitizing. (979,363, Dec. 20, 1910.)

Carbon Electrodes.—In the manufacture of large carbon electrodes, Mr. WALTER GASTON of New York City proceeds as follows to prevent them from cracking. The electrode as it comes from the baking furnace hot is plunged into boiling tar and pitch so as to impregnate it thoroughly. The pores are thereby filled. The operation really adds a second binder to the electrode. The absorption of moisture during storage and transportation is thereby avoided. The addition is stated not to impair the conductivity, but rather to improve it. (979,465, Dec. 27, 1910.)

Carbon Electrode.—In the manufacture of very large amorphous carbon electrodes, such as are now used in electric steel furnaces, considerable difficulties are experienced in baking them uniformly. Mr. RAPHAEL H. WOLFF, of New York City, proposes to overcome this difficulty by making the electrode essentially of two pieces, an exterior and an interior. The exterior part is hollow, for instance, a cylinder, the inner hole being wider at one end than at the other. This hole is then filled either with one solid piece or a combination of pieces, and tar pitch is infused to bind the whole together. (974,008, Oct. 25, 1910.)

Recent Progress in the Electrometallurgy of Iron and Steel.

A two-ton Roechling-Rodenhauser electric combination induction furnace is to be erected by the American Electric Furnace Company of New York some time during this year at Lansdowne, Pa. This is the first furnace of this type to be installed in this country.

The Girod furnace installed by the Simonds Manufacturing Company at Chicago is now being transferred to their new and modern plant at Lockport, N. Y., and will be used for producing and refining the high-grade steels required for their saws, machine knives, files, etc.

Interesting information comes in reliable private reports from Europe. Amorphous carbon electrodes are there in successful use in steel furnaces in sizes up to 600 mm (24 in.) diameter. In commercial practice with European Heroult furnaces the consumption of these carbon electrodes is from 3 kg to 4 kg per ton of steel. A new 20-ton to 22-ton Heroult furnace—the largest in size so far—is soon to be erected in a German works. In another German works using the Heroult furnace (seven tons capacity) the energy consumption (with fluid charge) is given as 105 kw-hours per ton of steel.

From Messrs. C. W. Leavitt & Company of New York, the American representatives of the Girod furnace, we have received the following data on the experience of the Cockerill Company in Seraing, Belgium, who have been operating a 4-ton to 5-ton Girod furnace for the past two years and are refining molten Thomas steel for rails, locomotive tires, cannons, automobiles, etc. The total cost of power, labor, furnace upkeep, electrodes and general expenses for the operation of this electric furnace is given as less than $4. The Krupp firm in Essen is installing a 12½-ton Girod furnace to be operated by three-phase currents.

* * *

As to the reduction of iron ore in the electric furnace interesting information comes from Scandinavia.

The large 2500-hp electric shaft furnace at Trollhättan was put in operation in November and appears to work perfectly well. The furnace was first tapped on Nov. 16, and from a private report written two weeks later we learn that the furnace had been working without difficulty since the start, and especially that none of the electrodes had been touched in that time.

From another report, dated Dec. 10, it is learned that the furnace had been running continuously since Nov. 15 and was working quite satisfactorily.

The furnace plant is equipped with apparatus for making gas analyses and taking temperatures at every 3 ft. of the whole stock. There is also a very elaborate laboratory for testing the raw materials and the product obtained.

The yield was for the first week 2090 kg (6578 lb.) per hp-year, but has since been raised to above three metric tons (about 6600 lb.) per hp-year.

The consumption of coal amounted to 24 hectoliters to 26 hectoliters (85 cu. ft. to 92 cu. ft.) per 1000 kg (2204 lb.) of iron.

The works are not permitted to be examined by outsiders at present. But all on the spot, even previous enemies of the system, are reported to be enthusiastic over the undertaking.

At Domnarpoet, when the first electric iron-ore smelting experiments by this system took place, a new furnace for from 4000 hp to 5000 hp is at present under erection. It is expected to be ready for operation in May.

At Tyssedal, Hardangen, Norway, there is an installation of two electric shaft furnaces, each for 3500 hp, one of which will be put in commission next summer.

The Aktiebolaget Elektrometall, in addition to the foregoing, has given licenses to several other works, one of which is the Uddeholms Iron Works in Sweden, for which they are now preparing drawings for three furnaces of 3000 hp each.

Heating Liquids by Direct Steam.

In this article a description will be given of some appliances, made by the Schutte & Koerting Company, of Philadelphia, for heating liquids by means of direct steam.

Fig. 1 shows a continuous water heating device. The steam enters at *S* and the water at *W*. Heating is effected by the mixture of steam and water taking place through nozzles in the instrument. *D* is the discharge.

This apparatus may also be used as a condenser and as an exhauster for heating coils and on the exhaust pipe of engines. It can be used with steam of any pressure or with exhaust steam for the maintenance of a partial vacuum, or as an exhauster for the removal of air only. The capacity and range in volume is regulated by the central hand wheel, and when used for heating water the temperature is regulated by throttling with a valve in the steam pipe.

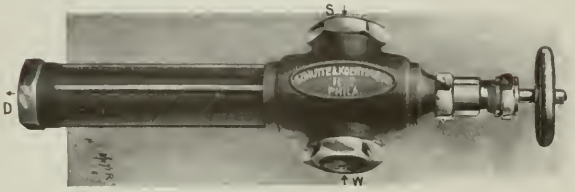


FIG. 1.—CONTINUOUS WATER HEATER

Fig. 2 shows the use of the attachment for the production of hot water. The water is supplied from the bottom, the steam from the top. Valve *B* is a stop valve in the water supply pipe, while the volume of the water is regulated by the spindle *A*. The temperature is regulated by throttling with valve *C* in the steam supply. This valve can also be used as a stop valve in the steam pipe, but it is more convenient to provide a separate lever valve *D* for turning on and shutting off the steam. To start, the water is first turned on at *B* and then the steam turned on at *D*. To stop, the steam is first shut off at *D* and then the water shut off at *B*.

Fig. 3 shows an attachment to an engine exhaust or heating coil. In this attachment it is necessary to have a check valve *E* to prevent a back flow of water, and also a free exhaust valve *F* is required. The discharge is run free to waste or all or part of it into a hot water tank for use.

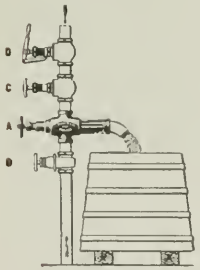


FIG. 2.—ATTACHMENT FOR PRODUCTION OF HOT WATER.

Fig. 4 shows the application of this continuous water heater for making warm water in different stories of buildings. The installation is very simple. *S* being the steam pipe, *P* the water pipe, *H* the heater.

The continuous water heater offers a means of heating the water exactly to the desired temperature, the regulation being obtained by the steam spindle.

The circulating heater, Fig. 5, does satisfactory work when water need not be heated above 160° and where steam pressure is not less than 35 lb. This circulating heater, within the above limits of temperature and pressure, works without noise or nearly so, and works very economically. *S* is the steam inlet, *D* the discharge of the hot water.

The disturbing noise and destructive effect on open tanks, when steam is introduced into the water through perforated pipes, is entirely obviated by attaching the noiseless heater to the end of the steam pipe. The heater discharges in the form of an inverted cone, offering a large surface and thus causing rapid condensation (Fig. 6).

If water is to be heated to a temperature not exceeding 165° Fahr. the air inlet on heater is closed with a plug and nothing but the heater is needed, as it will be noiseless with any pressure of steam from 5 lb. to 100 lb. and above. If water is to be heated to a temperature above 165° Fahr. and up to boiling point, then an air pipe must be attached. The purpose of this pipe is to permit air to be supplied to the discharge jet and by its admixture prevent the formation of steam bubbles which otherwise in rising to the surface would escape and collapse and thus cause the noise. Thus water is heated in less time and with less expenditure of steam than could be done by the use of a perforated pipe. The air is drawn in through the action of the heater if the steam pressure is proportionate to the depth of the water in the tank above the apparatus, viz:

Height of water above heater or depth of tank in feet	3	4	5	6	7	8
Minimum steam pressure, in pounds	4	8	12	18	24	32

Thus under the above conditions the heater will work without noise up to any temperature by the addition of the air pipe, which is best fitted with a cock or valve. The heater works also with exhaust steam. Fig. 7 shows an installation.

Bureau of Standards Analyzed Samples.

The following announcement of the Bureau of Standards in Washington, D. C., will be of interest to chemists and metallurgists at large.

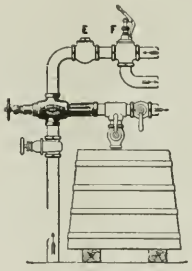


FIG. 3.—ATTACHMENT TO ENGINE EXHAUST.

The Bureau of Standards is ready to supply the following new analyzed samples:

No. 29, titaniferous magnetite ore from New Jersey.

No. 11a, basic open-hearth steel with 0.2 per cent carbon.

No. 12a, basic open-hearth steel with 0.4 per cent carbon.

The steels replace the original samples which are exhausted. Renewal samples of Bessemer steels with 0.1 per cent and 0.2 per cent carbon are in process of analysis, also a nickel and a chrome-vanadium steel.

The Glasgow Smoke-Abatement Exhibition.

By John B. C. Kershaw.

The Glasgow Smoke Abatement Exhibition, which was recently held in the Victoria Skating Rink in that city, has been an unqualified success, the attendance having averaged over 4000 persons per day, and a large amount of business having been obtained by the exhibitors as a direct outcome of their displays of smoke-abatement appliances.

The exhibition was promoted by the corporation of the city, and the three sections into which the exhibition was divided were under the charge of the leading officials connected with the gas, electricity and sanitary undertakings of the city. The success of the exhibition was no doubt largely due to the rivalry between the gas and electricity sections of the exhibition as to which should have the better display.

This combination of gas and electricity appliances under one roof is a new feature in smoke-abatement exhibitions, and the success of the Glasgow exhibition indicates that it is one worth imitating. There is no doubt that where, as in Glasgow, both the gas and electricity supply undertakings are owned and worked by the municipality a joint exhibition of cooking and

heating appliances can be easily arranged and can be made very attractive to the general public.

In their prospectus the promoters of the exhibition state "that for a considerable time the Corporation of Glasgow has been devoting much attention to the problem of air purification, recognizing how necessary an unpolluted atmosphere is to the mental and physical well-being of the community. Special committees have gone into the subject with extreme care—reports have been prepared by experts—and numerous experiments have been made and the results recorded. From these it is clear that the question of smoke abatement is a vital one in towns. Having regard to this fact the corporation has decided that it would be most desirable to bring before the citizens of Glasgow and the west of Scotland, in a practical and interesting fashion, the latest and most scientific devices in connection with smokeless heating and lighting, embracing the most up-to-date methods and contrivances for the complete combustion of fuel, as well as the most modern developments in heating, lighting and ventilation by means of gas and electricity. The readiest means of interesting the public in these matters is to hold an exhibition which will bring before them in a concrete form the advantages to be gained and the economies to be effected by the adoption of recent discoveries; and the corporation has accordingly entrusted to a special committee the organization of such an exhibition."

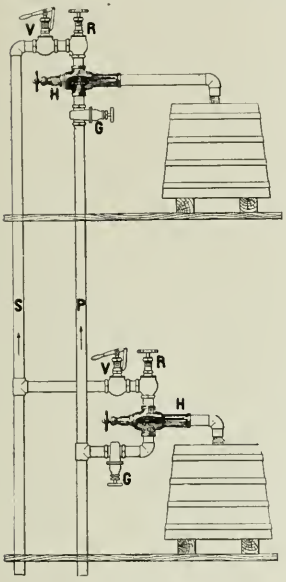


FIG. 4.—PRODUCTION OF HOT WATER IN DIFFERENT STORIES.

The appliances and materials exhibited at Glasgow related chiefly to the domestic smoke problem, and only a few of the exhibits dealt with the abatement of the smoke arising from the furnaces and steam boilers used in manufacturing operations.

FIG. 5.—CIRCULATING HEATER.

The exhibition was divided into three sections, the largest and most important being that devoted to the display of gas lighting, heating and cooking appliances. The second section was devoted to the use of electricity for similar purposes, and the third section was given up to exhibits of smokeless fuels and of patent stoves and grates.

The most important and striking exhibits in the first two sections were the model houses in which every lighting, heating and cooking appliance was operated in the first case by gas and in the second case by electricity.

In some of the smaller Scotch towns the gas committees are now installing gas grills, stoves and meters free of charge, it having been found that this free gift of gas heating appliances pays the corporations by the increased sale of gas. The gas committee of the Glasgow corporation is contemplating a similar step and when this is taken the electricity committee is almost certain to follow in their footsteps. With electricity at 2 cents per kw-hour and gas at 50 cents per 1000 cu. ft. the cost of lighting and heating one's house by either gas or electricity in these Scotch towns need not be prohibitive; and when the public has become accustomed to the comfort and convenience of the newer methods of lighting and heating the days of the coal-fired open grates will be numbered. Already there are stated to be 80,000 gas cooking appliances in use in Glasgow alone, and their use is extending.

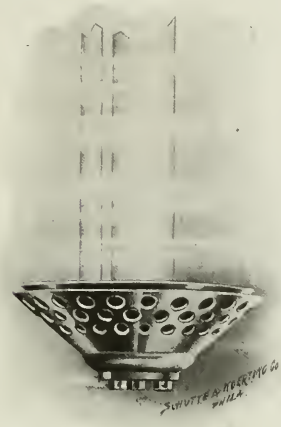


FIG. 6.—CONE DISCHARGE.

In Section II of the exhibition a wonderfully varied and attractive display of electric lamps and heating appliances was shown, and here again it would be invidious to single out any special firm's display for mention in this report. The exhibit of electric radiators and of cooking appliances was probably the best that has yet been collected in the United Kingdom and there is no doubt that the Glasgow Electricity Undertaking will have a largely increased day load as a result of the 1910 exhibition. It may be stated here that in Glasgow it is proposed to supply electricity for heating from the lighting circuits, the amount consumed being calculated from the total current registered by the meter, and the average lighting consumption in previous quarters. All above the average lighting consumption is charged at 2 cents per kw-hour and is assumed to have been used for heating purposes. By this system the expense of separate wiring and meters is avoided.

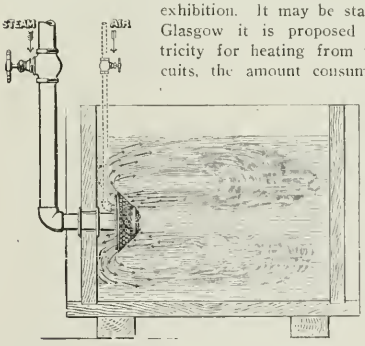


FIG. 7.—CONE DISCHARGE INSTALLATION

The smokeless fuel and patent grate and stove section of the Glasgow exhibition was the smallest of the three and attracted less public attention than the other two for reasons which are obvious, since electricity and gas lend themselves in numberless ways to the creation of striking and attractive exhibits. The smokeless fuel exhibits comprised peat from the

Glasgow distress committee's works at Palacerigg; anthracite from the Scotch and South Wales coal fields; coke, "charco," "coalseld" and "coalite."

The three last are patented smokeless fuels, with a limited sale in the United Kingdom.

"Charco" is made by subjecting ordinary coke or partially carbonized coal to a special treatment before cooling, as it comes from the gas retorts or coke ovens. It ignites readily, burns steadily and gives out greater radiant heat than either coal or coke, while being absolutely smokeless.

"Coalseld" is a similar fuel, produced from ordinary gas-coal by use of a modified form of gas retort. It is now being made at the Glasgow Corporation Gas Works, and sample orders are being supplied by the gas department of the city.

"Coalite" is coal carbonized at a low temperature (700° Fahr.) and supplied by the British Coalite Company, whose main works are at Barking, near London. It is unfortunate that the company producing coalite has not so far been a financial success, and that in the north of England it is practically impossible to purchase this fuel for trial purposes.

The improved grates and stoves at the Glasgow exhibition were chiefly of the well-known and widely used types, the only novelties being open grates fired on the under-feed principle, which has been successfully used for boiler firing. These grates have been placed on the market by the White Flame Fire Company, of Stirling, and depend for their action upon the partial coking of the raw coal before it reaches the well of the grate. A hopper at the back of the fire contains the day's supply of fuel; this hopper is closed with a sliding gas-tight cover.

The coal is heated and partially coked in this hopper, and the hydrocarbon gases can find no outlet except through the opening into the grate, where they are consumed as they pass through the glowing fuel. The solid fuel is raked forward through the opening of the hopper as required to replenish the fire.

The makers of this grate do not claim that it is entirely smokeless, since when the fire is first lighted it labors under the disadvantages of the ordinary open fire as regards imperfect combustion and smoke production. But when the fuel in the grate and the hopper behind it have once attained their maximum temperature the fuel is smokelessly consumed, while all the advantages of the ordinary open fire, good ventilation and cheerfulness, are retained.

The show of smokeless kitchen stoves for use with ordinary coal was also a good one. In these the conditions required for smokelessness are more easily attained, since most cooking stoves are now designed on the closed-in principle and only require scientific design to obtain perfect combustion of the fuel. In one of the kitchen stoves exhibited at Glasgow the combustion chamber of the stove is extended under the hot-water boiler, and the hot gases then pass under the hot plate and oven before escaping to the chimney. By this arrangement it is claimed that double the amount of hot water is obtained, with a saving of 50 per cent in fuel.

The use of glass for the doors of ovens and plate-warming chambers is another novelty found in some of the kitchen stoves exhibited by J. C. Parker & Company, of Glasgow, and attracting some attention.

A large number of anthracite and coke stoves were exhibited by various well-known makers, and there is no doubt that the popularity of the former for heating offices and rooms is widely extending, since they require little attention and are most efficient heating appliances.

The Glasgow exhibition, although confined to only one branch of the problem of smoke abatement, namely, that of "domestic smoke," has been, as regards attendance and results, more successful than the Smoke Abatement Exhibition held at Sheffield early in 1909, and its organization and management will probably prove the model for the succeeding exhibitions of this character which are certain to follow.

NOTES.

Chemists' Club.—The new home of the Chemists' Club is making very satisfactory progress and is rapidly nearing completion. The laboratory floors and the public stairway will be completed by February 1. Over half of the available laboratory and office space has already been rented.

The American Rolling Mill Company, of Middletown, Ohio, has just completed a splendidly equipped research laboratory costing approximately $40,000. The excellent organization which the company is known to have, has recently been strengthened by securing the co-operation of Dr. A. S. Cushman, of Washington.

The Hardinge Conical Mill Company have removed their offices from 35 Wall Street to 50 Church Street, New York City.

New Electrolytic Furnace Process for Metal Production.—On page 688 of our December issue mention was made of a patent of Mr. J. W. Beckman, of Niagara Falls, Ontario, for producing pure carbon-free metals and their alloys from their oxides and sulphides. The process is applicable to a great number of metals, since only those are excluded which are more electropositive. We learn that among other metals, vanadium and titanium have been produced by this process on a laboratory scale. The process depends upon the fusibility of a salt composed of CaO and the oxide of the metal to be produced. This fused salt provides a cheap electrolyte in which the oxide is dissolved and electrolyzed. On account of the high boiling point of this electrolyte, the metals, even those with a comparatively high melting point, can be deposited in a molten condition and the operation is very simple on this account.

The Stuart & Peterson Company, of Burlington, N. J., have sent us their new catalogue No. 219, which contains on 176 pages illustrations and concise descriptions of the many products of their manufacture, including plain and porcelain-lined ware, steam-jacketed kettles, hollowware, chemists' appliances and specialties, furnaces, heaters, vacuum pans, retorts, tanks, etc.

Calendars.—The Goldschmidt Thermit Company, of New York City, have sent us their pretty calendar for 1911 with a map of the United States and some interesting illustrations of thermit welds. The attractive calendar of the Lead Lined Iron Pipe Company, of Wakefield, Mass., has a fine picture of the new Pennsylvania Railroad Terminal in New York City, which contains over 100,000 feet of lead-lined iron pipe for hot and cold water, made by that company.

The Arizona Copper Company produced for the month of November, 1910, 1505 tons of copper.

The Bureau of Mines, Washington, D. C., has recently published two bulletins, Nos. 2 and 3, of the new series. They deal respectively with North Dakota Lignite as a Fuel for Power Plant Boilers, and The Coke Industry of the United States as related to the Foundry.

Bakelite.—Information No. 1, issued by the General Bakelite Company, 100 William Street, New York City, gives first a brief outline of what bakelite is and of the properties of bakelite C and liquid bakelite A. Then follow brief chapters on bakelite lacquer, bakelite protective enamels (for the protection of metallic surfaces, pipes and machinery against corrosion), bakelite-impregnated wood, bakelite adhesive cement (for joining iron, wood or other hard materials with a cement, which, after hardening, can withstand moisture, steam, alkalies, acids, solvents and heat), bakelite plastic mixtures, bakelite paper and bakelite-asbestos paper. The pamphlet, which is exceedingly neat, is concluded by directions for molding bakelite paper and other bakelite plastic mixtures and by a description of the bakelizer which is the apparatus for heating under increased atmospheric pressure to convert bakelite A and B into bakelite C.

Hardening and Annealing.—We have received from Mr.

C. U. Scott, of Davenport, Ia., his illustrated catalog on gas-blast furnaces, fuel gas plants, hardening furnaces for high-speed tool steels, tempering furnaces, annealing furnaces, etc. Mr. Scott acts as a consulting engineer for the heat treatment and hardening of steel, besides the manufacturer of furnaces and gas plant for this purpose. This firm also undertakes hardening of tools made from carbon or high-speed steels, also case hardening and heat treatment of steel in general.

Messrs. **Merck & Co.**, manufacturing chemists, of New York City, with works at St. Louis, Mo., and Rahway, N. J., announce the removal of their headquarters to 45 and 47 Park Place, New York City, on Jan. 1, 1911.

A Flux for Brazing Metals, notably cast iron, is composed of potassium cyanide, one part by weight, and borax glas, two parts. The pieces to be brazed are heated to redness, when the flux is applied and the parts immediately become cleansed and reduced. A brazing solder, such as Hungerford's No. 24, may then be applied with success. The flux mentioned was made the subject of a patent by W. W. McIver, of New York.

The **T. L. Smith Company,** Milwaukee, has issued a new catalog describing the Symons crusher. The Symons is a gyratory which is lighter and more compact than the usual form of this machine. The upright shaft is stationary, and the motion is imparted to the crusher head by means of an eccentric sleeve which revolves on the shaft.

Symons Brothers, Chicago, have recently designed a new disc crusher, not a grinder, which is said to have extraordinary capacity. It reduces rock or ore to ⅛ or ¼ in. and is designed to have a place in stage crushing. The manufacturers have issued a descriptive bulletin.

Camp Bird, Ltd., owning the Camp Bird mine at Ouray, Col., and the Santa Gertrudis property in Mexico, is prosecuting work at the latter in anticipation of the forced abandonment of the former within a year or so on account of exhaustion of ore. Developments at the Santa Gertrudis indicate that a modern mill will be erected, comprising stamps, tube-mills, cyanide tanks, filter presses, precipitation presses and appurtenant machinery. The plant will have a capacity of 600 tons per day. The November report of the company shows that the Camp Bird mill crushed 6577 tons of ore, producing 6218 oz. of bullion and approximately 723 tons of concentrate. The balance applicable to profit was $98,500.

The Arizona Copper Co. produced in the month of December, 1910, 1612 tons of copper.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in October, 1910, was 71. The total tonnage milled during that period was 1,916,801 tons of ore; 9756 stamps were in operation with an average duty of 7.31 tons per 24 hours. Tube mills in commission numbered 188. The yield for the month was 653,147 fine ounces gold. The statement shows a great advance over the same period for last year.

Evaporator Patent Suit.—The patent suit of the Kestner Evaporator Company versus the American Evaporator Company has recently been decided in favor of the defendant. The decision is written by Judge Holland. Infringement was claimed of patent 548986 of Oct. 28, 1895, of Peter Cooper Hewitt; the claim in question reads as follows: "In an evaporator provided with heating tubes and one or more return tubes, a separating chamber provided with a series of stationary tangential nozzles communicating with the heating tubes, substantially as specified." In the defendant's evaporator a film of liquid travels along the inner surface of the heating tubes under the influence of a vacuum to an upper chamber where the vapor is given a circular motion and the particles of liquid separated from the vapor by centrifugal force, after which the vapor passes off. But since the liquid is not returned for retreatment through "one or more return tubes," as the liquid passes once through each effect only, the above mentioned claim is not found to be infringed.

Gold pannings obtained in the field may be preserved for assay by using lead foil, 4 in. square, into which the values are washed after panning. The foil is bent into basin shape, and after receiving the washings, which are drained off, may be rolled up and marked with a number or letter. The gold is then in shape for cupellation, if very clean, or for fusion if mixed with much silica. The process avoids the possibility of loss or salting, and is suggested by Mr. J. E. Wood in *Bull.*, No. 75, I. M. M.

Solubility of Oxygen in Molten Silver.—Spitting and foaming of molten silver at the time of solidification and blistering of solid silver when it is rolled are due to dissolved oxygen. F. G. Donnan and T. W. A. Shaw recently determined the solubility of oxygen in molten silver and presented their results to the Soc. of Chem. Ind. They devised special apparatus and ascertained that the volume of oxygen (measured at 0 mm and 760 mm) dissolved in 10 g melted silver at 1020° C. and 751 mm pressure is 20.5 ccm. To combine with this quantity of oxygen 0.03 g of aluminium would be required to be mixed with each 10 g silver.

The Sulitelma Company, operating in Norwat, produced in the month of September 760 tons of copper concentrate by the Elmore process.

La Rose Consolidated Mines Co., operating in the Cobalt district, Canada, gives the following information relative to the milling of its low-grade ore, covering the period from Sept. 8, 1909, to May 31, 1910:

Average assay crude ore	28.58 oz. per ton.
Average assay tailing	3.68 oz. per ton.
Tonnage treated	18,423.56 dry tons.
Concentrate produced	628.02 dry tons.
Contents of crude ore	526,581.00 oz.
Contents of concentrate	416,629.59 oz.
Average assay of concentrate	663.00 oz. per ton.
Actual extraction	79.12 per cent.
Ratio of concentration	29.3 to 1
Net value of concentrate	$199,569.70
Cost of concentration	$61,351.54
Profit on low-grade ore	$138,218.16

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines for the month of August, 1910, was 72. Tonnage milled, 1,884,288; number of stamps dropping, 9685, with an average duty of 7.15 tons per 24 hours; tube mills in commission, 182. Comparison with last year shows fewer stamps and more tube mills at work, a larger tonnage treated, increased working cost per ton milled and decreased working profit. The yield for the month was 649,269 fine ounces, being larger than for any month in 1909 or 1910.

The Utah Copper Co. has issued its tenth quarterly report covering the third quarter of 1910. During July all three of the concentrating plants were in operation, but at the end of that month the Copperton plant, at the mine, was closed permanently and dismantled. The equipment is being used to remodel the Arthur plant at Garfield. The improvements at the Magna plant are completed and that plant is now treating in excess of 10,000 tons per day. When the Arthur plant is remodeled on the plans corresponding to those of the Magna plant it will have a capacity of about 6000 tons. The tonnage milled for the quarter was less than that for the preceding quarter, on account of the changes being made in plants as indicated. The production was 23,194,921 lb. of copper as compared with 25,124,052 lb. for the second quarter. The total net profit for the quarter was $1,329,966.88, from which a dividend of $1,168,882.50 was paid. The average cost per pound of copper produced, after smelter deductions were made, was 8.25c. as compared with 7.53c. for the preceding quarter. The increased cost of production is ascribed to the decreased tonnage treated. The average copper content of the ore was 1.5 per cent for the quarter and 1.4 per cent for the month of September.

Personal

Mr **W. H. Aldrich,** late managing director of the Consolidated Smelting & Mining Company, of Canada, is to have a voice in the management of two of the large Western copper companies, the Inspiration in Arizona and the Mason Valley in Nevada. His headquarters will be at Los Angeles.

Mr. **A. J. Bone** has been appointed smelter superintendent of the Copperhill smelter of the Tennessee Copper Company.

Dr. **Regis Chauvenet,** formerly president of the Colorado School of Mines, is at Mount Franklin, near El Paso, Tex., where he is engaged in tin mining and smelting.

Mr. **N. H. Darton** has been appointed geologist for the new Bureau of Mines. He has been connected with the United States Geological Survey for many years.

Mr. **Forrest Goody** has been in Denver for some weeks preparing plans and specifications for a metallurgical enterprise in Nevada.

Mr. **G. W. Heintz,** general manager of the United States Smelting, Refining & Mining Company, Salt Lake City, recently returned from a trip of inspection of the new baghouse system Company of Canada, with headquarters at Trail, B. C.

Mr. **Victor G. Hills,** Denver, investigated some tungsten deposits in Nova Scotia last month.

Mr. **Martin E. Hiltner,** Deadwood, S. D., is to have charge of the metallurgical opeerations of the New Reliance Mining Company, of South Dakota. He was recently in Denver purchasing supplies for the mill.

Mr. **Ernst Leitz,** of Wetzlar, Germany, the famous manufacturer of microscopes, etc., has received from the University of Marburg the honorary degree of Doctor of Philosophy, in recognition of his services rendered to science by having constructed for more than fifty years instruments of highest scientific value.

Mr. **Jay Lonergan,** Colorado School of Mines, '05, has accepted a professorship in the Imperial University, Tientsin, China, and has left Denver for his new field.

Mr. **R. H. Stewart** will succeed Mr. W. H. Aldrich as managing director of the Consolidated Smelting & Mining Company of Canada, with headquarters at Trail, B. C.

Mr. **A. H. Wethey,** who for many years has been in charge of Senator Clark's interests in Butte, retired from his position on Jan. 1. Senator Clark has disposed of so many of his former interests that the retention of a general manager is unnecessary.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ELECTRIC FURNACES (*Continued.*)

698,980 and 698,981, April 29, 1902. Johann Lühne, of Aachen, Germany.

Combined arc and resistance type. Material for making glass is melted by electric arcs and further heated by solid resistors. The melting apparatus is a trough-shaped receptacle having two flat vertically grooved side walls converging downward, jacketed with kieselguhr, ashes or glass wool. The negative electrode is a horizontal carbon bar, square in cross-section, extending from end to end of the trough near its bottom. It is hollow and perforated, and hydrogen, or a mixture of hydrogen and air, is supplied through it, being burned in the furnace to assist in melting the material. A series of horizontal positive carbon electrodes passes through each side of the melting chamber, their ends being in proximity to the angular sides of the negative electrode, and arcs being sprung between them and the negative electrode. The molten product passes down through a valved opening onto an inclined hearth. This hearth either contains resistors consisting of solid electrolytes, e. g., oxides of beryllium, calcium, zinc or titanium, or is composed of such materials containing embedded metal terminals. No current passes through these resistors, or resistor hearth, until they are heated by the descending molten material, or by hot air. As the molten glass passes over this hearth the light impurities derived from the electrodes are removed by a skimmer, or retained by a depending partition. The glass thence passes through a series of concentric annular chambers, partially separated by partitions or dams.

699,333, May 6, 1902, A. Ludwig, of Bernburg, Germany.

Arc type, for heating materials in an atmosphere of highly compressed hydrogen or other gas, e. g., for the production of artificial diamonds. The chamber is a thick metal tube, closed at the lower end and surrounded by superposed flat metal rings. The cover is a massive plate of metal, secured on the open end of the cylinder by a metal strap consisting of a series of plates bolted together. Opposed wedges are driven between the strap and cover. For very high pressures the strap is heated before the wedges are tightened. Th wedges may have division lines to regulate their position and the pressure of the cover on the cylinder. The gas pressure may be several thousand atmospheres. The electrodes are vertical carbon rods, one extending up from the bottom of the cylinder and the other depending and adjustably carried by a revoluble spindle journaled in the cover. A gas inlet pipe also passes through the cover. The spindle and pipe may be held by wedges.

699,654, May 13, 1902, Guillaume DeChalmot, of Holcomb's Rock, Va. Assignor to the Willson Aluminum Company.

Arc or resistance type, for reducing ores, or producing carbides or silicides. The furnace is a pot of iron lined with carbon, consisting of a lower and an upper section. It is mounted on a truck. The lower section has a tap hole for metal and may have one for slag. The upper section has a gas outlet. The pot constitutes one electrode; the other one is a depending carbon bar or group of bars. The furnace has a flat top made of closely interfitting hollow sections of cast iron, cooled by water circulated through them. Several electrodes may pass through openings in the cover. The electrode or electrodes and cover are maintained at the same potential by an electrical connection. The electrodes pass through stuffing boxes consisting of plates adjustably secured to the cover, with a packing of asbestos fiber. The cover is insulated from the pot by a 2-in. luting of brick, with or without clay or mortar. After the run the product or slag may be solidified in the pot and removed in one piece by first lifting off the upper section. Each electrode may carry 1000 or more amperes and may be independently switched into or out of circuit.

700,808, May 27, 1902, William R. Parks, of Chicago, Ill.

Arc type, especially for the production of calcium carbide. The furnace is a pot with brick sides. The lower negative electrode is a slab of carbon set in the floor of the pot, having a circular gutter leading to a central opening and drain. The positive electrode is a tube, vertically slitted and depending into proximity to the carbon hearth. The charge materials are delivered by a hopper and conveyer into the closed upper end of this electrode, and may be preheated therein by gas injected by pipes extending through the side of the electrode

701,218, May 27, 1902, Fausto Morani, of Rome, Italy.

The furnace is a shallow pot, receiving a depending electrode. The pot has a water-cooled metal base, or imbedded water pipes. The depending electrode is a square bar, or

group of bars, carried by water-cooled metal yokes. The terminals are also water-cooled and are clamped against flat faces of the bars. The taphole is a slit in a hollow water-cooled casing of steel. The product is tapped into a water-cooled metal pan. May employ several electrodes supplied by polyphase current.

701,986, June 10, 1902, E. G. Acheson, of Buffalo, New York.

Eelectrode and terminal for furnaces, consisting, as shown, of 72 horizontal square carbon bars, piled in contact, with a copper plug screwed into the end of each bar. A vertical copper plate is bolted to these plugs and has a marginal flange with openings, into which are soldered the terminals of the flexible cables, preferably of the same number as the carbon bars. Seventy-two graphitized-carbon electrodes, each 4-in. square, will carry a current of 37,500 amp. A receptacle to contain water is bolted to the plate flange, and is connected to an insulated reservoir.

702,081, June 10, 1902, A. Voelker, of Ehrenfeld, Germany.

Resistance type. Material for the manufacture of glass is intimately mixed with graphite or powdered charcoal, about 10 per cent, to give a conductive mass which serves as a resistor. Saltpeter is also added to eliminate the carbon as carbon dioxide. Pure carbon is necessary, and the ash slightly colors the glass. The carbon may be replaced by conductive oxides or metals, e. g., "brownstone," or powdered iron or silver. These are only suitable for the production of colored glass. A continuous or alternating electric current may be used.

BOOK REVIEWS.

Metallography. By **Cecil H. Desch,** D. Sc., Ph. D., lecturer in metallurgical chemistry in the University of Glasgow. 429 pages, 14 plates, 108 diagrams. Price, $3. New York: Longmans, Green & Company.

This is probably the most satisfactory book yet written on this subject in English. The treatment throughout is clear and free from the mysterious phrases of other treatises. It is comprehensible from beginning to end by any bright, industrious person, and will give a clear understanding and good foundation in the subject. Numerous references to original literature are given; the microphotographs are excellent. It is really a treatise on the physical and chemical properties of metals and alloys.

* * *

Elementi di Metallurgia. I: Siderurgia. By **Alfonso Greco.** 192 pages, 27 illus. Price, 1.50 lire ($0.45 retail price in New York). (Students' Library Series.) Livorno (Leghorn): Raffaello Giusti.

The characteristics of almost all Italian books, aside from their contents, are clear type, good paper, and wonderful cheapness. The equivalent of such a book as this in English would not be sold for less than $1. The metallurgy of iron as here set forth is elementary and slightly antiquated, about the equivalent of Bauerman's small English text-book, edition of 10 years ago. It can do good service in the Italian technical schools, however, for giving students their first ideas of the subject.

* * *

An Introduction to the Study of Metallurgy. By Sir **W. C. Roberts-Austen.** Sixth edition, revised and enlarged by **F. W. Harbord.** 478 pages, 189 illustrations. Price, $6 net. Philadelphia: J. B. Lippincott Company.

Mr. Harbord has given this well-known text-book a new revision and has improved it in many respects. His revision of Chapter III—the physical properties of the metals—is not altogether satisfactory, much valuable fundamental information being still omitted, such as many of the boiling points, all the latent heats of fusion and latent heats of vaporization. The claim that Siemens was the progenitor of the electrolytic furnace (page 111) is very far-fetched, and detracts from the reliability of the work as a whole. The chapters on pyrometry and metallography are practically new and well done. The chapter on fuels has been much enlarged on the descriptive side, but on the side of calorific powers and calorific intensities it is still as wofully deficient as in former editions, viz., at least 20 years antique. The chapter on thermochemistry of metallurgical reactions has been improved a little, yet it needed a great deal more.

Altogether, the book is now a mixture of good, fairly good, and pretty bad; brought up to date in spots and back in the last century in others. We confess to being disappointed with the general effect; we had hoped that the revision would be altogether thorough, satisfactory and up-to-date throughout Perhaps we expected the impossible; at any rate, we certainly hoped over-much.

* * *

Testing for Metallurgical Processes. By **James A. Barr.** 216 pages. 22 illus. Price, $2. San Francisco: Mining & Scientific Press.

A commendable book for use in research and metallurgical laboratories. Very practical directions are given for the use of the instruments and appliances useful in metallurgical testing and the handling of the data obtained, so as by calculation to reach further conclusions and results, is clearly set forth. In the calibration of pyrometers the variation of the boiling points of water and sulphur with barometric pressure should have been given, and the melting point of copper should be specified as 1084° in a *reducing* atmosphere. The treatment of the subject of fluxing and slags is particularly good.

The book is recommended to students of metallurgy and to those who, by practical work, are trying to master for themselves the principles of metallurgical processes.

* * *

Tables for the Determination of Common Rocks. By **Oliver Bowles,** M. A., instructor in geology and minerology, University of Minnesota. 16mo., 64 pages. Price, $0.50 net. New York: D. Van Nostrand & Company.

The author says: "Rocks can never be learned from a text-book alone." We will go further than this and say: An abridged text-book such as this, is of practically no assistance under any circumstances in *learning* rocks. If you know the rocks, the book can serve simply as an *aide de memoire,* if you do not know the rocks, it is of no assistance whatever in helping to know them. The compilation is occasionally somewhat careless; for instance, the red, bluish and green plagioclase feldspars are left out of the list of rock-forming minerals.

The publishers should be scored for putting 84 pages of advertisements in a book having only 64 pages of text. It is a rank imposition on the man who is supposed to carry the book around with him.

* * *

Elementary Crystallography. By **W. S. Bayley.** 239 pages, 311 illus. Price, $2. New York: McGraw-Hill Book Company.

This book, Part I of a general system of mineralogy, and published as a separate volume, is designed to introduce the reader to the subject of crystallography. Only a slight knowledge of physics, chemistry, and mathematics is required to comprehend the concise and generally well-written and profusely illustrated text.

But to the reviewer the work lacks one essential element in the presentation that unfortunately is missing from all modern books on this subject—that is, an adaptation of the usual academic presentation to practical use. Ninety-nine per cent of all students of crystallography intend to apply it only to the recognition of mineral forms, and there drop it. The presentation of the crystal forms as geometrically symmetrical solids does not assist in this practical application of the sub-

ject, and the use of the ordinary wooden models but little more. What is wanted is a work that gives more detail to the description of the non-geometrically symmetrical crystal forms as they actually exist—that is, the presentation to the student of the forms having missing planes and the crystals of unequal growth, in fact, of the crystal forms as they are usually found and not as the crystallographer thinks they should be. For instance, the author defines a cube as a solid consisting of six planes, each a *square* at right angles to the isometric axes. The actual number of square planes on the average crystallographic cube is rarely more than two, and later on illustrations are given of two cubes with but two square planes each. The author is not consistent in this regard.

The use of Naumann's symbols for indicating crystallographic forms is rather obsolete, all authoritative works on mineralogy and chemical crystallography using Miller's indices. Similarly, the terms used to name the various forms should have their more common synonyms given, as found in such standard works as Dana.

The chapters of Physical and Chemical Crystallography are welcome additions to an elementary book and are in keeping with the general plan of the more extensive chapters on Geometric Crystallography. The introduction of these chapters alone makes the work a notable improvement over any previous book of the same grade.

* * *

The M. C. C. Periodic Chart of the Elements, with introduction. Price, folded and bound, 8s. 6d.; roll form, 12s. 6d. London, W. C. (333 Grays Inn Road): The Metals Composition Company.

This is a chart 40 in. wide by 34 in. high, having on it 120 spaces 2 in. x 4 in., representing a periodic arrangement of the elements. For each element is printed in the space its principal chemical and physical properties. In the vacant spaces are printed condensed physical and chemical tables and other miscellaneous information. Ten groups of elements have been used, instead of the familiar eight, the distinction arising from making three groups of Mendeleeff's group VIII. The chart is clear, and will be of considerable interest in the chemical lecture-room or laboratory, hung up on the wall for casual reference.

Part I of the instruction is a straightforward and sensible explanation of the principles of the periodic classification of the elements; part II is a dissertation on "the periodic law and its relation to speculative thought." The latter part is distinctly out of place in connection with the educational aspects of the chart, for it deals with such philosophical abstractions as "Life: a Mystery" and "Universal Ultimate Relationship." It shows a wide range of solid reading and thinking on the part of the author, but the proper place for its presentation would be to read it before some philosophical society and see what would be the judgment of his peers upon his speculations.

Altogether, and considering by whom it is printed, it has such a tincture of the commercial and the alchemistical as "queers" it when we try to take it most seriously. Nevertheless, the chart may be found quite convenient and useful.

* * *

The Modern Manufacture of Portland Cement. By **Percy C. H. West.** Vol. 1, Machinery and Kilns. Price, $4 net. New York: The McGraw-Hill Book Company.

The manufacture of cement has increased in the past decade by leaps and bounds in this country, and Germany especially. The evolution from the old hand-raked reverberatory to the shaft kiln has been concomitant with the increasing uses of cement in the building trades. While text-books on the testing of cement and on concrete construction, especially reinforced concrete, are numerous enough in recent technical English publications, yet the practical and commercial side of making cement has been neglected. This we opine is due to the rapid flux which cement practice has undergone in the past ten years.

Accordingly it will be seen that this book of Mr. West is written at a moment when changes have become less and less noticeable and a standard of practice, or rather an approximate to a standard, has been obtained. Though the book in our opinion does not give due credit to the work of earlier American pioneers and to recent American pioneers as Edison, with the long rotary kiln and his pneumatic system of separation of the fines, yet the very fact that it does give prominence to the heavy and substantial machinery of the English cement works—a substantiality that often approaches mechanical corpulency—makes the book valuable to the American engineer, whose tendency is rather to a maximum of efficiency for a short period.

In the succeeding volume we hope that the application of physical chemistry to the practical problems of cement making and the question of thermal efficiency of the various systems, for each of which this journal is ever a constant advocate, will be discussed in the fullest proportions.

* * *

Outlines of Chemistry. A text-book for college students. By **Louis Kahlenberg,** Ph. D., professor of chemistry in the University of Wisconsin. 548 pages, 155 illus. Price, $2.60 net. New York: The Macmillan Company.

Why do our students spend so many years studying a foreign language and know so little of it at the end? Because they are troubled with the rules of good grammar before they can read or speak the language. Why do so many students know so little about real chemistry, although they "have been through" a chemical course? Because they have been pestered with theories and speculations before they knew the simple fundamental facts of chemistry, and the result has been to kill their proper interest in chemistry as an *observational* science.

We think Professor Kahlenberg exactly right when he puts only experiment and observation into the first five chapters of his splendid book, excluding from them even all reference to any theories whatever. This is the way to provide the student with facts to think about, to stimulate in a healthy way his curiosity, which gradually develops, as to what explanations there can be for these facts. This is the normal and natural way to acquire knowledge of theory; not to have it pumped into the observer just as he is acquiring his fundamental facts or even before he gets them. It is only after a student has tried to imagine for himself some rational explanations for the facts he has observed that he can properly understand, appreciate, and assimilate theory presented to him by others.

With these principles to start with and building thereon a system of instruction in elementary chemistry which is marked throughout by its appeal to the student's observing and reasoning powers (as to oppose a system of cramming into him a flood of facts mixed with cut-and-dried theoretical explanations) the experienced author and teacher has given us a wonderful book. There is only one other which we can compare it with—Mendeleeff's classical "Principles of Chemistry," and it is better than that for the purpose it is intended to serve—a one-year course in chemistry. Mendeleeff's work would need at least two years to work through, and is not so well arranged for the very beginner in the science.

The long experience of Professor Kahlenberg shows itself on almost every page of his book. It is clear, tactful, even sympathetic in places, in dealing with matters not easy to comprehend. The facts are carefully systematized, theories are skilfully introduced and presented in a larger-minded way, and the total result is a "work of art" in instruction in elementary chemistry which has probably no equal and certainly no superior.

We predict that our American educators will quickly appreciate the merits of this book and that it will have a phenomenal success.

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The New Chemists Building

Elsewhere in this issue the program will be found of the exercises with which the new Chemists Building at 50-54 East Forty-first Street, New York, will be opened on March 17 and 18. For chemistry in America history has been made in the old cozy clubhouse in West Fifty-fifth Street and with it are connected dear reminiscences of many a glorious achievement and many a happy hour enjoyed in good fellowship. Now the old spirit is to be transplanted into the new big and magnificent home. This is the meaning of the opening exercises. The new Chemists Building is a grand monument representative of the united democracy of American chemists. A graceful and thoroughly practical twentieth-century structure, it has been erected, without outside help, to represent the ideals of American chemists. Our hopes and wishes are comprised in the old Saxon miner's greeting: "Glückauf."

Continuous Copper Converting.

Some years ago we pointed out in these columns the essential difference between a continuous and an intermittent process. This was to the effect that continuous operation was advantageous when the end desired was a large tonnage eliminating impurities cheaply and that an intermittent operation was advantageous when the end desired was a refined or semi-refined product of a definite nature. In short, a continuous process makes for quantity of product and an intermittent process makes for quality of product. For the complete elaboration of this principle the reader may be referred to our Volume III, page 90.

Copper converting is peculiar because it is at once a roughing metallurgical action and a finishing metallurgical action. As it unites two functions in an operative manner it is efficient. But at the same time it has the inherent disadvantages of an attempt to do two things at one time. These peculiar disadvantages are the cost of lining the converter, the amount of power used for furnishing the blast and the danger of loss of values by rapid driving. The lining is before first blow from 20 in. to 24 in. thickness, and, as everyone versed in the metallurgy of copper knows, is made of a plastic silicious material. The lining furnishes the silica for the slag produced by the oxidation of the iron in the matte blown up to blister copper. It is certainly crude to use the lining of the converter as a flux and seems as foolish as nature would if we lived on the inside of our stomach. But like many other processes crude from a theoretical standpoint, it nevertheless, is a practical process, for it works.

Various attempts have been made to use a lining that is basic and hence refractory to the ferruginous slag made. Professor Baggaley made some exhaustive large-scale tests with a basic-lined converter about five years ago at Butte, Mont. Prior to that time Mr. George Westinghouse had conducted a long series of experiments at Pittsburgh and later at Ely, Vt.

For commercial reasons alone both these attempts were never followed out. Indeed, in the several fat years of the metal industry, from 1900 to 1907, improvements were not needed as much as they are now in the present times of diminishing returns.

During the past year large-scale commercial demonstration of a basic-lined copper converter has been made by Messrs. Smith and Pierce at Ely, Nev. Also that enterprising concern, the International Nickel Company, has started work at Copper Cliff, Ontario, on a basic bessemer plant for working up the 30 per cent copper-nickel matte to an 80 per cent matte suitable for smelting by the Orford process. In both these attempts the silica is thrown on the slag continuously, but the tapping of the slag and the blowing are performed intermittently.

It might be argued that the logical process is rather to make the entire smelting process continuous and to do the final refining and the production of the copper anodes in an electric furnace, and it is interesting to give briefly the argument in support of this view. The operation of blowing up a 50 per cent matte to blister copper produces an active scorifying slag that will dissolve to some extent either a magnesite or a chromite brick or tamped basic lining. Oxide of iron, especially when a small percentage of iron sulphide is dissolved with it, is about as corrosive material at high temperatures as is well known. Indeed, oxide of iron can play the rôle of an acid with the formation of ferrates. The proper way, of course, is to use a stationary furnace or converter and water-cool the shell. From a theoretical standpoint this is ideal. Like pyritic smelting and "blast roasting" of sulphide ores the heat balance sheet shows that when once the operation is started at a good rate there is a fair excess of heat generated over heat lost by radiation so that the right conditions can be held uniformly. If this is possible in an intermittent operation there should be no reason why it should not be attained in the right form of a continuous apparatus.

Of course, there is a large proportion of slag—on the average 50 per cent of the original charge—that must be handled. Naturally this is a difficult proposition and regard must be paid to this difficulty. But the great success of the Dwight-Lloyd roasting and sintering apparatus is due to the continuous nature of the process which is conducted in it. It is naturally desirable that continuous copper-converting be done in large units. This would give the operation a sufficient excess of heat so that conditions could not change easily. It is, of course, patent that some means must be used to bring the oxygen of the blast continuously in contact with a large surface of matte. The difficulty and painful slowness of making blister copper out of "white metal" in the old-fashioned reverberatory furnace lay in the fact that the slag protected the matte from the oxygen. Possibly the old Nichols and James reaction of roasted copper-matte on copper-matte with the formation of blister-copper would be applicable. For the final finishing reaction and the removal of the gases nothing can be better or cheaper than the arc-type of electric furnace. The experience in the metallurgy of steel proves this.

We are not indulging at all in prophecy, but only attempting to judge the future from the past. Of late we have seen the great increase in the use of continuous blast-sintering machines in roasting sulphide ores and the extending use of the reverberatory smelting furnace for fine concentrates and flue-dust. Now we see the basic-lined converter. It is at least not illogical to expect the continuous converter in the not far distant future.

Some Interesting Rail Production Statistics.

Statistics of rail production in the United States in 1910, as just presented by the American Iron and Steel Association, show that the year was the next best in the history of the rail trade, the 3,634,029 gross tons produced falling short of the record year, 1906, by 343,858 tons and exceeding by 375 tons the production in 1907, which had stood as the second best year.

Such wild forecasts are sometimes made of the prospects of rail demand, and such curious views seem to prevail in many quarters as to the uses to which our rail production is put, that a cursory review of the general position may be profitable. There was in existence in the United States at the beginning of 1910 in the neighborhood of 400,000 miles of track, laid in steel rails. There was about 350,000 miles of steam railroad track, the remaining 50,000 miles of our estimate covering electric and other public roads using steel rails and industrial tracks. The tonnage of rails in this track can only be approximated, but was probably not far from 50,000,000 tons, which would be sufficient to lay 400,000 miles of track with 80-lb. rails. Up to that time there had been produced in the United States about 60,000,000 tons of steel rails, of which about 2,750,000 tons were by the open-hearth process. In the early years there were heavy imports of steel rails, while in late years there have been heavy exports, and these balance roughly. The deficit of some 10,000,000 tons represents, very roughly, the tonnage which has worn out. The total tonnage produced includes some material which passed through the mill twice, having worn out in service and being rerolled to pass through service again.

It may be of interest to observe, *en passant*, that there was produced in the United States a total of a little more than 10,000,000 tons of iron rails, and a small portion of that tonnage survives, there being at the beginning of 1910 about 8800 miles of steam-railroad track still laid with iron rails, involving, with the iron rails surviving in mines and other industrial plant, well in excess of 500,000 tons.

The fact that the tonnage of steel rails worn out in the United States has been in the neighborhood of only about 10,000,000 tons should assist in dispelling the curious view which is evidently entertained in some quarters that the chief use to which our rail production is put is the replacing of rails which have worn out in regular service. That is not the case. Even of the rails which have become unfit for service a large part has not yielded under proper usage, but has required replacement because too light for the much heavier service now involved. Had it been the case that the rail production of 10, 15 and 20 years ago was required to make up for the wearing out of rails, the present demand would be enormous, for the freight ton-mileage of the country—the only statistical measure we have of the service put upon rails—has approximately doubled each decade.

While there are no precise data as to the mileage of new

track laid in 1910 data are available which support an estimate of approximately 10,000 miles, which if laid in rails averaging 85 lb. per yard would involve about 1,335,000 tons. Exports amounted to 353,180 tons, while of light and standard rails used for industrial purposes the quantity may be estimated at about 250,000 tons. This would leave 1,696,000 tons for all other purposes, the summary being as follows:

	Tons.
10,000 miles steam track	1,335,000
Industrial uses	250,000
Exports	353,180
All other	1,696,000
Total production	3,634,029

The balance found by difference, 1,696,000 tons, must cover rails used for trolley and other transportation lines outside of steam roads, probably about one-third of the total, the balance being the tonnage used by the steam roads for replacement purposes. The major part of this is now ordinary replacement, but a portion represents the replacement of light sections which cannot stand up under the heavy duty imposed by high speeds and heavy cars and locomotives.

A few of the railroad systems present statistics of the tonnage of rails used for replacement purposes. By analyzing these statistics it may be shown that, very roughly, a ton of rails is worn out for every 250,000 ton-miles of freight traffic. The factor is only an approximation, but has value when there is practically no other indication. The total freight ton-mileage of the country in 1907 was 233,137,000 and in 1909 it was 227,199,000, so that a rough-and-ready rule is that when rails are of section sufficient to escape abuse, and meet only legitimate wear, the steam roads of the country are wearing out about 1,000,000 tons a year.

The study of rail statistics merely for the purpose of ascertaining where the rails go can have only an academic interest. We have made this little study for a more serious purpose. In the past rails have been used chiefly for laying new track; in the future they will be used more and more for replacing other rails which have worn out. The period of greatest railroad building is past; as far back as 1887 occured the greatest building of new road in a year, and while there remains a great deal of double-tracking to be done the laying of new track cannot be expected to increase much, if at all. The tendency to wear rails out, however, increases steadily. As already observed, there has been, approximately, a doubling of the ton-mileage of freight movement every decade and while this rate of increase may not be continued, there is certain to be increase at some rapid rate.

A rapid increase in freight movement and a slow increase in track mileage obviously involve an increase in the density of traffic on given stretches of track, and we have merely expressed, in terms of the entire railroad system, that which is common knowledge as to individual railroads. As the traffic increases and the rails are subject to greater and greater annual wear, the annual sinking fund necessary to replace the rail increases. It would be a simple matter to write an algebraic formula to illustrate the principle we wish to emphasize, but this form of expression is not necessary. It need merely be remarked that when the wear on rails was light, and they accordingly lasted a long time, it did not profit railroads to pay a given premium to obtain a more lasting rail, because the premium would increase the annual interest charge, while with a long life even the poorer rail would require but a small sinking fund to provide for its ultimate replacement. As the wear upon rails becomes greater and their prospective life is shortened, the annual sinking fund increases and it becomes more and more desirable to use a better rail, gaining more in decreasing the annual sinking fund for ultimate replacement than is lost by increasing the interest charge.

Thus, while a few years ago the railroads were interested merely in getting a fairly good and a safe rail, now those with the greatest density of traffic are anxious to obtain a very durable rail and are willing to pay a premium in order to secure it. We stand at the threshold of a great development in the manufacture of special steel rails. Only for 1909 and 1910 are there statistics of production of special rails. These statistics are as follows. They are quoted as given, though it is not clear why a distinction should be made between titanium and ferrotitanium steel rails:

1909.	Gross Tons.
Titanium	35,945
Nickel-chrome	12,287
Nickel-steel and electric	1,464
Manganese	1,028
Total	50,724

1910.	Gross Tons.
Titanium	195,408
Electric	4,210
Ferrotitanium	532
Manganese	390
Nickel, chrome and vanadium	81
Total	200,621

Of the 1909 tonnage of special steel 35,699 tons, or 70 per cent, was made from Bessemer steel, while in 1910 the proportion was 87 per cent. We are not prepared to draw positive conclusions from the statistics. They are suggestive, but perhaps of negative rather than positive conclusions. The high percentages of Bessemer rails treated may mean that open-hearth rails are regarded as "good enough"; at any rate they show that as to the refining action of titanium (the "titanium rail" is not a titanium alloy steel, but a steel purified by titanium, the titanium compounds formed passing into the slag) the Bessemer metal can be treated satisfactorily. The practical disappearance from 1909 to 1910 of nickel-chrome indicates a loss of interest in this alloy. The electric steel rail showed a large percentage increase, and 4210 tons of electric steel rails produced in 1910 furnishes at least a foundation for observation, being enough to lay more than 30 miles of track. The "titanium rail" shows a remarkable growth, from 35,945 tons in 1909 to 196,000 tons in 1910, and it is regrettable that the statistics do not indicate precisely the nature and extent of the treatment to which the tonnage was subjected. It was not until 1907 that as large a tonnage of open-hearth steel rails was produced, yet the open-hearth steel rail was regarded as a factor and was provoking a great deal of discussion.

Program of the Opening of the New Chemists Building in New York City

The celebration of the opening of the new Chemists' Building, 50 to 54 East Forty-first Street, New York City, will take place on Friday, March 17, and Saturday, March 18.

On Friday, March 17, there will be a reception at the new club building from 3 to 6 p. m.

The opening exercises in the auditorium under the auspices of the Chemists' Building Company will take place from 3:30 to 5 p. m.

In the evening a meeting will be held in the auditorium under the auspices of the New York Section of the American Chemical Society, which will begin at 8:30. After an introductory address by the chairman, Dr. Charles Baskerville, papers will be presented by Prof. R. B. Moore, of Butler College, Indianapolis, on the rare gases of the atmosphere and by Dr. Jacques Loeb, of the Rockefeller Institute for Medical Research, on the characteristics of living matter from the physico-chemical point of view.

On Saturday, March 18, a session will be held in the auditorium from 10:30 to 12:30 under the auspices of the New York Section of the American Electrochemical Society. After an opening address by the chairman, Mr. E. A. Sperry, papers will be presented by Dr. W. R. Whitney on mental catalysis and by Dr. E. F. Smith on recent developments of electro-analysis.

In the afternoon a session will be held from 2 to 4 p. m. under the auspices of the New York Section of the Society of Chemical Industry.

After an introductory address by the chairman, Dr. Charles F. McKenna, papers will be presented by Dr. Charles F. Chandler on the history of chemical industry in New York City and by Dr. W. P. Mason, of Troy, N. Y., on the contributions of chemistry to sanitation.

In the evening a subscription banquet will be held at the clubhouse under the auspices of the Chemists' Club.

On Sunday, March 19, there will be in the afternoon, beginning at 4 p. m., a concert by the Olive Meade Quartet.

Owing to the limited accommodations admission to all functions will be strictly by cards and tickets. Ladies are invited to attend all the functions with the exception of the banquet.

American Electrochemical Society

The nineteenth general meeting of the American Electrochemical Society will be held in New York City on April 6, 7 and 8 (Thursday to Saturday).

A special feature of this meeting will be the fact that it will be the first convention of a national chemical society to be held in the new beautiful Chemists' Building in New York.

The full program of papers, excursions and social functions will be printed in our next issue.

International Congress of Applied Chemistry of 1912.

The organization of the Eighth International Congress of Applied Chemistry, to be held next year in this country, has now been completed and will shortly be made public. The dates of the congress have been fixed as follows:

The opening meeting will take place at Washington, D. C., on Sept. 4, 1912 (Wednesday). The other meetings, both business and scientific, will be held in New York City, beginning Friday, Sept. 6, 1912, and ending Friday, Sept. 13, 1912.

The President of the United States will be the patron of the congress. Dr. Wm. H. Nichols is the president, Dr. E. W. Morley the honorary president, and Dr. B. C. Hesse, 25 Broad Street, New York City, is the secretary of the congress.

American Institute of Chemical Engineers

The third semi-annual meeting of the American Institute of Chemical Engineers will be held at Chicago, Ill., from June 21 to 24 (Wednesday to Saturday).

Arrangements will be made to visit a number of the large technical plants in the vicinity. The committees on chemical engineering education and on standardization of boiler tests will have important reports to present.

The Western Metallurgical Field.

Leadville's Celebration.

Probably no more unique celebration was ever held than that which marked the so-called rejuvenation of Leadville by the discovery of zinc carbonate and silicate. The event was combined with the annual celebration of Robert Burns' birthday, although there is no direct connection between the Scotch poet and smithsonite or calamine other than that the Leadville miners are interested in both.

The novel feature of the celebration consisted in a noonday dinner served to about 300 guests in a stope of the Wolftone mine, 850 ft. underground. The Western Mining Company, Mr. S. D. Nicholson, president, was the host of the occasion. The guests were taken down the shaft and through lighted drifts to the banquet-room. The latter was a stope from which about 3000 tons of 33 per cent zinc carbonate had been extracted and shipped to smelters. It was brilliantly lighted and decorated with flags and national colors, and the whole effect was such as to dispel the idea of being underground. In the evening a more elaborate and pretentious banquet was served at the Elks' Club, at which the significance of the recent discoveries were considered.

The largest problem which lies ahead of Leadville, and one of the most importance in connection with the zinc discoveries, is the future treatment of low-grade ores carrying from 10 per cent to 20 per cent zinc. There is a vast quantity of such ore which, at present, can not be handled. The lowest grade material yet shipped at a profit is 20 per cent zinc, and there awaits professional fame and ample reward for the metallurgist who shall devise successful means of treating the lower-grade ores. It is a significant fact that coincident with the discoveries of Leadville's new zinc resources many eminent metallurgists are working on the problem of cheap and more efficient zinc reduction, and there can be no doubt that some process will be devised for local application which will make profit for both miner and metallurgist. It is also significant that Leadville mining men are proceeding at once in the search for suitable processes for low-grade ores without postponing such action to the day of reckoning which inevitably comes when the high-grade and easily treated ores have been exhausted. Although no definite announcement can be made, it is known that already negotiations are proceeding between the mine owners and metallurgists who believe that they can solve the problem.

Some Interesting Company Reports.

The Utah Copper Company has issued its report for the fourth quarter of 1910. The work of remodeling the Arthur plant to conform to the system adopted in the Magna plant is progressing satisfactorily, and the first remodeling unit will be placed in commission at once. The production of copper for the quarter was as follows:

October	7,582,219 lb.
November	7,468,915 lb.
December	7,137,585 lb.
Total	22,188,719 lb.
Average monthly production	7,396,239 lb.
Total production for 1910	89,019,511 lb.

The average net cost of copper produced for the quarter, after smelter deductions were made, was 8.227 cents per pound, as compared with 8.25 cents for the preceding quarter. The average grade of the ore was 1.56 per cent copper.

Of still more interest from the point of low cost of production of copper is the report of the Nevada Consolidated Copper Company for the fourth quarter of 1910. The output of copper for that quarter was but 15,698,595 lb., a decrease of about 15 per cent compared with the preceding quarter. The ore treated averaged 2 per cent copper, and the cost per pound of metal produced, after writing off depreciation on the Steptoe plant, is 6.7 cents. The total earnings of the company for the quarter were $1,046,527.

The Goldfield Consolidated Mines Company's annual report for the fiscal year ending Oct. 31, 1910, contains some interesting facts bearing on gold recovery. The mill ore treated amounted to 265,351 tons, of an average value of $36.99. The highest grade of mill ore was treated in April, 1910, when the value was $51.06, and the lowest in February, 1910, the value being $30.86. The average percentage extraction for the year was 93.98; the highest percentage was 95.78 in April and the lowest was 92.41 per cent in October, 1910.

Comparison with last year's report will show that there has been no decrease in operating costs coincident with increased tonnage at the 100-stamp mill. This is accounted for wholly by increased cyanide consumption, caused by the constantly diminishing proportion of oxidized ores and increase of sulphides from the lower levels. Leaving this factor out of consideration the plant has worked exceedingly well. In crushing efficiency, cost of operation and recovery of gold it has fully met expectations.

The capacity of the 100-stamp mill was increased during the year by the addition of six 6-ft. Trent Chilean mills and 24 No. 3 Deister slime tables. The installation has increased capacity from 650 tons to 850 tons per day and has been highly successful. If the cost for chemicals were the same this year as last the report of operations would show a reduction of 35 cents per ton for this year as compared with last. A new refinery is being built, located at some distance from the plant. It is of reinforced concrete construction and will cost about $25,000. It will effect a saving of $5,000 per month. A new and complete fire protection system has been installed since the fire last year. This cost approximately $50,000, but will prevent the recurrence of such accidents as that of last year, which cost the company $156,446.

The First Cyanamide Process Installation.

It is rather complimentary to the soundness of the principles of the Clancy cyanide process that within a few months after they were publicly announced a Colorado company decided to erect a mill embodying them. The Ajax Gold Mining Company, with mines at Victor, Cripple Creek mining district, Colorado, is erecting a 100-ton unit for the treatment of raw sulpho-tellurides by the Clancy process. The ore will be crushed dry, ground in tube mill with cyanide solution, agitated in tanks with the addition of potassium iodide and the application of the electric current; then, with the addition of calcium cyanamide, the solution will be regenerated in the same agitation tanks. The pulp will be filtered by the Moore system and the solution returned for further use.

Preliminary experiments in a laboratory way have thrown more light on the details of commercial operation. For one thing, it has been found that no such current densities as first proposed will be necessary for either extraction or regeneration. The extraction occurs with currents as low as 2 amp to 3 amp, and some experiments have indicated that still lower currents will produce the result desired. The regeneration of the solution also has been accomplished with similar currents and the same solution used for eight to ten weeks has shown little loss of strength. The proper substance to be used as anode is one that has not been settled, although the choice probably rests between fused ferric oxide and Acheson graphite.

Construction of the plant is now under way and the first operation should take place early in the coming summer. The result will be watched for eagerly, as its success will mean much to the Western field. Present laboratory indications are very favorable for success in the commercial application.

The Government-Anaconda Smoke Suit.

If reports are to be credited, and they seem reliable, the federal government is to withdraw its suit against the Anaconda Copper Mining Company in which the plaintiff sought to compel the smelting company to abate the smoke nuisance at Anaconda, Mont. By such an action the government will be credited with acting sensibly in a suit which, in the opinion of many conversant with the facts and conditions, never could have been brought to a successful issue. Personal inspection on the part of government officials has evidently shown them that the Anaconda company has at all times acted fairly and that it proceeded wisely in the first instance by selecting a good place for the erection of the smelter and by making every reasonable provision for getting rid of fume and smoke to the best advantage of all concerned. A change of heart sometimes can be produced by getting the opponent's viewpoint and this seems to have been accomplished in the present instance.

Utah.

The milling interests of some sections of Utah are exercised over the recent introduction into the State Legislature of a bill making it a misdemeanor to discharge mill tailings into a stream. The bill is in the interests of fish preservation, but the mill operators see in it a sure means of closing their plants if it should carry. Inasmuch as but few streams of the State are polluted by mill tailings, and many others are left untouched, it is probable that a compromise can be effected and the matter settled without resort to legislation. The spirit of live and let live, and the greatest good for the greatest number, must prevail if all industries are to thrive.

The Iron and Steel Market,

Bookings of orders for finished steel products during February averaged between 60 per cent and 70 per cent of the total available capacity, being in excess of capacity in wire products, heavy in pipe, less heavy in bars, tin plates and sheets, and relatively light in plates, structural shapes and rails. During January there was a large improvement in the rate of booking and the rate of the closing days of January was maintained through February without important change.

Production has increased steadily through February on account of the increased rate of booking. At the close of the month the steel industry as a whole is running at between 65 per cent and 70 per cent of full capacity, against a trifle less than 50 per cent in the closing week of December.

The increased rate of buying of finished products is due chiefly to the fact that on account of uncertainties as to price and the desire to withhold business until after the annual inventory a considerable volume of business backed up and began to be released after the middle of January. There does not seem to have been any particular change in fundamental conditions, and an improvement in general business is therefore expected to effect an increase in orders from the rate prevailing in February.

The United States Steel Corporation on Feb. 10 announced its "unfilled orders" at the clase of business Jan. 31 at 3,110,919 tons. This represented a gain of 436,162 tons during January, the first gain for a year. The total loss during 1910 was no less than 3,252,274 tons, the high point in unfilled orders since 1907 having been the 5,927,031 tons reported for Dec. 31, 1909. The capacity of the steel corporation, in terms of the products it sells, is commonly taken at about 1,200,000 tons monthly, of

which the January gain of 436,162 tons is 36 per cent, and its shipments in January may be estimated roughly at 40 per cent of capacity, with bookings at 75 per cent. The corporation's January bookings were particularly heavy by reason of the formal entering of a number of rail orders, which had practically been placed some time before. February has not witnessed such heavy booking of rail orders, and while in most products the month's bookings exceed those of January, the grand total may perhaps be found not as large.

The trend of finished steel prices has been upward, although not markedly so, the condition being important chiefly because it presents so strong a contrast to that supposed to exist in December, when there was no expectation of price changes, except in the downward direction. The wire advance of Jan. 23 and the tin-plate advance of Feb. 4 have served to show that finished steel prices can advance and while there is no general advancing tendency it has become patent that the price level is well held and that upon occasion advances can be effected.

The price of tin plates was advanced 10 cents per base box at a conference of producers held in Pittsburgh Feb. 3, the advance becoming effective at the opening of business Feb. 4. The new price basis is $3.70 per base box for 100-lb. coke plates, with corresponding advances for ordinary ternes and heavy-coated ternes and bright plates. The advance was made solely to cover the increased cost of manufacture due to the advance in pig tin effected by the foreign syndicate. There is no means of forecasting the future of the pig tin market, but it may be taken as settled that should the price advance to 48 cents or 50 cents a further advance of 10 cents or 15 cents per box would be made in tin plate. When the former price of $3.60 was established on tin plate, Nov. 12, 1909, pig tin was 30.50 cents and sheet bars $28.50. When the advance to $3.70 was made, Feb. 3, 1910, pig tin was 41.40 cents and sheet bars $24.

All the prospects of the iron and steel industry are distinctly favorable. There has been improvement since the first of the year, and with occasional backsets improvement is expected through the remainder of this year. The causes of the continued decline in prices and activity in 1910 are readily perceived, first, that the recovery of 1909 was overdone, and, second, that during 1910 general trade conditions became less favorable. In its adjustment the iron trade swung too far and at the beginning of this year was in line for a return to normal.

Pig Iron.

The pig-iron market experienced a substantial, though slight, recovery in February. Prices had reached a level giving very small profits to most furnaces and involving to the Mahoning and Shenango Valley furnaces an actual loss. The fractional gains in February were important as showing a change in the price trend rather than as making a material difference in the general level. The International Harvester Company purchased a total of about 100,000 tons of foundry iron, divided between Northern and Southern, and one or two other Central Western consumers were large buyers. Southern iron stiffened so that concessions from the $11, Birmingham, price on high-phosphorus irons disappeared and on standard Alabama grades the $11 price became applicable only to deliveries this side July 1, later deliveries being held at 50 cents advance. The Buffalo market, after making prices of $13.75, Buffalo, and perhaps lower, for No. 2 X, in competition with eastern Pennsylvania furnaces for New England business, advanced to $14 and higher. Basic iron in eastern Pennsylvania advanced to $14.50, delivered, after some large sales at $14.25, other grades being unchanged in the East. In the Pittsburgh Valley market Bessemer remained at $15, valley; $15.90, Pittsburgh, as it has been since August, there being a number of sales of small tonnages, including one lot of 1000 tons for prompt delivery and another of 1250 tons for delivery 250 tons per month. The foundry-iron market firmed up, all prices at $13.75, furnace, being withdrawn and leaving the market squarely on the basis of $14 at furnace, equal to $14.75 to $14.90, Pittsburgh, depending on location. A few small lots have been sold at $14.25, furnace. Basic iron has furnished by far the most interest. Owing to the disappearance of one merchant furnace seller through the sale of the Midland properties to the Crucible Steel Company one or two other furnace interests marked up their price from $13.25, valley, at which sales were made early in February, to $14, furnace, which during the second half of the month was the lowest price made by furnaces. Meanwhile sales were made at advancing prices by middle interests, the course being pretty steadily that after one or two sales at a given price a slightly higher price would be demanded on the next inquiry. A sale of 10,000 tons was made at $13.50, then sales of 5000 tons or more were made at $13.60, this being followed by a sale of 2000 tons at $13.65 and this in turn by several odd lots at $13.70 and $13.75, leaving the market in the closing days of the month on the basis of $14 asked by furnaces, and last market sales at about $13.75, equivalent to $14.65 to $14.90, Pittsburgh.

Steel.

The market has been very steady at $23 for billets and $24 for sheet bars, f.o.b. Pittsburgh or Youngstown, for either Bessemer or open-hearth, and $29, Pittsburgh, for rods. A decided disinclination to sell open-hearth sheet bars at $24 developed about the middle of the month. One interest, which had sold a special quality of open-hearth bars, suitable for deep stamping, at $24, obtained $25 on a lot of upwards of 2000 tons. Specifications on contracts for billets and sheet bars have improved and one branch of the consuming trade, the tin-plate mills, have been specifying practically their maximum tonnages on regular term contracts.

Finished Steel.

Finished steel prices are firmly held all along the line, as has been the case since early in December, and are as follows, f.o.b. Pittsburgh, except where otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3 in. to 15 in., inclusive, zees and angles 2 x 3 and larger.

Steel bars, 1.40 cents, base; iron bars, 1.40 cents, Pittsburgh; 1.30 cents, Chicago.

Wire nails, $1.75, base, per keg; plain wire, 1.55 cents, base; galvanized barb wire, 2.05 cents; painted barb wire, 1.75 cents.

Black sheets, 28 gage, 2.20 cents; galvanized, 3.20 cents; blue annealed, 10 gauge, 1.65 cents; painted corrugated roofing, $1.55; galvanized, $2.75 per square.

Tin plates, $3.60 for 100-lb. cokes.

The Non-Ferrous Metal Market

Since our last report the metal market has been without feature except as to transactions in tin and copper. A fair volume of business continues to be transacted, but the uncertainties of the market confine purchases almost to immediate needs.

Copper.—The Copper Producers' Association report for January was made public on Feb. 8 and showed an increase in surplus. The production and imports for January were 115,676,591 lb. and although this figure showed a decrease of 7,642,628 lb. as compared with December the decrease in exports gave an ultimate surplus on Feb. 1 of 142,439,490 lb., which was an increase of 20,409,295 lb. This increase is not viewed unfavorably by a Western authority, who regards the increase as due to natural causes occurring at or near the end of the year when producers and refiners make a special effort to clean up accumulated stocks. The market has been unsettled and the agencies have effected business at a concession from the asking price. Lake copper is quoted at 12.5

cents to 12.75 cents, and electrolytic at 12.25 cents to 12.30 cents.

Lead.—The market has been quiet and without feature. The American Smelting & Refining Company has maintained a price of 4.50 cents, but other sellers have not reached this figure. New York quotation is 4.45 cents to 4.50 cents and St. Louis, 4.27½ cents to 4.30 cents.

Tin.—Since our last report tin has reached a high figure, selling at about 43 cents, but has dropped again quite as suddenly. The market has been spasmodic owing to the uncertainty of the intentions of those who control the foreign market. The last New York quotations are about 40 cents.

Spelter.—The market has been unsatisfactory, but a fair volume of business is reported and the prospect is for better conditions. The New York market quotes the metal at 5.42½ cents to 5.47½ cents; St. Louis at 5.27½ to 5.32½, which is lower than last month.

CORRESPONDENCE

Efficiencies of Electric Furnaces.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In the last volume of the *Transactions* of the American Electrochemical Society a statement was published, apparently with deliberation, which, if left unanswered, might seriously mislead those who accept it.

The statement is that the lower the temperature of the outside of furnace walls of different materials the less the heat loss through the walls, hence the higher the efficiency of the insulation of the walls. The thickness of the walls is assumed to be the same, and for the purpose of comparing different materials composing the walls of this given thickness (as proposed by that author) it is assumed, for discussing the present feature, that the nature of the outside covering is the same. In other words, all conditions are the same except the material of the walls, the loss through them and the outside temperature.

It is furthermore stated by that author, by way of explanation, that the higher the external temperature the less the heat gradient from the inside to the outside and, therefore, the less heat is being taken out of the furnace. This evidently contradicts the first statement. The explanation is correct when applied to any one particular furnace—that is, to any one particular wall material; but it would certainly be incorrect to say that, in comparing different materials with each other, a smaller heat gradient necessarily means less loss. The loss is dependent not only on the heat gradient, but also quite as much on the conductivity of the particular material. Hence the outside temperature of the furnace is absolutely no criterion of the losses through walls of different materials, although that author claims very specifically that it is, whatever else he may have meant by his other remarks, which are not very clear.

The total fallacy of this statement and its alleged explanation is best shown by an actual example. Let us take the same small, simple furnace for which the calculation of the loss was given by me on page 15 of your January issue. The interior is 6 in. x 6 in. x 6 in. The wall thickness is 4 in., and for simplicity let us omit the second layer of brick. Let the temperature of the inside be 1500° C. and on the outside of the wall it will then be 175° when the wall is made of silica brick, which is assumed to have a thermal resistivity of 47 thermal ohms in cubic inch units. The loss through the walls will then be 3½ kw, as was there shown.

Now, for testing the truth of the above mentioned statement, let us assume the walls of the same furnace to be made of copper. This, of course, is an exaggerated case, but the results apply exactly the same, only to a less degree, to any other material. The thermal resistivity of copper is about 0.1 in cubic inch units. And let it be assumed that the outside temperature now is half what it was before, hence 88° C. If the statement under discussion were correct the loss should now be considerably less because the heat gradient is much greater.

Calculations, however, made just as before show that the loss now is 1760 kw, or more than 500 times as great.

If the outside were allowed to get to a bright red heat, say to a temperature of 1000° C., nearly to the melting point of copper and therefore very much higher than the 175° for the brick, the loss will be 625 kw, or nearly 200 times as great, instead of being much less.

The above examples show that for either a higher or a lower outside temperature the loss may be enormously greater. By repeating the calculations for a very high resisting material it could similarly be shown that for both higher and somewhat lower outside temperatures the loss is less than in the original brick furnace. Hence the temperature alone can be no criterion whatsoever in such comparisons of different materials.

Hence the statement referred to is certainly incorrect; one would be committing very serious errors by relying on it. Different materials for walls, therefore, cannot be compared in this way, as that author had proposed. A smaller heat gradient is by no means an indication of a smaller flow of heat through the walls of different materials, but only for the same material. Such a statement should not have been left unchallenged at a meeting of the American Electrochemical Society. Too many incorrect statements have already been made concerning problems involving the flow of heat; as a matter of fact, such problems are generally extremely simple, when correctly understood.

Philadelphia, Pa. CARL HERING.

The Cyanamide Process in the Metallurgy of Gold.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—In your January, 1911, issue there appears an abstract of a paper presented before the Electrochemical Society, Dec. 16, 1910, on the "Clancy Process" of ore-treatment. In the paper referred to Mr. Clancy claims as novel the application of calcium cyanamide as a cheap cyanogen agent in the place of the more expensive sodium or potassium cyanide universally employed in the treatment of gold and silver ores by cyanidation.

I beg to take issue with Mr. Clancy as regards his claim to originality to employ calcium cyanamide in ore-treatment in place of the more expensive sodium or potassium cyanide as I can show priority in the application of this idea to the extraction of gold and silver from ores. And while Mr. Clancy may claim some novelty in adapting the use of calcium cyanamide to gold and silver extraction by means of electrolysis instead of the more direct method which I have proposed, viz: its transformation into calcium cyanide in a much more simple manner, by subjecting the cyanamide at a red heat to acetylene, water gas or any hydrocarbon gas rich in carbon. This can be simultaneously effected during the process of manufacturing the calcium cyanamide from calcium carbide.

I cannot understand how Mr. Clancy could possibly have presented calcium cyanamide as a cheap and novel substitute of his own invention in place of the more expensive KCy or Na Cy for use in the cyanide treatment of ores, when I already wrote up its application several years ago, first in: My United States patent 911,254, filed Sept. 4, 1907, and issued Feb. 2, 1909, from which I quote the following:

"My invention relates to the treatment of ores and tailing with ammonia-cyanide solutions, for the purpose of extracting their gold and silver content.

"My invention is applicable, not only to the treatment of ores and tailings containing gold and silver alone, but also to such ores and tailings as contain, in addition to gold and silver, such ammonia-soluble metals as copper, zinc, nickel and cobalt.

"The object of my invention is to effect economy in the process; and, to this end, my invention consists in the treatment of the ores and tailings with a solution of ammonia and alkaline earth-metal cyanide, as I shall now fully describe.

"In the treatment of ordinary gold and silver bearing ores and tailings by the cyanide process a cyanide solvent must be used which is not only stable in dilute solutions, such as 0.05 per cent to 0.50 per cent KCy, but must withstand exposure to the atmosphere as well, without decomposition. It must also not contain impurities, such as sulphide, carbides, etc., which would greatly depress the gold and silver extraction coefficient of the cyanide solvent. The cyanides ordinarily used, therefore, are the potassium and sodium salts, KCy and Na Cy.

"These are highly refined, snow-white and almost chemically pure. In this state they are sent to market for precious metal extraction purposes, and are, necessarily, very costly, from 20 to 25 cents per pound. In recent years, however, there has been great development in the cheap production of alkaline earth-metal cyanides, such as calcium cyanide, Ca $(CN)_2$ and Barium cyanide, Ba $(CN)_2$, by means of the electric furnace, from atmospheric nitrogen.

"I have found that these electric furnace products, even impure as they are, if they contain 80 per cent to 90 per cent of the theoretical quantity of cyanogen $(CN)_2$, are available for and may be adopted and used in ammonia solutions, as gold and silver solvents.

"In carrying out my invention, the crude calcium cyanide or barium cyanide is dissolved in ammonia solution of sufficient strength; the insoluble residue allowed to settle and the clear solution applied in the treatment of gold and silver bearing ores and tailings, not amenable to ordinary cyanide treatment, either on account of excessive cyanide consumption due to cyanicides, or on account of percentage of metals such as copper, nickel, zinc or cobalt, which render inert the gold or silver solvent powers of a dilute cyanide solution (KCy or NaCy) such as is commonly used, and cause an excessive loss of the cyanide.

"A working solution, according to my invention, may be made up as follows: Solution of ammonia, 1 per cent or less to 10 per cent (NH_3). Calcium cyanide or barium cyanide, 0.05 per cent to 1 per cent. If this solution be too alkaline, a salt of ammonia, such as the sulphate, may be added to counteract such excessive alkalinity.

"The treatment of the ores or tailings with this solution is carried out in any of the usual or known manners, as by leaching or agitating them with such solution; and, where the ores or tailings contain, in addition to the gold and silver values, such metals as copper, nickel, zinc or cobalt, all these metals including the gold and silver are simultaneously extracted, and the separation of the gold and silver afterwards effected in any ordinary manner.

"It will be seen that by my improvement a very cheap gold and silver solvent is obtained, which overcomes all the objections and conditions above referred to.

"Having thus described my invention, what I claim as new and desire to secure by letters patent is:

"1. The improvement in treating ores and tailings containing gold and silver, which consists in extracting the precious metals with a solution of ammonia and an alkaline earth-metal cyanide.

"2. The improvement in treating copper, nickel, zinc and cobalt ores and tailings with gold and silver values, which consists in simultaneously extracting all the metals with a solution of ammonia and an alkaline earth-metal cyanide."

Then the following periodicals published articles and abtracts relative to the subject: *Mining and Scientific Press*, May 15, 1909; *Engineering and Mining Journal*, April 17, 1909 (Letter), D. Mosher; *Engineering and Mining Journal*, Feb. 27, 1909, (Letter), S. E. Bretherton; *Pacific Miner*, March, 1910; *Mexican Mining Journal*, August, 1910; *Metallurgical and Chemical Engineering*, May, 1910 (Abstract); *Mineral Industry*, Vol. 18, 1909, page 365 (Abstract).

In the above articles, abstracts and letters, calcium cyanide as a cheap cyanide salt substitute produced from calcium cyanamide as per equation.

$$Ca_2\,CN_2 + C = 2\,Ca\,(CN)_2$$

was several times noted and its use recommended by me as a cheap cyanogen agent for extracting gold and silver from their ores.

I trust I have made clear my position in regard to the "Clancy Process" as relates to the use of cyanamide. The fact that Mr. Clancy employs iodide in conjunction with the cyanamide is of no importance to the intrinsic claim involved of employing cheap cyanamide as a substitute for the expensive cyanides of sodium or potassium.

San Francisco, Cal. D. Mosher.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—After a careful perusal of Mr. Mosher's communication I fail to see any cause for discussion as to "priority" in the use of cyanamide in ore treatment. Nowhere, including his patent specifications, does it appear that he suggests the use of cyanamide per se as a solvent for the precious metals. On the other hand, he simply proposes, as many others before him have done, the use of calcium cyanamide as a starting point in the manufacture of calcium cyanide by heating to a red heat calcium cyanamide with other materials in an electric furnace and the product of this mixture—calcium cyanide—he proposes to apply to the treatment of ores.

The German chemists, Messrs. Frank and Caro, many years ago attempted to produce cyanides by this means, but found that this method did not work out. I am reliably informed by those actively interested in the manufacture of cyanamide that fortunes have been spent to make available the application of cyanamide in a thoroughly practical and economical way and these parties have very generously and frankly conceded that I have solved the problem. It seems to me that it would be more to the point if Mr. Mosher took issue with Frank and Caro for the determination of the question of "priority," for there appears to be little, if any, difference in the result finally accomplished by either of them, for they were apparently working along the same lines and the results quite naturally were the same.

As to the idea of my infringing any rights and prerogatives of Mr. Mosher, it will be apparent to anyone reading Mosher's specification that he does not use cyanamide in the cyanide solution, but proposes to use his electric furnace product, which contains calcium cyanide of about 80 per cent cyanogen content prepared by secondary electric furnace treatment of cyanamide, as already described.

In the Clancy process any amide or amidine compound may be used in conjunction with the cyanide solution, such, for example, as urea, guanidine, etc.

It is somewhat surprising that Mr. Mosher does not also claim priority to an amide or amidine compound in conjunction with the cyanide solution.

I trust that I have made the differentiation clear in regard to the Clancy process, as it relates to the use of cyanamide in the cyanide solution and not the use of the calcium cyanide product prepared from calcium cyanamide by special treatment as described by Mosher.

It is abundantly clear from reading Mr. Mosher's specification that he does not attempt to claim the use of raw cyanamide in the ore treatment process, for the simple reason that he, like many others, was not aware of the fact that cyanamide, amide or amidine compound could be used in conjunction with the cyanide solutions until the publication of my patent specifications and the articles which I have written disclosing the same.

New York City. John Collins Clancy.

A New Process for the Refining of Pig Iron.

By J. B. Nau.

The process to be discussed in this article has for its purpose the preliminary refining of pig iron, more especially of phosphoretic pig iron too high in silicon to be treated directly and economically in the open-hearth furnace.

Other methods with the same purpose in view are in successful operation to-day. Thus, for instance, the so-called duplex process, which, I believe, was first recommended by me in 1890 as the best method to treat the highly siliceous and phosphoretic pig iron of the South. As is well known the duplex process, in principle, consists in submitting the raw liquid pig iron to a blow in the Bessemer converter, whereby the silicon and part of the carbon are eliminated from the iron before the latter is taken to the open-hearth furnace.

Naturally in the Bessemer blow some iron is burnt besides the manganese, but no phosphorus is touched. The resulting metal will, therefore, still contain all its phosphorus and whatever carbon it is deemed necessary to leave unburnt in order to keep the metal fluid enough for subsequent operations. The blown metal being of a uniform and regular composition the process has become quite popular in spite of the loss of iron entailing the blow.

It was principally to avoid this loss of metal, as well as to remove phosphorus instead of carbon in the preliminary operation, that the present method was devised.

Several years ago I obtained a United States patent (786,048, March 28, 1905) on a refining method in which I proposed to submit pig iron, running from the blast furnace, to the refining action of solid pieces of iron ore. This method consists in running the liquid iron through a column of heated ore and let a bath of metal accumulate on the bottom of the apparatus in such a manner that the lower part of the refining ore column is kept immersed in the liquid iron.

The intimate contact between iron and ore at a high temperature will rapidly effect the refining to the desired extent and if the final temperature remains high enough both iron and resulting slag will remain fluid enough to be run easily and without difficulty from the refining apparatus.

The final temperature in the operation will be influenced by: (1) The surplus or available heat stored away in the liquid iron running from the blast furnace. This temperature is mostly several hundred degrees Centigrade higher than necessary for good fluidity and the surplus heat thus carried can be made to largely contribute to the success of the refining operation. (2) The heat brought into or developed by the reaction taking place in the refining. This heat depends on several factors, such as amount of gangue in the refining ore, sensible heat carried by the refining ore, which can easily be heated up to 1000° C. or more without melting.

It also depends on the amount of silicon, phosphorus, etc., present in the original pig iron.

The elimination of silicon by means of Fe_2O_3 will be accompanied by a rise of temperature, while the elimination of phosphorus, and especially carbon, will under the same conditions be accompanied by an absorption of heat and consequently a lowering of temperature.

Hence a pig iron with a higher silicon content will be more satisfactory from a calorific standpoint.

Thus with a good temperature of raw pig iron and refining ore and satisfactory chemical composition of the same a final temperature can be obtained high enough for all practical purposes.

But a metallurgical process cannot be declared successful without its being so under all reasonable conditions. There would certainly occur many instances where the preliminary conditions necessary for success could not be obtained readily and where, therefore, the final temperature could not be kept high enough to have the metal sufficiently fluid.

The special purpose of the present invention is to provide the method patented by me several years ago with means that will make it successful at all times and under all reasonable conditions.

This can be best obtained by carrying out the operation in an electric furnace, especially built to create easy immersion of the refining ore in the metal to be refined.

Outline Description of Electric Furnace.

The furnace used for the process may vary.

In the present instance I have devised an electrical furnace capped by a water-cooled C. I. roof and surmounted by an ore shaft with a lateral inlet for the liquid pig iron to be refined. Vertical electrodes penetrating the roof are placed around the shaft. The bottom is made of carbon paste and the electric current playing between the electrodes and the carbon bottom traverses the liquid metal and slag, thereby keeping it heated up to the desired temperature in case there is not enough heat developed by the reaction.

If the heat conditions are enough no electric current need be turned on. Thus success is made possible under all conditions.

The furnace is so arranged and the operation will be so conducted that the refining ore necessary will always be kept immersed or made to be immersed to any desired depth in the metal to be treated and as soon as the ore in the bath is consumed and has to be replaced by new ore the latter will always be made without any trouble to be immersed in the liquid bath. The means employed for doing this are of the simplest kind and the buoyancy of the liquid metal can never prevent the much lighter ore from sinking down into the heavier liquid metal wherever the special means employed for that purpose are put in operation.

As it will be shown hereafter that on the other hand perfect fluidity of the refined metal and the resulting slag will at all times be maintained through electrical heating the method cannot but be successful.

Balance of Materials in the Elimination of Silicon, Phosphorus and Manganese.

In an article published in the *Iron Age* of March 23 and April 6, 1905, I have established a thermal calculation of the process which, owing to a mistake in the balance of material, I have since found to be not quite correct.

Therefore, in this paper I again establish that calculation and base it on a temperature of 1400° C. for the inflowing liquid iron from the blast furnace and 800° C. for the refining ore and guided by this thermal calculation I figure out how much electrical power I will require to keep the final temperature of the refined metal and the resulting slag at 1400° C.

I also calculate more roughly how much electrical power I will require when availing myself of the heat stored away in the pig iron to be refined to the extent of allowing the temperature of the refined metal to drop from 1400° C. to 1250° C. The calculations follow here in kilograms and Centigrades.

Assumed composition of pig iron to be refined:

	Si	Ph	Mn	C
In percentage	2%	1%	1.5%	3.5%
Per ton	20 kg	10 kg	15 kg	35 kg
Removed in the refining.....	19 kg	8 kg	13 kg	0
Refined metal will contain.....	1 kg	2 kg	2 kg	35 kg
In percentage	0.1%	0.2%	0.2%	3.5%

The refining ore is supposed to contain 90 per cent of Fe_2O_3 (63 per cent Fe) and 10 of gangue supposed to be SiO_2.

Slag to be formed is supposed to be in the nature of a puddle cinder with 50 per cent of metallic iron.

Results obtained in the refining:

19 kg of Si yielding 40.72 kg of SiO_2 will require 21.72 kg O
8 kg of Ph yielding 18.32 kg of P_2O_5 will require 10.32 kg O
12 kg of Mn yielding 16.78 kg of MnO will require 3.78 kg O

40 kg eliminated 75.82 in slag 35.82 kg O

This O being solely derived from the refining ore the amount of the latter must be sufficient

(1) To oxidize the elements Si, P and Mn to the extent indicated,

(2) To obtain a slag containing 50 per cent of metallic iron, corresponding to 64.283 per cent of FeO.

The unknown quantities in our problem are:

The weight of the slag obtained..... $= x$

The weight of the ore used..... $= y$

We know that the ore used contains 10 per cent of SiO_2 and (90 per cent of $Fe_2O_3 = 63$ per cent Fe and 27 per cent O), hence $y = 0.63\ y$ iron $+ 0.27\ y$ O $+ 0.10\ y$ SiO_2.

We also assume that the slag contains 64.283 per cent FeO and 35.714 per cent gangue = 100 per cent, hence $x = 0.25714 \times$ gangue $= 0.64286\ x$ FeO.

The total oxygen required in the refining is 35.82 kg and the weight of the oxidized elements removed is 75.82 kg.

As the gangue of the slag is composed of the gangue derived from the ore and the oxidized elements eliminated from the iron we have equations

(1) $$0.35714\ x = 75.82 + 0.10\ y$$

On the other hand the total oxygen will be derived from the ore, hence equation

(2) $$35.82 + \frac{16}{72} \times 0.64286\ x = 0.27\ y$$

The two equations will finally give us:

$y = 287.6$ kg of ore and

$x = 292.83$ of slag = 104.582 kg gangue and 188.248 Fe O (50 per cent Fe).

The 287.6 kg ore containing 90 per cent = 258.85 kg of Fe_2O_3 with..... 77.652

kg O, of which will be used for purifying..... 35.820

Leaving for FeO in slag..... 41.832

On the other hand, the slag contains 50 per cent = 146.415 Fe, corresponding to $146.415 \times 2/7 = 41.833$ kg O, obtained from 209.164 Fe_2O_3.

This leaves $258.85 - 209.16 = 49.69$ Fe_2O_3, yielding 34.77 kg directly reduced iron.

Composition of resulting slag:

		In %
SiO_2 derived from ore $287.6 \times 0.10 = 28.76$)		
SiO_2 derived from refining $= 40.72$ (	69.48.....	23.72
P_2O_5	18.32 kg	6.26%
MnO	16.78	5.73
FeO	188.25	64.286
		99.996%

The balance sheet of materials will be:

Put in Refiner.		Taken Out of Refiner.	
Pig iron	1000	Refined iron	960
Iron ore	287.6	Slag	292.83
		Iron reduced	34.77
Total	1287.6	Total	1287.60

Thus, under the conditions considered, each 1000 kg of pig iron treated will yield: 994.77 kg of refined iron containing very nearly 0.1 per cent of Si — 0.2 per cent Ph — 0.2 per cent Mn and 3.5 C and 292.83 kg of slag with 50 per cent of Fe.

No addition of flux, such as, for instance, limestone, was considered. The presence of limestone, on one hand, would no doubt reduce the amount of Fe found in the slag and, on the other hand, it might make it of considerably less value than a slag containing 50 per cent of iron and 5.73 per cent MnO.

Heat Balance in the Elimination of Silicon, Phosphorus and Manganese.

Specific heat of ore containing 10 per cent SiO_2 and 90 per cent Fe_2O_3 from 0° C. to 800° C. = ... 0.21

Specific heat of iron from 0° C. to 800° C..... 0.157

Specific heat of slag from 0° C. to 800° C..... 0.21

Total heat carried by slag at 1400 C..... 500 calories

Total heat carried by liquid iron at 1400° C.... 300 calories

Silicon burning to SiO_2 develops..... 7830 "

Phosphorus burning to P_2O_5 develops..... 5760 "

Manganese burning to MnO develops..... 1723 "

Iron burning to Fe_2O_3 develops..... 1796 "

Iron burning to FeO develops..... 1352 "

The heat of formation of slag is not known, but considering that wherever heats of formation of phosphates and silicates were established the heats of formation of phosphates per unit of acid were always found to be much greater than the heats of formation of silicates uniting with the same amount of the same base one, no doubt, will remain far below the truth when taking for the phosphates the same heats of formation per unit of acid as for the corresponding silicates.

It may also be noticed that heat of formation per unit of a silicate of lime of lower basicity is considerably less than the heat of formation of a silicate of higher basicity, hence per analogy one may assume that the heat of formation per unit of acid of a silicate of iron of higher basicity is much higher than the heat of formation of FeO, SiO_2, the only one that to my knowledge was ever established.

Taking, therefore, the same heat of formation for phosphates and silicates and adopting as unit 150 calories obtained when one unit of SiO_2 combines with FeO to form FeO, SiO_2 I undoubtedly will remain far below the truth and therefore remain within very conservative limits.

The heats of formation of silicides, phosphides and carbides are too uncertain or too little known to be considered here. Their dissociation would no doubt be accompanied by an absorption of heat and a lowering of temperature, but it may be fairly assumed that their omission will be counterbalanced by the omission of the source of heat resulting undoubtedly from the higher heat of formation of the slag than the one considered.

The heat lost by radiation was based on the figure of 440,000 calories per ton of iron established by Gruner in his studies of the blast furnace. As Gruner obtained this loss by difference between the total heat supplied and the actual heat absorbed, both in reactions and in the gases leaving the furnace, and as his calculations were based on a run of 24 hours of a blast furnace making less than 40 tons of iron per day—and as his loss includes also the heat carried off by the cooling water—the item of heat loss due to radiation in a refining apparatus of a capacity of 500 tons per day, having no cooling water and with very much less radiating surface than Gruner's blast furnace, may be considered very ample.

The itemized heat balance per ton of iron treated will follow here:

LIQUID IRON AT 1400° C.[1] ORE AT 800° C.

Heat Supplied:	Calories.	
19 kg of Si will give 19×7830.....	148,770	
8 kg of Ph will give 8×5760.....	46,080	
13 kg of Mn will give 13×1723.....	22,399	
59.04 SiO_2 and P_2O_5 will give 59.04×150....	8,854	
SiO_2 from ore is omitted.		226,103
Heat Spent:		
35 kg iron—reduced from $Fe_2O_3 = 35 \times 1796 =$	62,860	
heated from 800° to 1400° = 35		
$(300 - 800 \times 0.157) =$	6,104	
$293 - 40 = 253$ kg of slag		
Heated from 800° to 1400° = 253		
$(500 - 800 \times 0.21)$	83,996	
The 40 kg of constituents removed were at 1400°		
209.16 kg of Fe_2O_3 reduced to FeO		
$209.16 \times 444 =$	92,867	
Heat lost by radiation (per ton).....	25,000	
		270,827
Heat deficiency		—44,724

Balance of Material in the Elimination of Carbon.

All conditions remaining the same the above heat balance might be considered a constant.

So far we have not considered the carbon, whose elimination will largely depend on the temperature and the length of contact. Still the elimination of a small percentage only of carbon as CO will constitute a great disturbing factor in the heat balance and it is, therefore, necessary to consider its elimination within the limits that it can reasonably be expected to take place.

We apply the calculations to the elimination of 1 kg of C. We shall have again:

Ore necessary: $y = 0.63\ y$ iron $+ 0.27\ y$ O $+ 0.10\ y$ SiO_2

Slag: $x = 0.35714\ x$ gangue $+ 0.64286\ x$ FeO

With the removal of carbon the weight of the slag is increased only by a corresponding amount of gangue and FeO (50 per cent Fe) derived from the ore.

Hence in this case we have

(1) $0.35714\ x = 0.10\ y$

The total oxygen entering into the reactions will be composed of 4/3 kg necessary to remove C as CO plus the oxygen of the FeO of the slag. Hence,

(2) $4/3 + 16/72 \times 0.64286\ x = 0.27\ y$

which equation will give us

Slag $x = 1.623$ kg with 0.579 gangue and 1.044 FeO with	0.232	O
Ore $y = 5.7964$ kg with 0.579 gangue and		
5.218 kg of Fe_2O_3 having 3.6526 iron and..	1.5654	O
Oxygen necessary to remove 1 C.....	1.3333	
Leaving O in Fe_2O_3 of slag.....	0.2321	

Metallic iron reduced: $3.6525 - 0.812 = 2.8406$ kg.

In regard to 1 kg of C the balance sheet of materials will therefore be:

Put in Refiner.		Taken Out.	
Ore = ...	5.796	Metallic iron reduced.	2.8406
Carbon = ...	1.000	Slag	1.623
		CO escaping	2.333
Total	6.796	Total	6.796

COMPOSITION OF SLAG DUE TO THE ELIMINATION OF 1 KG OF C.

	In %	
SiO_2 = ...0.579.....	35.68	
FeO = ...1.044.....	64.32	(50% Fe)
Total .....1.623 kg	100%	

HEAT BALANCE DUE TO THE ELIMINATION OF 1 KG OF CARBON.

Heat Giving Factors:	Calories.	
1 kg of C burning to CO.....	2403	
5.79 kg ore at $800° - 5.79 \times 800 \times 0.21 =$	972	
1 kg of carbon at 1400° C. = ...	300	
0.579 of SiO_2 uniting with FeO, 150 calories = ..	87	
Total		+ 3762
Heat Consumed:		
2.84 of Fe reduced from $Fe_2O_3 = 2.84 \times 1796 =$	5100	
2.84 of Fe heated from 800° to 1400° and melted $= 2.84\ (300 - 800 \times 0.17) =$	465	
1.16 of Fe_2O_3 reduced to FeO $= 1.16 \times 444 =$ ..	515	
1.623 of slag heated from 800° to 1400° C. = 1.623 $(500 - 800 \times 0.21) =$	538	
Heat carried off by 2.333 kg of CO at 500° = $\frac{(2.333}{1.25} = 1.8664$ m.³) $\times (0.303 + 0.0027 \times 5) \times 500 =$	294	
		− 6925
Deficiency of heat per kg of carbon eliminated =		− 3163

Considering that the whole operation necessary to treat one ton of iron will not last more than 10 minutes, which is actually more than there is necessary in the wash process where hardly any carbon at all is removed, at a higher temperature of refining ore we suppose the carbon reduced from 3.50 per cent to 3.30 per cent corresponding to a total of 2 kg of carbon per ton of iron, the loss of heat due to this cause will be:

$$2 \times 3163 = 6326 \text{ calories}$$

Thus per ton of iron treated the whole loss of heat will be:

	Calories.
Removal of silicon, phosphorus, manganese to the extent shown	44,724
Elimination of 2 kg of carbon.....	6,326
Total loss of heat per ton of metal treated.....	51,050

Electrical Power Necessary.

We propose to supply the above deficiency of heat by electrical power so as to keep the temperature of the refined metal and the resulting slag at 1400° C.

If we apply the method to the refining of all the iron made in 24 hours in a 500-ton furnace we will have to refine per hour $500 \div 24 = 21$ tons and we will have to supply $21 \times 51{,}050 = 1{,}072{,}050$ calories per hour.

Taking the efficiency of an electric furnace as 80 per cent the current would have to supply

$$\frac{1{,}072{,}050}{0.80} = 1{,}340{,}062 \text{ calories}$$

corresponding to

$$\frac{1{,}340{,}062}{640} = 2091, \text{ say, } 2100 \text{ electric hp-hours.}$$

This electrical horse-power would be enough to refine the iron as fast as the furnace can make it.

The calculations are based on heat factors as they are known to-day and on that basis the results obtained should be correct.

Still, some of them are to a certain extent doubtful and their use may influence the results and show the heat conditions to be different from what they might be found in practice.

There will be no harm if practice shows a smaller heat deficiency than the one established; but, on the other hand, the heat deficiency might be found larger.

When it is considered, however, that the final temperature of 1400° C. is far above the temperature that perfectly fluid metal requires to easily run from the refining furnace, we have at our disposal a factor of safety of great value of which we have made no use. The temperature of the metal and cinder in a puddling furnace reaches hardly if ever 1250°. The temperature of pig iron running from the cupola seldom reaches 1250° C. and is mostly below 1200°. Yet at these temperatures both puddle-cinder and cupola metal are very hot and very fluid.

It would therefore appear to be unnecessary to have a final temperature of 1400°. If we allow this temperature to drop to 1250° excellent results are assured and the refining operation will become far more economical in so far that now we actually make use of the surplus heat unnecessarily stored away in the liquid iron and a correspondingly large amount of heat to be supplied by electricity will be saved. A cursory examination of the problem made under this new aspect will give the following results:

Allowing the temperature to drop from 1400° C. to 1250° C. would liberate an amount of heat as shown below:

1000 kg of iron from 1400 to 1250 liberates $1000 \times 150 \times 0.23 =$	34,500 C
293 kg of slag liberates $293 \times 150 \times 0.30 =$	13,185 C
Total loss	47,685 C

Leaving out all other factors that would make the case appear under a still more favorable light the amount of heat to be furnished by electricity will now be reduced to $51{,}050 - 47{,}685 = 3365$ calories per ton, or per hour $3365 \times 21 = 70{,}665$ calo-

ries, requiring a horse-power of $\frac{70,665}{0.8 \times 640} = 138$ hp.

Since, roughly speaking, the 47,685 calories correspond to 150° C., 51,050 calories would correspond to about 160°, which means that no electricity would be required when allowing the temperature to drop to 1240°, high temperature of very fluid metal running from the cupola.

Thus at times when conditions are quite favorable no electricity need be applied for success; but with less favorable conditions some electricity would be necessary.

Probably at no time the full amount of 2100 hp will be required, but with the electrical power at hand always ready to be applied success is assured under all conditions.

It might further be stated that the size of the hearth or crucible of the furnace can be determined so as to keep the metal to be refined and the refining ore in contact for any reasonable length of time, thereby making it possible to extend the heating of the metal over a greater length of time, which naturally would afford every facility to again sufficiently raise the temperature of the outflowing metal should this latter have cooled down more than desirable.

New York City.

Some Electric Furnace Notes.

By Joh Härdén, E. E.

The valuable contributions to the knowledge of the proper dimensions of furnace electrodes made by C. Hering in several issues of this paper will no doubt be greeted by furnace designers with acclamation. Many were the difficulties confronting the designer in the earlier days of the electric furnace era, both regarding the best size of electrodes and the most efficient way of arranging them, and also, last but not least, where to get them.

Some twelve years ago the author was given the problem of designing a graphitizing furnace for carbon rods in Russia.

The current available was direct current, about 3000 amps by 65 volts, no alternating current being obtainable at the works; in fact, the furnace was to be operated in series with a row of electrolytic cells and intended to absorb surplus power in a useful way. The carbon rods to be graphitized were about 3½ in. diameter by 4 ft. long. Owing to these two facts the operation could not be carried out "in bulk" as is the practice at Niagara, but had to be in a continuous operation. It was found subsequently that the best method under the circumstances was to pass three rods at a time slowly through the furnace, followed by three others in succession.

At first, attempts were made to produce a large arc in a carbonaceous atmosphere. This method had, for obvious reasons, to be abandoned very soon. The arc behaved very well for 6 to 8 hours, but as soon as the interior of the furnace got well heated the furnace walls became part of its path being conductive at that temperature, and even directing magnets had no beneficial influence owing to the irregular paths taken by the arc. Also, the rods to be treated got scorched and injured by the action of the arc.

In this connection it might be mentioned that the suggestion made by A. Mahlke in Vol. IX, No. 1, p. 42, to broaden the active ends of the electrodes, so as to obtain a better distribution of the arcs and the heat in the furnace, is not likely to be successful. Anyone who has had an opportunity to observe the arc, even in a furnace of the largest size, will have noticed how the base of the arc is always concentrated to a comparatively small area, wandering about on the surface of the electrode, how large this also may be in cross section. This is no doubt due to the fact that the arc requires carbon vapor of high temperature (or any other ionized medium of high temperature), and as this cannot take place at the same instant over the whole surface of the electrode, the arc will always be concentrated at its base. The same may also be observed in a Mercury-vapor lamp; the arc forms only a comparatively small spot on the surface of the mercury cathode.

Reverting again to the graphitizing furnace a further attempt was made to convert it into a resistance furnace in the following way: Two electrodes, about 6½ in. square, were placed inclined on opposite sides into the furnace chamber of fire brick; the bottom consisted of electrode stumps and stamped coke. The furnace was partly filled with granular charcoal, in which the electrodes were embedded about 7 in. to 8 in.; due provision was taken to prevent the current passing through the furnace walls, similar to the arrangement used in the pig iron smelting furnace by Gronwall & Stalhane.

This arrangement would work satisfactorily for about a day or so; after that period, the trouble began. In spite of the air being excluded as far as possible from the electrode collars the electrodes were worn out just inside the furnace room very rapidly, while the active ends grew in thickness by the setting-on of carbonaceous matter to such an extent that the electrode took the shape of an irregular mushroom, causing great difficulties in the regulating.

A final alteration was then made, which proved successful. The bottom of the furnace was built up on a hollow cast-iron plate, similar to that in a carbide furnace, two of which were in operation at the works as well. Electrode stumps, about 8 in. high, were placed in carbon cement on the iron plate and the brickwork erected to form the furnace chamber.

This bottom plate formed the positive electrode, while the negative one consisted of a 6½-in. x 6½-in. carbon suspended from above, by means of a water cooled holder, through an opening in the furnace roof. The furnace chamber was filled with granular charcoal as before, in such a manner as to reach the upper electrode and sloping down at the sides, which prevented the current passing through the walls. The rods to be graphitized were passed three by three slowly through the glowing mass of charcoal and allowed to cool in coal dust on a gantry. The rods were examined as to degree of graphitizing by tapping with a steel bar; the sound emitted gave a good idea of the process; also the polishing effect by rubbing.

One difficulty remained to be solved; the roof was soon destroyed in each case, and we may assume that this difficulty is still a great stumbling block, even to-day, in all electrode furnaces. In the above instance, the problem was solved in this way: The roof of the furnace was arched in by means of scrapped or curved carbon electrodes, separated and wedged in by means of intermediate wedge-shaped high-grade fire bricks, to prevent leakage of current. After some adjustment this kind of roof could be made to last for 8 to 9 months in spite of the continuous run day and night at a tremendous temperature, and despite the fact that the furnace was emptied and fresh charcoal refilled every twenty-four hours, which caused a heavy fall in temperature in fact, this refilling was the primary cause of the rapid breakdown of the original brick roof. One top electrode would, under these circumstances, last four to five days, its dimensions being, as already stated, 6½ in. x 6½ in. x 4 ft. 6 in., and the current 3000 amps to 3500 amps at 40 volts to 45 volts. As far as is known to the author the bottom needed no repair at all during a campaign of four years.

A curious incident in the case of the trials with the arc-design was the upshot of a number of further experiments and also some patents.

When the large arc was started, violent rushes of current took place, the whole furnace being cool. This was strongly objected to by the power station master, as it caused very strong surging between the large engines running parallel; the regulating device of the steam engines was of such nature as to be particularly sensitive in this respect.

While awaiting the construction of a suitable starting resistance (an iron band immersed in running water) a makeshift was made in trying to start the arc, and also to prevent

its extinguishing, by means of a spark coil, made to discharge between the electrodes. In order to prevent the spark taking any other way than was intended, a condenser was connected to the sparking circuit and a primitive reaction in the power circuit.

By this means it was quite possible to start the arc without any serious current rush and the tendency of the arc to go out before the furnace was heated ceased if the spark was kept going.

This device was first tested with a smaller arc and narrower electrodes; it was then observed that the tips of the electrodes got more quickly graphitized and that the graphite formed was of a more flaky appearance when the spark had been in operation for some time, though not always, only under certain conditions. The whole phenomenon was at first thought to be accidental only, but on further investigation it was found that the graphitizing could, under favorable conditions, be somewhat accelerated by introducing condenser discharges of practical magnitude. A specially made spark coil and a suitable condenser were therefore installed and put into permanent use; the workmen were protected from the discharge by means of heavy mica collars, surrounding the auxiliary electrodes through which the discharges were introduced into the furnaces.

Other and more pressing work prevented a more systematic and exhaustive investigation as to the economical efficiency of the method; it may be said in this respect, however, that its use appeared to be justified. The workers, a man and a boy per shift, were paid after piecework for a minimum number of rods, passed by the foreman after each shift, and an additional bonus for each extra rod passed. It was regularly found that if the spark coil had been out of operation for some

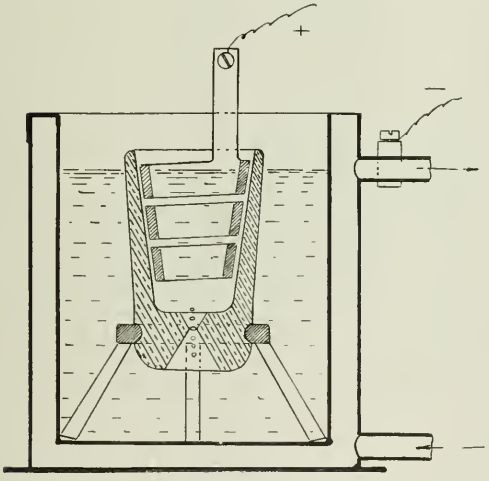


FIG. 1.—INTERRUPTER DESIGN.

time during a shift the number of rods passing examination decreased and the rejected rods had to be sent through the furnace again; therefore the workmen showed a keen interest in keeping the coil in going order.

As there was no alternating current available at the works, the coil had perforce to be operated by direct current, 125 volts. Anyone who has had the task of keeping a large coil, giving a 12-in. to 14-in. spark of great volume, going day and night without breakdown for several months at a time will testify that the finding of a suitable interrupter is by no means an easy problem.

At first, the ordinary mercury make and break, consisting of two motor-driven vertical pins, was tried, but it was soon found that this did not answer. The pins were rapidly worn out, regardless of what metal they were made; besides, the mercury soon got clogged up in spite of a thick strata of oil over it. Carbon rods lasted longer, but the spark was greatly diminished, owing to insufficient rapidity of break.

A turbo-interrupter was then tried, in which a mercury jet in the well-known manner was projected against a toothed iron wheel, moving in oil or cooling liquid. Even this proved futile, as the teeth of the wheel were soon worn off by the sparking; the mercury got so clogged in three or four days that it had to be replaced. To prevent this latter nuisance, a jet of coal-gas was introduced into the vessel, so as to prevent oxidation, but even this measure, though it did some

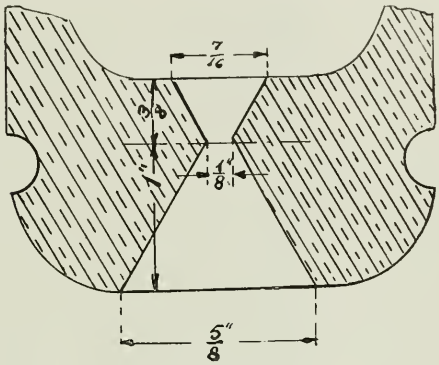


FIG. 2.—CORRECT SHAPE OF BORING.

good, did not secure from breakdowns to the extent of commercial efficiency.

The electrolytic interrupter was finally resorted to and, in spite of its low electric efficiency, proved to be more satisfactory than the others. A platinum point, as introduced by Dr. Wehnelt, was first tried, but was soon destroyed by the current. After a good deal of experimenting it was found to answer the purpose best if a large porcelain insulator drilled through the top, without any metal point at all, was used.

As indicated in Figue 1, the insulator was suspended in a doubled-walled vessel of lead, cooled by running water. A lead spiral formed the inner, or positive electrode; the active liquid was sulphuric acid of about 1.14 Sp.W. The paper condenser in the base of the coil was not in circuit while this interrupter was at work.

With this device it was found quite possible to keep the coil going day and night for many months, only stopping about every fifth day for cleaning and replacing of the acid, which was gradually vaporized; but such stoppage only required about 15 to 20 minutes.

The hole drilled in the insulator was gradually widened by the action of the current. It had to be replaced about every three months, for which purpose a spare one was always kept.

At the beginning, some difficulty was found in the tendency of the apparatus to act as a pump. In some cases the acid was driven out through the bottom hole into the outer vessel, causing the working to be irregular and producing much heat. Again, with another insulator substituted, the acid rose in the inner room so as to overflow and cause short circuit.

It was soon found that the shape of the hole was responsible for this behavior. If the hole sloped too much upwards, the gas bubble causing the interruption exploded *upwards*, thus driving the pile of liquid up into the inner vessel, causing a surging from underneath, so that the liquid in the outer vessel rushed inwards. If this happened rhythmically, as was often the case, the insulator was soon filled to overflowing.

On the contrary, if the boring widened too acutely *downwards* the bubbles burst outwards and drove the liquid into the outer vessel.

In Fig. 2 is indicated the correct shape of the boring, with which no surging took place. The hole should be well polished, or preferably glazed, when the action was much minimized. The period of interruption was adjusted by means of a variable inductance in the primary circuit, so as to give the best effect.

Large glass condensers were used in the secondary circuit to produce the oscillations necessary.

With this arrangement the breakdowns were very rare, and the whole apparatus could be left for four to five days without inspection.

London, England.

Metallurgical Engineering Education.

By Victor C. Alderson.

The forces which influence the course of study in our technical schools are different from those which influence the cultural college. In the case of the latter these influences are from within. The members of the faculty offer such courses as they deem wise, treat them as they choose, and develop such a man as seems good to them. The graduate goes out trained for nothing in particular; if he succeeds in business, for example, his success is due rather to his natural aptitude for business than to any special training the college of general culture has given him.

In the case of the technical school the influences which determine the training a man shall receive come from without. He enters a profession and must understand its fundamental principles and must start at the bottom. The standard for graduation—the level he must attain—is determined by the profession he intends to enter.

For this reason, the work of our technical schools is more exacting than is the work of the cultural college. In the one case the standard to be reached is determined within the college circle and is more or less indefinite; in the other case, it is fixed by an outside, independent authority and is rigidly determined by the demands of the profession. Naturally, then, in our technical schools there is an exceptionally fine spirit for work, for close application to details, for a close co-ordination of laboratory and lecture room work, for clean-cut, usable knowledge, and for thorough training.

Not many years ago the mining school that offered a course in the general principles of underground work, of timbering, and of shaft sinking, with some little chemistry, and a superficial knowledge of geology, met the needs of students who desired to become mining engineers, and satisfied the demands of the mining industry. Within the last few years, however, the demands of the industry have become so broad and so deep that the older course satisfies neither the needs of the prospective mining engineer nor the demands of the profession.

Mining is now subdivided into coal mining, mining geology, non-metallic mining, metal mining, and metallurgy. The mining school of to-day which would be up to date must recognize all of these branches of mining. It must not make the mistake of allowing its graduates to be such one-sided specialists that, on leaving college, they can do only one thing. The function of the technical school is not to graduate unfledged specialists, but to develop a young man so that he can enter the profession at the bottom and eventually become a man of power and of resources. To this end a thorough grounding in fundamentals is absolutely necessary.

Long observation of men, trained and untrained, leads one to see that the mere practical man soon reaches the limit of his development. This is due primarily to the fact that his observations are limited, he cannot assimilate the condensed experience of others, and he lacks fundamental concepts and general laws. He is filled merely with facts and personal experiences. The trained man, however, has the basal laws of physics, of chemistry, and of mathematics thoroughly implanted in him as a mental substructure into which all future facts and personal experience can be logically placed. In the one case there is, almost universally, a limit of development beyond which a man cannot go; in the other case, there need be no limit to growth and development. If this were not so, our technical schools would have no good reason for existence.

The mining school of to-day must not only feel the pulse of the mining industry but must recognize early the range of pulsation. It is not sufficient to be satisfied with the accomplishments of a decade ago. Coal mining, for example, has become a settled industry. The rewards are not great but they are fairly sure. The most recent demands in this branch of mining are for a thorough knowledge of the action of mine gases, the explosibility of coal dust, rescue work, and "first aid to the injured." A school giving a course in coal mining to-day would be sadly derelict in its duty if it did not include these subjects in its curriculum. The ideals for this work have risen steadily in the past decade.

The mining of the non-metallic minerals has been developing so rapidly of recent years that this branch of mining is even now demanding attention. The clays, shales, granites, lava stones, and marble are now being sought as never before. The cement industry is growing prodigiously and the uses of cement are becoming so numerous that only a specialist can become familiar with them.

Mining geology, also, has only recently come into its own. Herein lies the union of the mining engineer and the geologist. The old time prospector, with his burro, his outfit, and his "nose for ore," is gone forever. In his place is the trained mining geologist whose prospecting is done quite as much underground as on the surface, who understands the principles of ore deposition and the play of dynamic forces, and who can read earth changes with little to guide him. The mining industry needs this man and the schools should supply the material from which he can be developed.

But after all is said the field of ore dressing and metallurgy contains the great problems for the mining engineer. When coal has reached the surface it is ready for market, but when the metal miner brings his ore to the surface his troubles have only begun. If his ore can be shipped directly to the smeltery he is merely relieved of the burden; he passes the problem on to the metallurgist. If his ore must be concentrated, or subjected to some form of local treatment like cyanidation, a new field of work opens before him. In this realm of ore treatment lie the serious problems of the future. Herein lies the great work of the mining schools—the work of training the future mill-man and the metallurgist.

The industry does not need a man versed merely in the theory of inorganic chemistry; there are many such who are not metallurgists. Certainly the metallurgist must be a master of applied inorganic chemistry, but he must be much more. Instead of thinking in grams he must think in tons; he must know not merely the refinements of the laboratory, but also the difficulties to be met in commercial practice. The steady decline in the cost of smelting during the past half century is a silent, though potent, witness to the improvements made. These have not been made by the practical man but by the trained man. There are many problems yet to be solved; it is the duty of the schools to develop the men who can solve these problems. Herein lies the ideal for the mining school. It is well known that neither the practical man working alone nor the theorist working alone can solve these problems. The man who has just the right combination is the ideal man.

The problem, then, before the schools is to make that combination of theory and practice which will develop the man to conquer the problems of ore treatment. Theoretical instruction combined with the ordinary laboratory is the combination generally used. The efficacy of this plan is doubtful. Some-

thing new is demanded in this age. That something is laboratory work on a full commercial scale, so that both the refinements of the laboratory and the difficulties of standard size work may be met and appreciated.

At the Colorado School of Mines this ideal is being realized by the erection and equipment of an experimental ore dressing and metallurgical plant, in which the equipment will all be of standard size and where ore in carload lots can be treated. In planning the building, the idea of flexibility has been kept constantly in mind so that any line of experimental work can be undertaken. The different sections cover ore dressing, cyanidation, roasting, and a section for metallurgical work. The plant will serve various purposes—a laboratory for the students; a place of research for advanced students, for members of the faculty, and for the mining engineer; and an experimental plant for the use of the mining industry.

The ideas involved in the plan are high but no higher than the needs of the industry demand. Since the great problems of the industry lie in the treatment of the ore, it is proper that the means adopted should include a great experimental plant for ore treatment. Such a plant will also provide adequate means for training the future metallurgist so that the school will be doing its duty not only to the mining industry but also to the young men who are looking forward to becoming the metallurgical engineers of the future.

Colorado School of Mines.

The Manufacture of Standard and Alkali-Resisting Portland Cement In Colorado.

Editorial Correspondence.

The statistics of the United States Geological Survey show that the production of Portland cement in the United States made a new high record in 1910, notwithstanding the fact that prices were not satisfactory, particularly in the East. In 1900 the manufacture of Portland cement in the United States

FIG. 1.—NIOBRARA LIMESTONE BED AT PORTLAND, COLO.

amounted to 8,482,020 bbl. The production increased steadily and rapidly until, in 1909, the output was 63,508,471 bbl. This output was further increased in 1910 about 15 per cent to 20 per cent, and is estimated at 73,500,000 bbl. to 75,000,000 bbl.

Portland cement manufacture in Colorado began about twelve or thirteen years ago, and there was but one plant in operation until 1908, when a second was started. In 1909 the third commenced production of a special alkali-proof cement, and the prospects are that all three will be operated at capacity this year. The raw materials used for standard Portland cement in this State are a fairly pure limestone from the Niobrara formation (Fig. 1) and an overlying argillaceous cement rock locally called shale (Fig. 2), and so designated in these notes. These materials are exposed on either side of the Arkansas River between Pueblo and Cañon City, and two of the Colorado plants, those of the Colorado Portland Cement Company and the United States Portland Cement Company, are located on the Arkansas River in Fremont County, at Portland and Concrete respectively.

The Portland Plant.

The Colorado Portland Cement Company operates a 3000-bbl. plant at Portland, Col., employing about 240 men under the direction of Mr. F. W. Brown, superintendent, and Mr. L. E. Foster, chemist. This plant is the oldest in the State. The accompanying illustrations show the lime and shale beds from which the raw materials are taken. Between the strata of limestone occur very thin seams of clay, which, in quarrying, break free from the limestone. It is, however, mixed with the spalled lime rock and sometimes used as part of the raw mixture.

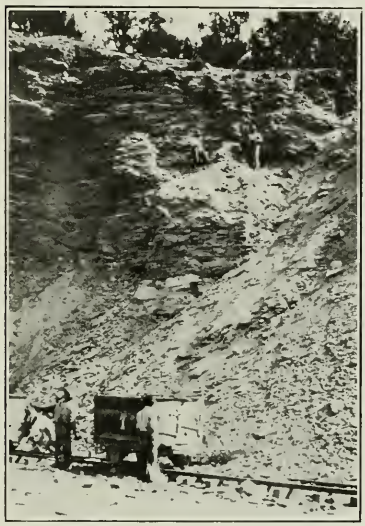


FIG. 2.—SHALE BANK AT PORTLAND, COLO.

Electric haulage is used for transportation to the stock house, where separate large bins are provided for lime rock, shale and spalls. These are drawn off through gates as needed and hauled to the crusher-room, where they are reduced by Gates No. 6 gyratories and rolls to about 1-in. size and elevated to separate bins. From these the crushed materials are properly proportioned and diverted to blending bins located in another building. This series of 20 blending bins is a special feature peculiar to this plant, and provides a means for securing uniformity in the raw mixture, which is drawn from several bins at once. Thus any error which might have been made in proportioning the materials in one bin does not materially affect the composition of the raw mixture drawn from several.

The method of proportioning the raw cement mixture is as follows: Beneath the lime and shale bins are weighing hoppers, one for each ingredient. The shale carries about 67-68 per cent $CaCO_3$ and the lime rock about 90 per cent. The shale is nearly a cement mixture, and lime rock is added in suitable proportion to make a mixture containing 74 per cent $CaCO_3$. The two materials are mixed in 8000-lb. lots, equivalent to about 12 bbl. of cement, and may contain from 7200 lb. to 7800 lb. of shale and 800 lb. to 200 lb. lime rock; or other proportions may be used according to the varying composition of the raw materials. A variation of 400 lb. of lime rock makes a change of about 1 per cent $CaCO_3$ in the mixture.

By this system of controlling the mixture the effect of a change in composition can be determined in one to two hours according to running conditions in the plant; if both dryers are running the change will be effective in about an hour.

This practice of mixing the raw materials before comminution differs from that at the other Colorado plant, but it is apparently effective, inasmuch as the mixing continues through the dryers and conveyors. The dryers are coal-fired and are of the direct-heat rotary-blast type, 50 ft. long and 54

in in diameter. The moisture is reduced to about 0.5 per cent.

The dried mixture is elevated to bins above the comminuters for preliminary grinding. A variety of grinding machines have been used at this plant, both on the raw and finishing sides, and there has been good opportunity to compare their work on different materials. For comminuters the preference at this plant favors the Jeffrey for raw mixtures and the Smith for clinker, as fine grinders, the Fuller mill is preferred for raw mixture and the tube mill for clinker. There is little preference between the Fuller and the tube mill for fine grinding of coal.

One Jeffrey and three Smidth mills are in use as raw comminuters. Of their product 85 per cent passes a 20-mesh screen. This material is conveyed to bins above a battery of five Fuller mills and five tubes, all used for finishing the raw mixture. The Fuller mill has a capacity of about 15 bbl. of rock per hour and requires about 75 hp. The tube will grind about 22 bbl., with a power requirement of 125 hp. Colorado pebbles are used in the tubes, the consumption being at the rate of 45 tons per month, or 1 lb. per barrel. Screen tests of the finished raw mixture show 92 per cent through 100 mesh and 82 per cent through 200 mesh.

The kiln-room contains nine rotary kilns, six being 60 ft. and three 120 ft. long. The 60-ft. kilns are 6 ft. in diameter; one of the 120-ft. kilns is 9 ft. in diameter and the other two are 7 ft. at the feed end and 7.5 ft. at the discharge. The raw mixture enters the kiln with about 8 per cent added moisture and is burned at a temperature of about 2800° Fahr. Southern Colorado slack dried and ground to 90-95 per cent through 100-mesh screens is fired under 3-oz. pressure delivered by Sirocco fans. The coal consumption is 135-140 lb. per barrel of cement. The kilns are operated by variable-speed motors and require about 6 hp for the smaller and 35 hp for the larger.

The hot clinker elevated from the kiln-room may be diverted to the stock pile or, if needed immediately, is cooled in a rotary cooler, 6 ft. x 120 ft., operated by a 32-hp motor. Clinker can be withdrawn from the stock pile by conveyor operating in a tunnel between the clinker storage and the finishing end.

The finishing side of this plant parallels the raw side, the direction of flow of the material being reversed after the clinker is burned. On the finishing side three Smidth comminuters and one No. 8 Krupp mill are used for preliminary grinding and eight tubes for finishing. The former grind about 85 per cent through 20-mesh. At this stage the retarder is added to control the setting time of the finished cement. Gypsum is used, and a sufficient quantity added to give about 1.6 per cent SO_3 in the finished product.

The quantity of gypsum added is controlled by a ratchet-and-pawl device which operates a screw conveyor at the bottom of the gypsum bin. There are several pawls of different lengths, any one of which can be thrown into use, and by using a longer or shorter pawl the speed of the screw can be increased or diminished, with proportionate regulation of the quantity of gypsum discharged. The mixed cement and gypsum are elevated, weighed on a Richardson automatic scale, and conveyed by screws to hopper bins above the tube mills. The tubes grind about 94 per cent through a 100-mesh screen and 78-80 per cent through 200-mesh, which is well above the requirements established by engineering societies, viz., not less than 92 per cent through 100-mesh and 75 per cent through 200-mesh.

The chemical and physical tests made are those customary at cement plants. The raw mixture is sampled hourly at the various stages of its comminution, and tested for fineness and percentage of $CaCO_3$. An average day and night sample of the finished raw mixture is similarly tested. The finished cement is sampled hourly, analyzed for sulphur and tested for fineness, setting time, tensile strength and effect of boiling temperature. Carload shipments are controlled with tensile strength tests on 7-day and 28-day briquets. The following figures are representative of the tensile strength of neat and sand (1:3) briquets, as developed in the times specified:

	24h.	7d.	28d.	90d.
Neat	345	735	850	850±
Sand	...	260	25 per cent gain	15 per cent gain

The following analyses are typical at this plant:

	Raw Mix.	Clinker.	Finished Cement
SiO_2	13.6	23.0	22.3
Fe_2O_3	5.8	3.2	3.2
Al_2O_3	5.8	7.5	7.4
CaO	42.3	64.5	64.0
MgO	0.5	1.0	1.0
SO_3	1.6	0.5	1.4
Ignition loss	36.8	0.1	0.1

The fuel used in the kilns carries about 18 per cent ash, and the determination of this factor is the only control exercised over coal purchases.

The company is making extensive improvements in its plant

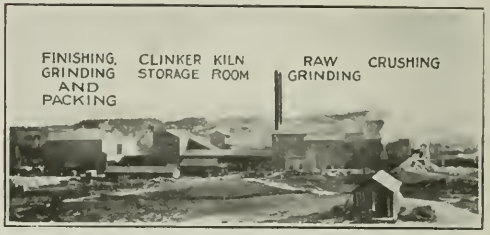


FIG. 3.—PLANT OF THE UNITED STATES PORTLAND CEMENT COMPANY, CONCRETE, COLO.

which will make for more economical operation. New crushing and drying equipment will be installed, and the power plant considerably enlarged, including the installation of a gas-producer plant. The cost of improvements will be $200,000

The Concrete Plant.

About one mile east of Portland is the town of Concrete and the cement plant of the United States Portland Cement Company (Fig. 3). This plant commenced production in 1908, almost a decade after the first operation of the Portland plant, and consequently is more modern in construction and in some features of operation. The plant has a capacity of 1500 bbl. per day and employs about 100 men. Operations are under the direction of Mr. William Fowden, superintendent, and Mr I. J. Dilts, chemist.

The same raw materials are used as at Portland, with the exception that the lime is hand picked and the clay seams and spalls are rejected. The argillaceous material also is carefully selected, only that rich in lime carbonate being used. The clean limestone averages about 91.5 per cent $CaCO_3$ and the clayey cement rock about 71.5 per cent. The plant is so situated with respect to the quarries that the rock is transported by gravity, and the empty cars returned by gasoline locomotive. The successive units of the plant are arranged in a straight line, differing in this respect from the plant at Portland, where the raw and finishing sides are parallel.

The rock is dumped directly to two Symons gyratories, Nos. 5 and 6. These crushers are the same as used in metallurgical plants in the West, a brief reference to which was made in our issue for February, 1911, page 92. The crushed rock is elevated to conical bottom steel bins, three of which are for shale and one for lime rock. These bins hold sufficient material for a 48-hour run of the plant. Each class of material is separately dried and ground before mixing, and in this respect the practice differs from that at Portland, where the raw materials are mixed immediately after crushing and before drying. Belt

conveyers from the bottom of each bin run to two Ruggles-Coles dryers, with exhaust fans and Cyclone dust separators attached.

One of the cardinal points of cement manufacture is the fine grinding and intimate mixing of the raw materials. The tendency at this plant is toward finer grinding, both on raw and finished products. One Williams and two Jeffrey mills are used as raw comminuters, grinding about 88-90 per cent through 20-mesh screen and 45 per cent through 100-mesh. The ground materials are elevated and diverted to bins, of which there are four placed in line, three for shale and one for lime rock. It is from these bins that the raw mixture is made and controlled by a system which is very accurate.

A short screw conveyor draws the material from each bin. The three screws on the shale bins are operated by one variable-speed induction motor, and deliver to a large screw conveyer running at right angles to them. The screw on the lime bin is similarly operated by a variable-speed motor, and likewise delivers to a second screw running at right angles to it. These cross-conveyors deliver to a common point where the lime and shale are mixed. The speed of the motors and screws is controlled by the chemist through rheostats placed in the laboratory. The standard mixture desired is one containing 75.5 per cent $CaCO_3$, and is made up of about 25 per cent lime rock and 75 per cent shale.

By previous experiment the chemist has determined the percentages of lime rock and shale which will be delivered at various speeds of the screws, also the lime carbonate content of the mixture which result from various combinations of speed of both motors. These figures have been tabulated with reference to the position of the pointers on the controlling rheostats, which thus becme an arbitrary index of the mixture being made. The tabulations have been made for a combination of the one lime bin with two shale bins, or with all three shale bins.

Determinations for $CaCO_3$ in the raw mixture are made every 10 minutes, and should the result be much higher or lower than the average desired, the chemist can, by reference to his tables, immediately set his rheostats at a combination which will correct the error. It will be seen that this system exercises a very close control over the composition of the raw mixture, which is another important point in cement manufacture. The effect of a change can be determined in from three to five minutes, and no great quantity of material of wrong composition can pass on to the bins; and even this quantity will become more thoroughly mixed in its subsequent treatment and cannot exercise much influence on the entire mixture.

Three 6-ft. x 22.5-ft. Smidth tube mills are used for fine grinding. A screen test of the tube-mill product shows 95 per cent passing a 100-mesh screen, which has been standard practice at this plant for over a year. The tendency is in favor of still finer grinding, however, and the probability is that all will be ground through 100 mesh.

Two Allis-Chalmers rotary kilns, 8 ft. x 125 ft., are fired with powdered coal delivered by blast through two 6-in. feed pipes. The fuel consumption is about 110 lb. of coal per barrel of cement. The total consumption of coal for drying and clinkering is about 130 lb. per barrel. Southern Colorado slack is used, dried in a Ruggles-Coles dryer and ground in Williams and tube mills. The Williams mill grinds about 86 per cent through 20 mesh and 40 per cent through 100 mesh; the tube grinds about 94 per cent through 100 mesh.

The provision for clinker storage is one of the features of this plant, a large area with concrete walls and floor being reserved for this purpose. The clinker is elevated from the kilns and distributed over the storage yard by a link-belt bucket conveyor running on a 100-ft. truss beam supported by steel towers. Beneath the floor are three lateral tunnels connecting with a main tunnel leading to the finishing unit. Hoppers are placed in the floor at suitable intervals along the lines of the tunnels, and through these the clinker is drawn into scale cars in which it is weighed and trammed to the finishing department.

A weighed quantity of gypsum is then added from a scale hopper and the weight checked on the second beam of the tram car. The whole weight is then checked on platform scales. The quantity of gypsum added is variable, but amounts to about 12.5 lb. per barrel of cement, which gives about 1.5 per cent SO_3 in the finished product.

In the finishing end Smidth comminuters and tube mills are used, there being two of each. The comminuters grind 88 per cent through 20 mesh and 40 per cent through 100 mesh; the tubes give a finished product of 98 per cent through 100-mesh and 82 per cent through 200-mesh screens. One of the advantages of clinker storage is noticed in the increased capacity of the machines used for fine grinding. When grinding weathered clinker the tubes will handle up to 20 per cent more material than when fresh clinker is used.

For the storage of finished cement four stock bins of 1250 barrels capacity each are placed in the packing house. Each bin is controlled with the usual tests before being released for packing and each car shipment is similarly tested. The usual routine test is applied at various stages in the process. The cement has an initial set in about 2.5 hours and final set in 5 hours. The boiling test is applied for 24 hours and the pats show no deterioration. A sulphur determination is made on the finished cement twice daily. The following figures show average tensile strength tests:

	24 hr.	7d.	28d.	90d.
Neat	350	650	800	850±
Sand (1:3)	...	290	325	375

The finished cement analyzes in per cents: SiO_2, 22.75; Fe_2O_3, 2.25; Al_2O_3, 8.0; CaO, 64.0; MgO, 0.75 to 1.1; SO_3, 1.6.

The plant is operated by electric power generated at Cañon City 15 miles further west, but the company has under consideration the installation of a gas-producer plant.

Alkali-Proof Cement.

The manufacture of cement which will resist the corroding and disintegrating action of alkali soils and waters, and at the same time develop the strength of standard Portland cement, is a new departure. The third Colorado cement plant mentioned before is that of the Western Cement Products Company, located at Minnequa, Pueblo, Col. The product is an alkali-proof cement, marketed by the Colorado Portland Cement Company, Denver.

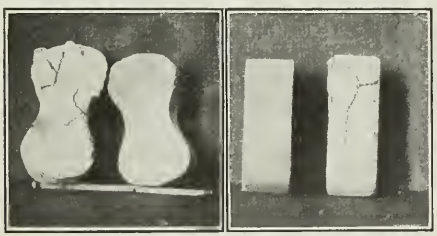


FIG. 4.—EFFECT OF STRONG ALKALI WATER ON STANDARD PORTLAND CEMENT BRIQUETS (TO THE LEFT) AND ALKALI-PROOF PORTLAND CEMENT (TO THE RIGHT).

Reference to the soil survey maps of the United States government will show the wide distribution of surface and subsoil alkali in the western part of the country. Comparatively small areas of the Rocky Mountain region and adjacent plains are free from it, and inasmuch as this region includes some of the vast reclamation projects in which cement and concrete are extensively used the importance of an alkali-resisting cement becomes apparent.

The alkali usually present in Western soils has its origin in disintegrating rocks and in deposits of gypsum, and is mainly calcium sulphate. There are also present magnesium sulphate, and the sulphate, chloride and carbonate of sodium. The action of these salts on structures of ordinary Portland cement is disastrous, causing either partial or total destruction in a comparatively short time. This action may be shown in a laboratory way by making briquets of standard Portland cement and subjecting them to the action of strong water. In Fig. 4 is shown a comparison between briquets of standard Portland and the alkali-proof cement treated in this manner for 60 days. At the end of that time the standard cement briquet is shown to be cracked and on the verge of disintegration, while the briquet of the alkali-resisting cement is sound and strong.

The cause of this action of alkali solutions is found in the composition of standard Portland cement. It contains approximately three times as much lime as silica, iron and alumina, giving an excess of lime over the quantity required for chemical combination with the silica. This excess lime is attacked by the acid radicals of the alkali, forming compounds which, in their subsequent crystallization, expand and disintegrate the mass. Once begun this action continues until, by the processes of solution and mechanical erosion, the structure becomes honeycombed with cracks and pores and finally fails entirely. A peculiar feature of this disintegration is that it may not be apparent on the surface of the work until the central portion has failed. This may be due to the fact that the excess of lime in the surface has early combined with carbon dioxide, forming carbonate which is not attacked by the alkali. Capillary attraction, however, forces the alkali solution into the center, where it accomplishes its work of destruction.

The peculiar qualities of alkali-proof cement which give it resistance to alkali are the following: It contains less than twice as much lime as silica, all of the former being chemically combined and the latter in slight excess. Thus there is no free lime to be attacked by the alkali sulphates. The alkali-proof cement is finer than ordinary Portland, giving maximum strength and minimum permeability, with little or no opportunity for disintegrating solutions to enter the mass. It has a lower specific gravity (2.96) than standard Portland cement (3.1), and hence greater volume for a given weight.

In the physical tests alkali-proof cement shows a slower setting time and develops tensile strength more slowly, but more steadily and for a longer time than standard Portland. A comparison of the two will show about equal strengths at 7 to 10 days, after which the alkali-proof cement will increase more rapidly. The following tabular statement will illustrate this and also show that the new cement is ground much finer than standard products:

	Portland.	Alkali-Proof.		
Fineness.				
Per cent through 100 mesh..	94.9	98.8	98.6	99.1
Per cent through 200 mesh..	77.3	88.6	87.0	88.7
Setting Time.				
Initial	3:22	6:00	5:40	5:40
Final	7:13	7:45	7:55	8:20
Specific Gravity	3.13	2.96	...	...
Tensile Strength.				
Neat.				
24 hr.	408	220	225	170
7 d.	703	600	490	515
28 d.	763	975	865	860
Sand (1:3).				
7 d.	272	285	340	315
28 d.	361	445	540	475

Another interesting manner of testing the alkali-resisting properties of the new cement is to make briquets with sand and varying quantities of gypsum and test their strength from time to time. It will be found that ordinary cement will develop strength as usual for a short time, depending on conditions, and then suddenly deteriorate and ultimately disintegrate. On the other hand, the alkali-resisting cement will show a steady increase in strength over a long period. These points are illustrated in the accompanying curves, Fig. 5, which will require no further explanation.

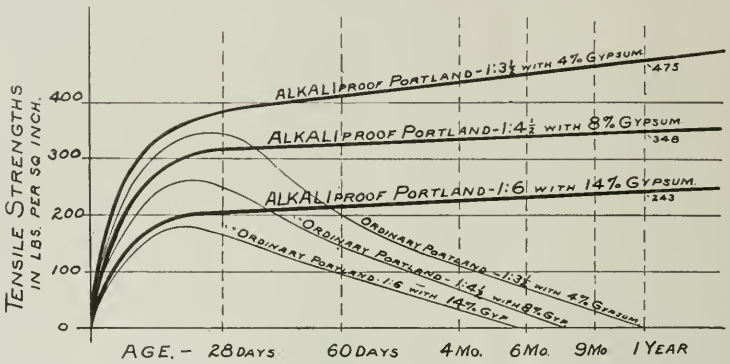


FIG. 5.—EFFECT OF ADDITION OF GYPSUM ON STRENGTH OF STANDARD AND ALKALI-PROOF PORTLAND CEMENT.

Additional comparative data are obtained by exposing briquettes to the action of air, fresh and alkali water for various periods. The alkali solution used in the tests tabulated below is about five times the strength of naturally occurring alkali waters and has the following composition:

	Grains per Gal.
$MgSO_4$	650
$CaSO_4$	900
Na_2CO_3	200
NaCl	200
Na_2SO_4	1300

In testing briquets with this solution it is renewed fortnightly to keep up the strength. Below are the results of tests of four different cements exposed to air, fresh water and alkali solution. The abbreviation "dis" indicates disintegration.

Tensile Strength Tests.

	American Portland.	German Portland.	Special German.	Alkali-Proof.
7 days.				
1:3 air	310	142	112	335
water	267	165	145	342
alkali	175	185	160	342
1:5 air	158	120	90	212
water	160	122	92	195
alkali	128	93	140	208
28 days.				
1:3 air	270	197	205	320
water	280	165	247	390

alkali	342	180	240	405
1:5 air	242	175	185	190
water	140	140	175	245
alkali	dis	127	138	235
90 days.				
1:3 air	455	268	278	367
water	405	320	237	452
alkali	dis	370	310	505
1:5 air	237	165	145	205
water	215	168	170	240
alkali	dis	175	180	245
6 months.				
1:3 air	465	335	270	440
water	415	335	245	490
alkali	dis	dis	215	480
1:5 air	255	175	155	205
water	225	180	175	245
alkali	dis	dis	dis	250

It will be observed that each of the ordinary cements disintegrated at or after 28 days, while the other remained sound. Another peculiar fact noticed from a study of several thousand tests is that just prior to disintegration the standard Portlands show a temporary increase in tensile strength.

Process of Manufacture.

Alkali-proof cement is a mechanical mixture of cement materials, and in this one respect resembles pozzuolan cement. Both cements contain low lime and high silica, but the pozzuolan develops comparatively low strengths while the other is equal or even superior to ordinary Portland in this respect, with the additional value of resisting the attack of aklali. The ingredients used in the alkali-proof mixture are iron blast-furnace slag, standard Portland cement and a mixture, the exact nature of which is not given, but which contains gypsum and other materials. The use of this mixture gives the slag increased hydraulic value and makes the cement strong as well as resistant to alkali.

The mixing of these ingredients is controlled so as to give a cement which will pass standard specifications as to strength, while giving maximum resistance to alkali. The excess lime in the ordinary Portland and the soluble silica in the slag are determined chemically and enough slag is used to provide an excess of silica over the quantity required to hold all of the lime in combination. The ingredients are then intimately mixed and finely ground. The resulting cement does not deteriorate with storage and finds special use in the field mentioned.

The plant at Minnequa has a capacity of 500 barrels daily and is under the direction of a technical staff including Tracy Bartholomew, manager; O. P. Harpel, superintendent, and H. C. Emery, chemist. Slag is obtained from the furnaces of the Colorado Fuel & Iron Company, being granulated at the furnace and dried, screened and ground to about 30 mesh at the cement plant. After being properly mixed the ingredients are ground in a tube mill with Colorado pebbles and "cylpebs" in different compartments of the same tube. The product has the following composition: SiO_2, 32.9; Fe_2O_3 and Al_2O_3, 13.2; CaO, 48.0; MgO, 2.5; SO_3, 1.8; and ignition loss, 0.3 per cent. It is the result of about seven years' study and experiment, during which time it has been subjected to many practical tests.

In conclusion, acknowledgment is due the members of the staff at each of the plants for their courtesy in giving the data presented herewith.

Inside amalgamation of free gold is not practised except under special conditions which seem to warrant its use. In case the outside plates become covered with a slimy film which cannot be easily removed, it may be profitable to apply inside amalgamation, with the frequent addition of mercury.

Manufacture of Carbon Electrodes.

In the London *Electrical Review* of Jan. 20 Mr. William Clacher, F. C. S., gives some details of the manufacture of carbon electrodes as carried out at the works of the British Aluminium Company, Ltd., in Greenock and Kinlochleven.

The author first gives the general methods used at Greenock. At Kinlochleven the methods have been somewhat modified, due to improvement in the plant, but the principles underlying both systems are the same. The difference in favor of the carbon produced at Kinlochleven is due to the new furnace there erected, which is of the regenerative type, giving a much higher temperature than the Greenock tunnel furnace.

The properties essential to a good carbon are: High conductivity, non-liability to crack, efficient contact arrangement between the carbon and the metal conductor.

The characteristics accompanying high conductivity are: (a) a good ringing sound when struck; (b) a high specific gravity and low porosity. The specific gravity of different forms of carbons varies thus:

Calcined anthracite has a specific gravity of.....1.80
Calcined petroleum coke has a specific gravity of....2.00
Kilned retort carbon has a specific gravity of.....2.05

Taking blocks made from these carbons with the same porosity, the retort-carbon block has the highest conductivity; the petroleum-carbon block has the next highest conductivity, and the anthracite-carbon block the lowest. This refers to anthracite calcined at 1000° C., not at 3000° C.—that is, non-graphitized anthracite.

The outline of the process for making the carbon is as follows:

1. The petroleum coke is first calcined to remove volatile matter and increase the specific gravity and conductivity.
2. The calcined material is pulverized.
3. The pulverized material is mixed with the binding medium in a steam-jacketed mixer.
4. The mixed mass is molded.
5. The block is kilned in the furnace, the kilning taking five days. After this the blocks are cooled, brushed, and stocked ready for use.

The petroleum coke is obtained in the distillation of shale oil; it is the residue left in the iron retorts. There is left in it from 5 per cent to 13 per cent of volatile hydrocarbons, which are not driven off with the heavy oils. There is also a small amount of inorganic material averaging 2 per cent. Most of the latter is in the harder parts; the softer and more porous parts are almost ashless. Mr. Clacher has analyzed pieces with a difference of over 5 per cent between the separate portions. Surface coloring he has found to indicate the presence of inorganic material; such carbons are none the less considered of lower grade.

The pieces as they come from the oil works are usually of large size. This is good for stacking purposes. There is at the same time a percentage of smalls and dust, varying with the quality of the coke, the softer coke having by far the largest amount. This is a serious matter to the Aluminium company; as the small dust chokes the calciners and is not easily worked away, only a certain amount can be added with each charge of coke.

The calciners are vertical rectangular brickwork chimney-like constructions; the coke is put in at the top and the calcined material drawn out with long rakes at the bottom. The charges are given every two hours, and a half charge is drawn every hour; 2 cwt. (101.6 kg or 224 lb.) was the charge in the Greenock calciners. The time the coke takes to travel from the top to the bottom is five hours.

The temperature in the calciner is approximately 2000° C. at its hottest part. The calcining is a very costly and, as at present carried out, wasteful process, for 30 per cent of the coke put in is burned away during the process; the volatile

matter in the coke before calcining averages 8 per cent, so that 22 per cent of carbon is used to raise the temperature to 2000° C., representing fuel at $17.50 per ton—considerably more costly than the best smokeless coal.

Calcination has been carried out in horizontal retorts; these are more expensive at first, but they are much more economical with the coke, 20 per cent being the loss usually found with them. In cases where the calcination was effected in a current of producer gas the loss was little more than the percentage of volatile matters in the material; 15 per cent was the figure obtained from 10 per cent volatile in the coke, and the temperature is nothing like as high as in the vertical retorts, not exceeding 1000° C. The reports on anodes made from coke calcined thus were of a satisfactory character; the specific gravity of the carbon was slightly less than that calcined in the vertical calciners, 1.95 being obtained.

Calcined coke was also once made by packing the blocks before entering the tunnel furnace with it, and then passing it through; the coke swelled up and the blocks thus packed were unusable.

In the vertical calciners experiments were begun before Mr. Clacher left with the object of saving the excessive loss by passing producer gas into the calciners and using that in place of the coke to get the required temperature. The work was not far advanced when he left, and he is not aware that anything on these lines has been worked at by the Kinlochleven staff.

After calcination the coke at Greenock contained 0.7 per cent of inorganic material, consisting of 0.5 per cent silica, 0.15 per cent oxides of iron and alumina, and 0.05 per cent soluble sodium salts.

At Kinlochleven the arrangements of the stores for the oil coke and the transporting and grinding machinery have been totally altered. In addition, breakers to crush the coke to the size required for calcining have been erected; vertical rotary crushers and high-speed vertical grinding mills are used, and the sieving machinery is more up to date. At Greenock the breaking of the coke was done by men with hammers.

The grinding of the calcined oil coke is done in two stages: (1) An initial crushing to pass through a ⅜-in. mesh; (2) a grinding to get the correct gristing of the powder. The gristing is an exceedingly important item; the amount of fine dust and large must be properly proportioned to get the block with the highest specific gravity and least porosity.

A good gristing is: Through 100-mesh, 40; 60-mesh, 15; 30-mesh, 20; 16-mesh, 15; between 16-mesh and 8-mesh, 10.

The cost of grinding at Greenock was from 75 cents to $1 per ton; there was a large loss of carbon in the open pan-mill and the machine for the finer grinding amounting to 7 per cent of the total. This loss is done away with when using the new machinery.

The supply of petroleum coke, a by-product in the paraffin industry, is dependent on the quantity of shale oil distilled, and this is strictly limited in amount.

In view of an expected increase in the demand for aluminium and a consequent greater need for a good carbon, a search was made for likely sources. Of retort carbon, which is used for making the furnace linings, the supply is limited to the quantity of coal gas made. Of anthracite there is an unlimited store, and provided the inorganic contents were within the limits of the supply of calcined oil coke, this was considered a possibly valuable article.

The limits of impurity allowed were 0.5 per cent silica and 0.2 per cent iron oxide; this was the average analysis of the ash contents and the kilned blocks at the time, in the year 1904.

A great amount of work was performed on this for a period extending over three years. After that the work was stopped for a time. After Mr. Clacher had left the company he continued the experimental work, and after overcoming some of the difficulties left unsolved in the Greenock factory, he patented the process (British patent No. 2071, 1909).

The following forms of carbon were treated at Greenock: 1. The returned blocks from the Foyers furnaces. 2. Retort carbon. 3. Anthracite, Scotch and Welsh. 4. Bituminous coal. 5. Graphite.

The work on (5) was carried on exclusively after Mr. Clacher had left the company.

1. The returned blocks from Foyers contained approximately 2 per cent inorganic constituents, of which a little over 1 per cent was silica and iron oxide; this was crushed to pass through an 8-mesh sieve, with the usual gristing and, after treating, the inorganic contests were reduced to 0.5 per cent. An analysis of the untreated and treated material is given as follows:

	Before.	After.
Ash	1.56%	0.53%
Soluble	0.33	0.17
Silica	0.46	0.14
Iron oxide	0.26	0.14
Oxide of aluminium	0.39	.
Difference (CaO, MgO, SO_3)	0.12	...

2. Gas retort carbon as used at Greenock contained from 2 per cent to 4½ per cent of inorganic constituents. Crushed to pass through an eight mesh, and of the usual grist, the inorganic was reduced from 3 per cent to 0.75 per cent. Analyses:

	Before.	After.
Ash	2.35%	0.68%
Soluble	0.10	...
Silica	0.91	0.20
Iron oxide	0.62	0.26
Oxide of aluminium	0.41	0.08
Lime	0.17	...
Magnesia	nil	.
Sulphur trioxide	0.11	..
Difference	0.03	...

Another analysis:

	Before.	After.
Ash	2.87%	0.71%
Soluble	0.08	...
Silica	1.13	0.16
Iron oxide	0.86	0.19
Oxide of aluminium	0.60	0.36[1]

3. Scotch Anthracite.—The seams tested contained from 4 per cent to 12 per cent inorganic. Analyses are given of some purified; in this case of somewhat smaller grist, the whole passing a 16 sieve:

	Before.	After.
Ash	9.24%	1.81%
Silica	5.00	0.83
Iron oxide	1.06	0.22
Oxide of aluminium	2.02	...
Lime, magnesia sulphate and soluble	1.16	0.19

	Before.	After.
Ash	4.99%	1.08%
Silica	2.47	0.49
Iron oxide	0.46	0.08
Oxide of aluminium	1.69	0.19
Lime	0.09	
Magnesia	0.05	
Sulphur trioxide	0.08	0.32
Difference	0.15	

Ground to pass through a 60 mesh the totals were reduced in proportion. Analyses:

	Before.	After.
Ash	6.50%	1.25%
Soluble	0.17	included in diff.
Silica	3.55	0.38
Oxide of iron	0.88	0.05
Oxide of aluminium	1.55	0.31
Difference	0.35	0.51

[1] Soluble and oxide of aluminium.

With Welsh anthracite he got from a seam analyzed:

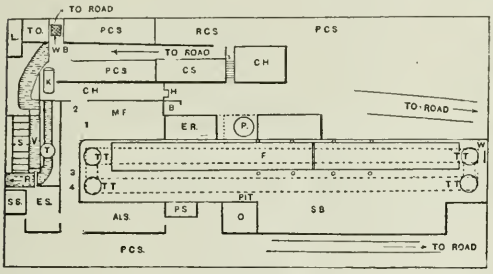
	Before.	After.
Ash	2.08%	0.57%
Silica	0.78	0.20
Oxide of iron	0.32	0.12
Oxide of aluminium	0.43	0.12
Soluble and difference	0.53	0.13

The material was milled to pass a 16 mesh.

4. With a good bituminous coal containing 3 per cent of ash the silica was reduced to less than 1 per cent. The iron oxide was not removed, and work was discontinued on that account.

The process used was a hot digestion with a strong caustic solution and the removal of the silicates, sparingly soluble in caustic, with a weak acid solution—iron oxide being subsequently extracted.

The accompanying diagram is a sketch plan of the general arrangement of the carbon factory; *PCS* is the petroleum coke stock; *RCS* retort carbon stock; *CH*, calcining house; *GH*, grinding house; *H*, hoist; *MF*, molding floor; *F*, tunnel furnace; *TT*, turntables; *W*, winch; *SB*, stock finished blocks; *PS*, stock pitch; *CS*, coal; *B*, boiler; *ER*, engine-room;



SKETCH PLAN, SHOWING GENERAL ARRANGEMENT OF CARBON FACTORY.

P, producer; *L*, laboratory; *TO*, time office; *ES*, engineers' shop; *SS*, smiths' shops; *ALS*, aluminium stock; *O*, offices; *S*, flight of stairs; *V*, overflow; *R* runaway from turbine; *T*, water turbine; *WB*, weigh bridge; *K*, kilns for drying carbon. 1, 2, 3 and 4 represent the position of the machines, etc., shown in Figs. 1 to 4 respectively.

From the grinding mills the fine powder is taken away to the store hoppers, from which it is transferred to the Werner and Pfleiderer mixing machines; these are seen in Fig. 2 and are of the steam-jacketed kneading type, with reversing gear for changing the direction of motion. The temperature of the mixture when ready for discharging is 90° C. In the mixers shown the tipping is effected by means of an upright screw; in the large new mixers for Kinlochleven a worm quadrant and screw is used in place of the upright screw. The mixers in the illustration hold a charge of 300 lb. of mixture.

The hopper for weighing the carbon is seen over the right-hand mixer and the pitch-weighing tank over the left-hand mixer. The amount of pitch used is 22 per cent of the mixture; it is a medium hard brand and has volatile constituents to the extent of 50 per cent, as volatile is determined quickly; only 40 per cent volatile goes away during the slow kilning.

The tar used for making the linings for the electric furnace is much softer, containing up to 60 per cent volatile as determined by the quick method. This lining for the furnaces was made at the mixers in Greenock, bagged, and sent to Foyers, where it was heated and rammed in the furnaces at the side and bottom and then baked with current passed through before charging the furnaces with the electrolyte. The life of the linings varied from a few days to months. Latterly the mixture for the furnaces was made at the electric reduction works, saving the trouble and expense of the reheating.

For making the tar to the required degree Mr. Clacher used an instrument for measuring its fluid quality at the temperature of mixing (115° C.), the time a given weight took to run out of an orifice giving the indication required. He found tests so performed were usually in agreement with the volatile determination and a test of the fluid nature is of more value as a criterion for the binding material in the tar than is the percentage of volatile matter in the semi-liquid.

The bending point of a slab of pitch 3 in. long and ½ in square is a very good test for the pitch. This test is certain;

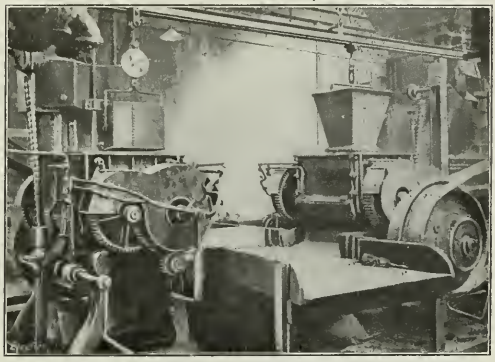


FIG. 1.—MIXING MACHINES.

the determination of the volatile content is anything but certain. Mr. Clacher has found that when the greatest care is taken the limit of accuracy of volatile determination is such that a determination of that kind is not a standard of value.

Fig. 2 shows the arrangement of the mixing and molding floor. On the right-hand side are two melted-pitch tanks; they are wrought-iron vessels, steam heated by internal coils. The pitch is lifted up by means of a small windlass and emptied into the tanks as required. A third tank was afterward added; this was a steam-jacketed vessel and worked more satisfac-

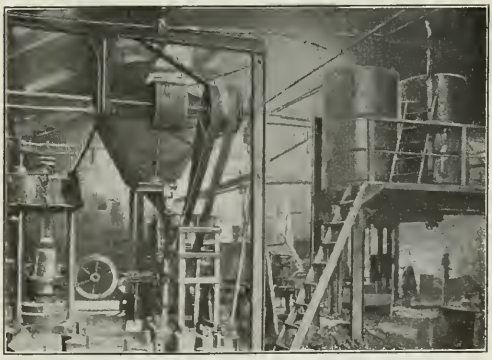


FIG. 2.—MIXERS AND PITCH TANKS.

torily than the two shown; the leakage of the joints in the coils caused frothing and overflowing of the tanks heated with the internal coils. In the center are the two mixers and the store hopper containing the carbon; on the left hand are the hydraulic molding machine and some of the molded blocks.

The mixture when tipped from the mixer to the platform is transferred in scoops to the container of the mold, which is in the center of the machine. When the mold is full the water valve is opened, the container rises and the block is formed.

When the gage registers 1 ton to the square inch the valve is shut and the runaway valve opened, the container lifted and the block discharged; it is taken away and put on the floor, there to remain for a day to harden previously to being packed in the trucks for putting through the furnace. Half a ton to the square inch will form a good block. One ton is usual, and as much as 4 tons has been used.

It is not possible to make a good block with mixture that is cool, and arrangements are made to work with as little stock of mixture as possible. The capacity of the machine illustrated was 40 blocks an hour. The Kinlochleven machine is more complicated and is designed for a larger output.

The blocks are 10 in. square, with cutaway corners, and are

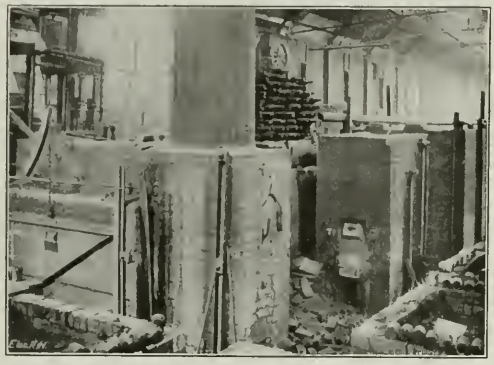


FIG. 3.—HEATING END OF FURNACE.

12 in. high. Those shown in the illustration are flat topped. This has been improved and for some time all the blocks have been made with the top rounded. The top of the block, efficient only as a means of making contact, is thus reduced in weight, and consequently a rounded-top block of equal weight to a square-top block has more carbon usable for the reduction of the alumina

The contacts seen in the blocks are of mild steel and are molded in the blocks, the internal end being forked; the forks are thoroughly cleaned and coated with pitch previously to being put in the clamp for molding. It is noticeable that the contacts are not central in the block. When the contact is made at the furnaces the point of suspension becomes central and the blocks hand vertically without strain.

The form of contact shown is the best between the carbon and the metal; but these are very expensive to have in stock. A large number is required, and the wear and tear on them is great. The later forms of contact are made by screwing the metal into a molded thread in the block; the number of contacts required in stock is much lessened and the wear and tear comparatively inappreciable.

The weight of a block before kilning averages 70 lb. with the claw; after kilning, 66 lb. with the claw. The claw weighs 8½ lb. The loss of 4 lb. in the block corresponds to 8 per cent on the block and 40 per cent on the pitch used. The arrangement of the pitch tanks at Kinlochleven was much altered and in no way resembles the Greenock arrangement. This was on account of the much larger output that was expected.

When made the blocks are stored till they are hardened somewhat and are then placed in the trucks for calcining. At Greenock the blocks were stored on the molding floor; a few days' stock was kept.

The trucks seen in Figs. 3 and 4 are oblong and made of firebrick; they are mounted on bogies. The blocks are placed, 42 in number, in a truck and are packed with the fine ashes free from dust; the metal contacts are covered with fireclay covers and the fireclay covers are completely hidden with the ashes. The truck is then pushed into the furnace and at the same time a truck with the kilned blocks is pushed out of the other end of the tunnel by the same movement, which moves each truck in the furnace one stage on. A hydraulic ram is used for this purpose. As soon as the truck with the unkilned blocks is in and the truck with the kilned blocks out the iron furnace doors are dropped and the furnace is at normal again.

Producer gas is the fuel used for heating the furnace, this being made in a Mason producer; the coal found best for that purpose was a Lanarkshire singles containing 3 per cent to 5 per cent of inorganic and 30 per cent volatile constituents; the analysis of the gas made is:

Carbon dioxide	4.0
Carbon monoxide	22.0
Hydrogen	16.0
Methane	4.0
Oxygen	2.0
Nitrogen	52.0
	100.0

The flues from the producer to the furnace require cleaning once every eight days; the regulation of the quality and quantity of the gas also requires attention. When Mr. Clacher went to Greenock the producer was consuming 40 tons of singles a week. By regulating the feed and the steam and the air he got the consumption down to 27 tons per week. At the same time a more constant temperature was produced.

The furnace is seen in Figs. 3 and 4. It is 200 ft. long. On either side are four gas valves. These are regulated to give an even temperature over the middle of the furnace. The escaping gases heat the incoming trucks and the outgoing truck is somewhat cooled before being ejected. The maximum temperature at the top of the truck is usually 1150°, at the bottom of the truck 700° C. The latter is the temperature to which the bottoms of the anodes made at Greenock are kilned. This is

FIG. 4.—TUNNEL FURNACE AND ROW OF TRUCKS.

doubtless a disadvantage for the purpose to which the furnace is now applied—the calcination of alumina to be used at the electric furnaces.

The furnace at Kinlochleven is built of firebrick and is of the type used at Witton, in that it is regenerative, the chambers in which the articles kilned are cooling being used to heat the air for the combustion in the hot chambers; the temperature normally attained as a maximum is said to be 1400° C., and the

ame time is taken for the cycle of operations as at the Green-ck factory—5½ days. The chambers are filled with the boxes ontaining the blocks, which are packed as at Greenock with ne ashes; they are stacked one above the other in the cham-er. The boxes are of fireclay and are made at Kinlochleven. When the chamber is filled to the requisite capacity the open-ng is sealed with brickwork, and this chamber is then made he last in the circuit of the gases passing through it previous o entering the chimney flues. The accompanying plan, Fig. 5, ives an idea of the arrangement of the chambers.

The blocks kilned in this furnace are said to be much more fficient at the electric furnaces than the Greenock kilned ones.

At Foyers during the past few years the Greenock blocks ave a yield of 1 lb. aluminium for 0.7 lb. carbon used; the ost that is expected is 0.64 lb.

A great amount of trouble has sometimes been caused by the urning of the blocks by the air at the electric furnaces; this

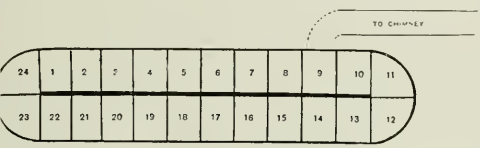


Fig. 5.—Plan of Regenerative Furnace.

erious reduction of the efficiency of the carbon was gone into, nd many coatings were given the blocks to prevent the oxida-ion, whitewash among others. Mr. Clacher has reason to hink this difficulty is solved. Cracking of the blocks at the lectric furnace is a trouble due to the block not being to tandard. A block should last from 100 hours to 140 hours, and vhen conditions are normal in the furnace the time factor is nost valuable. When blocks last less than 100 hours they are ot classed good.

The specific gravity of the Greenock blocks was 1.63 after ilning, and the porosity 20 per cent; different parts of the Greenock blocks had often different porosities and considering he difference in temperature between 1150° and 700°, the tem-eratures at the top and floor of the truck, this is not to be vondered at. Before kilning the specific gravity was 1.53.

Mr. Clacher experimented with the making of blocks from a ery heavy pitch with less than 40 per cent volatile matter. Blocks thus made in a gas-heated mixer were harder than the sual block; their conductivity he did not have tested.

[In connection with this article it may be interesting to refer o some progress which is reported as having recently been nade in the manufacture of large amorphous carbon elec-rodes. It is reported by reliable correspondents that in Euro-ean electric steel furnace plants amorphous carbon electrodes n sizes up to 600 mm (24 in.) diameter are in use and that no roubles due to cracking of electrodes are experienced.]

The Arbuckle process of ore treatment as used in South Africa is thus described by H. A. White, in a paper read before the S. A. Association for the Advancement of Science: It nvolves the treatment of the pulp as a whole without sepa-rating sand from slime, and thus solves that difficulty by the nethod of evasion. The pulp, mixed with about three times ts weight of weak KCN solution, is agitated by an air lift in deep tanks with conical bottoms. After about 20 hours' agita-tion, the whole pulp is transferred to a separating apparatus, where the solid portion, with about 25 per cent moisture, is drawn off for further washing, and the clear solution sent to the precipitation tanks. The drawback lies in the agitation of the whole pulp for the entire period, when perhaps the slime portion would not require so long an agitation. Nevertheless, it is possible that owing to decrease in working costs in larger units, greater skill in handling material, and other important reasons, this method, with modifications, may become of im-portance in South Africa.

A New Form of Gauze Electrode for the Rapid Electrolytic Determination of Lead and Copper.

BY R. C. BENNER.

The present trend for lead and copper determinations being toward the use of electrolytic methods, any increase in rapidity, simplification or lightening of labor in the use of this method is of vital importance.

When the interfering elements are removed and the deposition is carefully made, the electrolytic determination of copper is generally recognized as the standard, and is, as well, the most accurate of the various methods employed in technical work. The determination of lead by this method until recently has been more likely to error than that of copper, as the PbO_2 deposited was likely to have a varying composition depending greatly on the conditions of precipitation.

Every person who has made a quantitative, electrolytical determination of lead has found necessary the use of some factor other than the theoretical. This has made the determination of an arbitrary factor necessary in each case. When the peroxide of lead is deposited on the inside of a platinum dish it is possible to ignite the PbO_2, changing it to PbO, in which form it can be weighed with faultless results.[1]

When the deposition is made on the gauze electrode it is impossible to ignite over the Bunsen burner, as some of the PbO is volatilized, or some of the PbO_2 remains undecomposed,

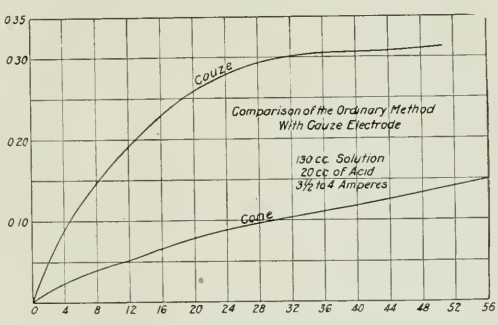


Fig. 1.—Comparison of ordinary electroanalytic method with the use of gauze electrode.

giving low or high results respectively. This may be obviated with the gauze electrode by igniting in a muffle below a red heat (500° C.) or in a small electric furnace made for the purpose.

Recently Sand[2] has been enabled to deposit the peroxide of lead in such a condition that the error caused by the use of the theoretical factor can be neglected when the amounts of lead found in a 0.5 gram sample of ore are taken. This he accomplished by means of a rotating cathode and gauze anode, depositing the lead from a strong nitric acid solution, free from NH_4NO_3 and the oxides of nitrogen, at a temperature just below the boiling point (94-97° C.), using a current of 5 amperes. When the deposition is made in this manner the use of the factor 0.8651 gives correct results. This factor is so near the theoretical (0.866) that the latter can be used without error larger than other errors involved.

The simplest form of apparatus in use at the present time consists of the two ordinary platinum electrodes, with which it takes from five to ten hours to deposit the amount of copper found in 0.5 gram of ore. The shapes of these electrodes have been modified indefinitely but serve essentially the same purpose. The rapid precipitation of the various elements in such a form that they can be weighed accurately is possible with

[1]Treadwell Analytical Chemistry, New York, 1904, Vol. 2, p. 140.
[2]Chemical News, 100-269.

several different forms of apparatus, all of which are dependent on the necessity of agitating the electrolyte, so that as long as there is any metal in the solution some of the metallic ions will be touching the cathode at any instant and be deposited.

In order to accomplish this, Smith, at University of Pennsylvania, rotated the anode, while Gooch, at Yale, rotated the cathode. Both of these forms, however, while giving a rapid method for obtaining excellent results, possess the great disadvantage of requiring more or less complicated pieces of apparatus with which to agitate the electrolyte, and more attention than the ordinary apparatus in use with the slower methods.

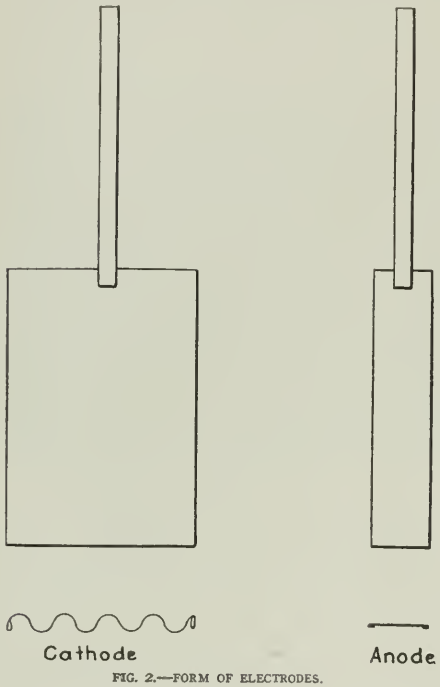


FIG. 2.—FORM OF ELECTRODES.

The advantage of rapidity for technical laboratories is more than counteracted by the disadvantage presented in this complexity. The need, then, of apparatus which combines the advantages of both and eliminates their disadvantages, becomes evident. It recently has been possible to accomplish this to a certain extent by the use of a cathode made of platinum wire gauze.[3]

The gauze electrode makes possible the use of stationary electrodes as simple as the most simple foil electrodes, at the same time allowing the deposition of copper and lead in a much shorter period than in the case of the ordinary electrodes. This will be seen by examination of the curves, Fig. 1.[4]

The deposition on the gauze electrode, even though it can be made much more quickly than by the old method, cannot equal the rapidity of the rotating cathode or anode. The nearest approach to the rapidity of the gauze electrode when stationary electrodes are used is found in the device of Messrs. Guess and Haultain. This consists in the use of very light electrodes, 1.2 grams in weight, and the precipitation of small quantities of copper (less than 0.1?) in the presence of a nitro compound. Using a current of 1.5 amp it takes from 1.5 to 3 hours to make a deposition. The apparatus is very compact and handy for this work and has been put into use by many assayers through the western United States.

As this apparatus has such an extended use it seemed desirable to design an electrode which would be suited for use in connection with it and yet possess some of the advantages of the cylindrical gauze electrode employed for our previous work on lead and copper. The electrode designed and used in our experiments is shown in Fig. 2 and consists of sand-

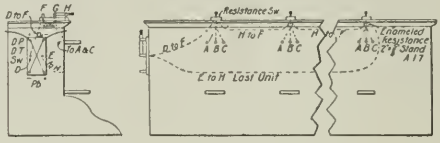


FIG. 3a.—WIRING ARRANGEMENT.

blasted gauze. These electrodes can be connected by the method shown in Fig. 3, *a*, *b*, and *c*, which is used in the Guess-Haultain cabinets, or by the arrangement shown in Fig. 4, used by Dr. W. H. Ross and the author for general electrolytic work. By this arrangement it is possible to use each deposition cell independently. In Fig. 4, R_1, R_2, R_3, are small resistance boards of about 10 ohms, made of nichrome resis-

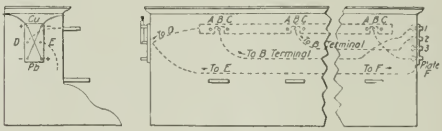


FIG. 3b.—WIRING ARRANGEMENT.

tance wire, manufactured by the Driver Harris Wire Co., of Harrison, N. J. This is the most satisfactory resistance wire with which the author has had experience. Each deposition cell has its resistance box for independent regulation. The amount of current for the system as a whole is regulated according to circumstances. The switches, S_1, S_2, S_3, are the main switches for opening and closing the circuit for the

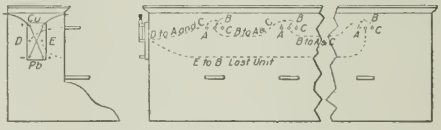


FIG. 3c.—WIRING ARRANGEMENT.

operation of each cell. When a cell (C_1) is in operation alone, the other switches S_2 and S_3 remain open when S_1 is closed. AS_1 is the ammeter switch. By closing AS_1 and opening S_1 we get the ammeter reading for that particular cell. Closing VS_1 we get the voltmeter reading. The terminals C_1, C_2, C_3, are those of the Guess-Haultain apparatus with very slight modification, as shown in Fig. 5. These were slightly more satisfactory, due to the fact that our electrodes had much thicker handles than those used with the apparatus mentioned. The voltmeter and ammeter readings may be obtained in the same manner for any cell when any number of cells are in operation.

Process for Lead Ores.[5]

Since it is possible to remove the sulphur from galena, as well as from some other sulphides, in the form of H_2S by warming with HCl (2-1), it is convenient to use the following method for those galena ores containing no interfering sul-

[3] Stoddard, J. Am. Chem. Soc., 31, 385 (1909); Palmer and Palmer, Trans. Am. Electrochem. Soc., 15, 489 (1909); Benner, J. Ind. and Eng. Chem., Vol. 2, No. 5 (May, 1910); Benner, Ibid., Vol. 2, No. 8, (Aug., 1910); Frary, 17th Gen. Meet. Am. Electrochem. Soc., Pittsburg, May 5-7, 1910.

[4] Palmer and Palmer. Ibid.

[5] J. Ind. and Eng. Chem., Vol. 2, No. 8.

phides. Weigh out from 0.5 to 1.0 gram of finely pulverized ore, warm with 15 to 20 cc. HCl (2-1) until completely disintegrated and all of the sulphur evolved as hydrogen sulphide, but not to such concentration that lead chloride crystallizes out, causing bumping. Then add 20 cc. concentrated HNO_3 and boil until all of the HCl is expelled and the nitric acid has a volume of about 10 cc. Dilute to 75 cc. and electrolyze in the usual manner. Those ores containing sulphides which render the preceding method impossible are disintegrated with HCl and HNO_3, then treated with 15 cc. H_2SO_4, and the solution evaporated until heavy white fumes of sulphuric anhydride are evolved. After cooling, 50 cc. of water are added

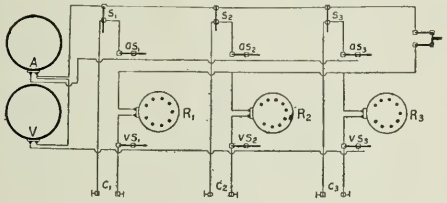


FIG. 4.—ARRANGEMENT OF AUTHOR.

and the solution boiled until all of the basic iron sulphate is dissolved. Dilute the solution to 20 cc., filter and wash free from sulphuric acid, washing by decantation as much as possible. The lead sulphate is then washed, with the smallest possible amount of water from the filter into the beaker containing the bulk of the sulphate washed by decantation. Pour 10 cc. of a warm saturated solution of ammonium carbonate through the filter, receiving it in the beaker containing the sulphate of lead, and warm until all of the sulphate is converted to the carbonate. The lead carbonate is then filtered through the same paper, washed with water until free from ammonium carbonate, dissolved in 40 cc. HNO_3 (1-3), the filter paper washed free from lead, and the solution electrolyzed in the usual way. The following results on a lead ore containing copper show the accuracy of the method.

Analysis of the ore by ordinary methods: Ore No. 1, 2.7 per cent copper and 21.5 per cent lead; ore No. 2, 2.28 per cent copper and 22.2 per cent lead. Electrolytic determination of lead: No. 1, 21.52, 21.51, 21.59; No. 2, 22.2, 22.06 per cent.

Where it is desired to do away with the use of the factor, and at the same time use the electrolytic method of determination, it is possible to do so by igniting the gauze electrode on which the peroxide of lead has been deposited, in a muffle or small electric oven at a temperature of about 500° C. The following table shows the accuracy of this modification of the electrolytic method when a C. P. lead solution is used:

Lead taken.	Lead found.
0.2509	0.2511
0.2500	0.2506
0.5018	0.5035
0.5018	0.5026
0.7527	0.7518

The lead peroxide can be readily removed from the gauze electrode by means of dilute nitric acid containing a small amount of potassium nitrite.

Process for Copper Ores.[6]

Dissolve from 0.5 to 1.0 gram of the ore, containing none of the interfering elements, in nitric or nitro-hydrochloric acid. When aqua regia is used, or if lead is present, sulphuric acid is added and evaporation continued until the heavy white fumes of sulphuric anhydride are evolved. The excess of sulphuric acid is neutralized with ammonia, and 3 cc. of concentrated nitric acid added. Providing no sulphuric acid is added it is necessary to evaporate to only 3 cc. In either case the solution is diluted to 75 cc., transferred to a tall battery beaker without filtering (if the residue does not settle readily it is better to filter), and the copper deposited by means of a current of 5 to 8 amperes and 3 to 4 volts. If the deposit is black or dark colored it is dissolved in 3 cc. nitric acid and redeposited in the same way. This redeposition is, as a rule, easier and quicker than filtering. Results are as accurate as with the older method if proper care is taken.

[6]J. Ind. & Eng. Chem., Vol. 2, No. 5.

Process for Copper Slags.

In case of slags and other low grade material it is necessary to select a method of disintegration suited to the particular material in hand. Slags, which always give large quantities of gelatinous silica and contain small percentages of copper, give the most accurate results when disintegrated with nitric acid, followed by hydrofluoric and sulphuric acids. After evaporation until heavy white fumes are evolved the analysis is finished as for ores. Redeposition is more frequently necessary with low grade material than with high, but in all cases it is more accurate and rapid than filtering.

Elements which interfere, and at the same time are likely to occur in copper ores, are arsenic, antimony, lead and bismuth. The separation of antimony and bismuth from copper was tried under the preceding conditions, but without success. Arsenic in small quantities will not be deposited with the copper, but in amounts about equivalent to the copper present begins to be deposited with the last of that metal. This can easily be removed by dissolving the deposited copper and reprecipitating under the same conditions as before.

The removal of the interfering elements by chemical means can be accomplished by the following method: The ore is disintegrated with nitric and hydrochloric acids, and then treated with 5 or 6 cc. of sulphuric acid and evaporated until heavy white fumes are evolved. Cool and dilute with water to about 150 cc. boiling to insure complete solution of the basic sulphate of iron, and adding a few drops of hydrochloric acid if silver is present. Filter and wash with water containing a little sulphuric acid. The filtrate is now heated to boiling and saturated with hydrogen sulphide, allowing the solution to cool during the process. The precipitated copper is filtered and washed with water containing hydrogen sulphide. Rinse the precipitated sulphides from the filter paper into a beaker, using as little water as possible, and warm with a little colorless sodium sulphide to remove antimony and arsenic, and again filter through the same paper, using wash water containing sodium sulphide. Bismuth is now removed from the copper by warming the mixed sulphides with a solution of potassium cyanide made alkaline with ammonia. This dissolves the copper, leaving the bismuth. Filter, and boil the copper solution with nitric acid to destroy the cyanide, and electrolyze in the usual manner.

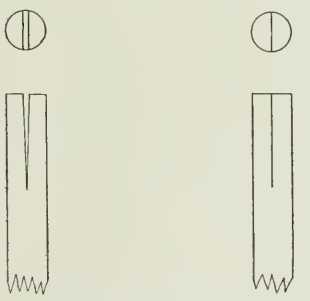


FIG. 5.—TERMINALS.

Copper Ores Containing Large Amounts of Iron.

When copper ores contain large amounts of iron it is impossible to deposit the copper successfully from a nitric acid electrolyte, as frequently before the deposition is completed a resolution of the copper often takes place from the combined action of the iron salts and the nitric acid. This phenomenon is well described by Classen in his work on electrochemical analysis, as observed by him and others.[7, 8] "Three grams of

[7]Fairlie and Boon. Electrochem. & Met. Ind., Vol. 6, p. 58.
[8]J. Am. Chem. Soc., Vol. 32, No. 10.

a sample of copper containing from 40 to 50 per cent iron were taken, dissolved, diluted to 150 cc. and electrolyzed in the presence of 3 cc. of nitric acid, with a current of three amperes. As the precipitation of the copper began the solution took on a brown color. As it proceeded suddenly the solution was decolorized together with the solution of the copper, after which the electrolyte gave only a slight test for ferrous iron."

"This is explained as follows: During the electrolysis the hydrogen liberated reduces the iron to ferrous form, which with the decomposition products of nitric acid, forms the unstable compound $xNOyFeSO_4$. As long as the reduction of the iron takes place the precipitation of the copper continues, but as soon as this reaction reaches an equilibrium under the existing conditions, the unstable compounds suddenly decompose with the resolution of the copper and the oxidation of the iron, which is indicated by the decolorization of the solution."

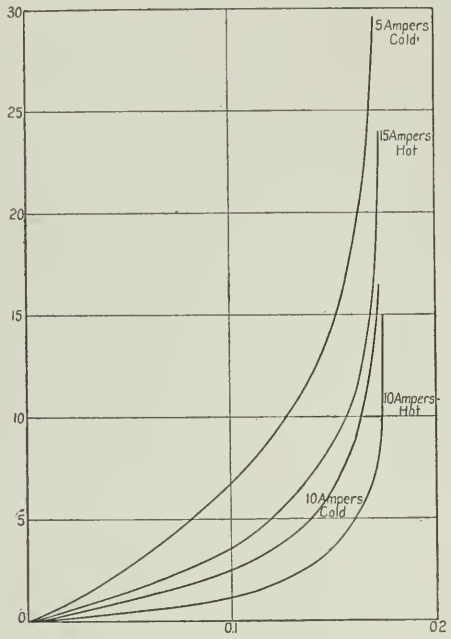


FIG. 6.—DEPOSITION CURVES FROM HOT AND COLD SOLUTIONS.

Blaisdal and Harrison[8] have shown that it is possible to prevent the formation of nitrous acid and consequent solution of copper by the addition of 5 grams of urea to the acid solution of copper before electrolysis.

Figs. 6 and 7 give the rate at which lead and copper are precipitated with the electrode shown in Fig. 2 and under the conditions used for the precipitation of these elements in ores. Each point determined on the curves was made as a separate determination with a new solution containing the same amount of copper or lead in each case.

The following points may well be observed in the electrolytic determination of these metals:

1. Always keep the cathode completely covered by the electrolyte.

2. Care must be taken not to stop the electrolysis until the precipitation is complete. The time of deposition varies with different electrodes, volume of solution, amount of metal in solution, etc., even if the current is the same in each case. These facts are especially true with rapid methods and with the gauze electrode in particular.

3. Dissolve and redeposit all dark colored copper precipitates, as frequently they give high results. This may be due to the oxidation of the copper, or to the presence of interfering elements.

4. Use great care in adjusting the electrodes so that the current density is as nearly the same as possible at all points. When the density is higher at some points than at others it is likely to cause decolorization of the copper at these points.

5. In selecting gauze electrodes choose those which will allow the best circulation of the electrolyte and as large a surface as is convenient.

6. Under some conditions the deposited copper is likely to be spongy but will give good results if the deposit is not brushed off. Handle the electrodes in this case with the same care used with those on which lead peroxide has been deposited.

7. Do not try to deposit more than 0.2 to 0.3 gram of copper or lead, as such deposits are not as good as when small amounts are used.

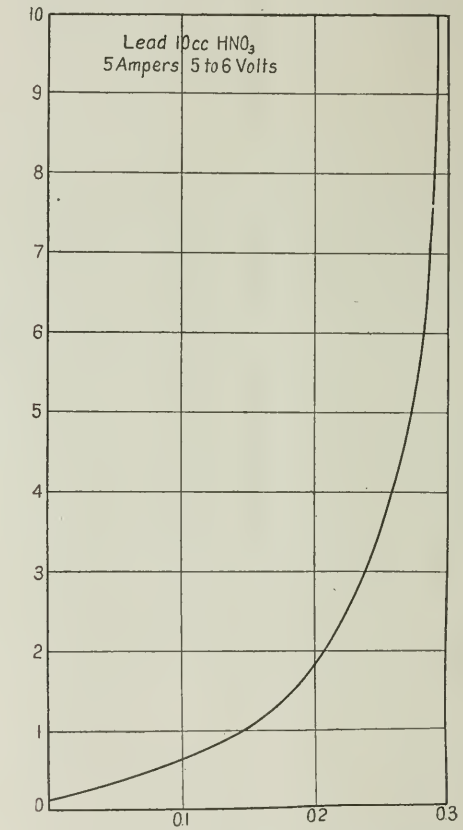


FIG. 7.—DEPOSITION OF LEAD. (ABSCISSAS REPRESENTING GRAMS PbO_2).

8. Do not allow the electrolyte to boil away, as the concentration of the acid is likely to cause the solution of the metal.

University of Arizona, Tucson.

American Society of Mechanical Engineers.—The sixty-third meeting of the American Society of Mechanical Engineers will be held in Pittsburgh, Pa., from May 30 to June 2. This is the first meeting of the society in Pittsburgh since 1884. Col. E. D. Meier, of St. Louis, is president this year.

Chemical Engineering in the Manufacture of Metallic-Filament Lamps.

It is unnecessary to emphasize at this place that the progress in modern lighting methods has been rendered possible essentially by the work of the chemist. This is true both for electric incandescent and arc lamps as well as for gas lamps.

In the manufacture of incandescent lamps the adventure of the metallic-filament lamp (first osmium, then tantalum, finally tungsten) was revolutionary after the carbon lamps had reigned supreme for so many years. In the preparation of the tungsten filament chemical engineering plays a large part in all the different processes devised for this work. Undoubtedly the most important recent advance is the method developed in the research laboratory of the General Electric Company under the direction of Dr. W. R. Whitney for producing ductile tungsten, as noted in an American Electrochemical Society paper of one of the inventors, Dr. C. G. Fink (our vol. VIII, p. 340).

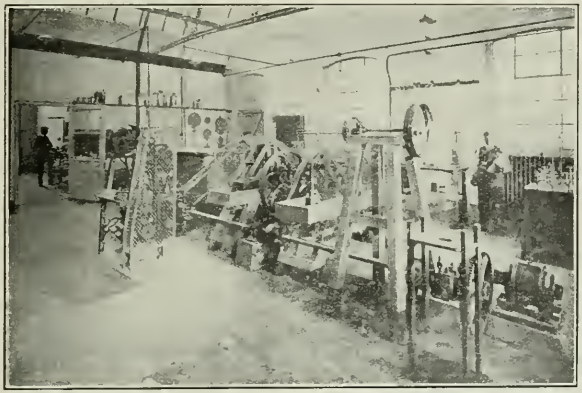


FIG. 1.—CHEMICAL DEPARTMENT, SHOWING MECHANICAL MIXERS IN FOREGROUND.

Nevertheless the older methods of making tungsten filaments are still of great importance and considerable interest and in view of the secrecy which generally surrounds metallic-filament lamp works some notes on a British tungsten-lamp factory—the Brimsdown Metal Lamp Works—are particularly interesting. They are taken from the London *Electrician* of Jan. 27 and London *Electrical Engineering* of Jan. 26.

The process of Dr. Kuzel in which the colloidal state of tungsten is made use of is employed in this factory. The filaments in the Brimsdown metal lamps are produced from colloidal tungsten, and not from tungsten powder. The colloid is mixed with water in the chemical department, and the solution is placed in bottles which are clamped securely in frames and rotated slowly through gearing by a small motor. These frames are shown in the foreground in Fig. 1. The process is one that ensures the thorough mixing of water and the colloidal tungsten.

The mixture is then reduced to a plastic state by another process and passed between water-cooled rollers. From these rollers it emerges as an extremely thin sheet of metal, and is twisted up and is pressed into a cartridge.

This cartridge is passed into the squirting department, a general view of which is given in Fig. 2. The actual squirting of the filament is carried out both by hand pressure and hydraulic pressure. In the illustration (Fig. 2) three hydraulic presses will be noticed on the right and six hand presses on the left. The material is squirted through diamond dies, which vary in diameter according to the candle-power and voltage of the lamps for which the filaments are intended. The filaments for the lamps of ordinary size are finer than a human hair, ranging from about one-fortieth to a millimeter upward in diameter.

The squirting is carried out vertically downward, and the thin material is received on a large white card, which is moved backward and forward so as to spread the filament over the card. When the card is full it is passed on to benches in another part of the same shop, and in the soft state the filaments are divided off into U-shaped loops. They are then laid in metal trays, which are placed in baking ovens. In these ovens the moisture is driven off and the filament loops are then cut off to standard lengths.

The forming department of the works is one of the most interesting, because it contains a quantity of special apparatus which has been designed and manufactured by the company's staff. This apparatus is fitted up in units, each of which is complete in itself, and mounted in groups on the benches, which run transversely across the shop.

The filament-forming process consists of a chemical and electrical treatment of the filament under a glass bell. The U-shaped filament is held in two clips, which serve both as supports and terminals for the testing circuit. A tungsten wire is hung at the bottom of the U as a weight. The bell is then partially exhausted and ammonia gas is admitted in a certain quantity into the bell. An exhausting tube is also provided to extract the air and other products during the forming process. Current is applied to the filament, and the voltage is gradually raised from practically nothing to 50 per cent above normal. This is controlled absolutely automatically by a motor, so that the current is gradually increased up to a certain value and then switched off. During this process the filament changes its form materially due to the weight of tungsten wire attached to it.

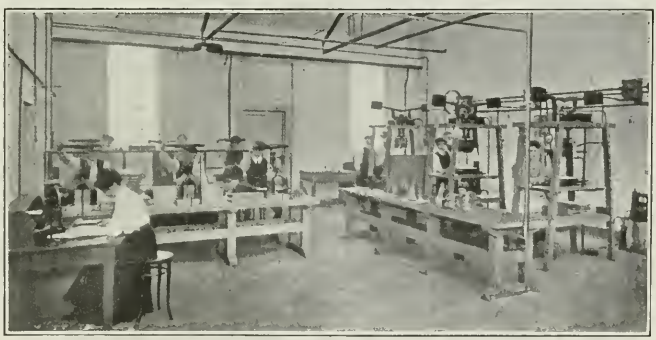


FIG. 2.—VIEW OF FILAMENT SQUIRTING SHOP.

All the binding material is driven out of the filament and only the tungsten metal remains. The entire operation is automatic after the bell has been lowered, and four filaments are treated at a time consecutively. All the operations are carried out by girls, who, of course, have no technical knowledge and do the work entirely mechanically. Further, the

quality of the product is uniformly good. This indicates the ingenuity of the automatic apparatus.

After the filaments are removed from under the forming bells they can be taken by the ends and twisted backward into a circle, a test which speaks highly for their strength and flexibility.

The forming benches occupy the bulk of the space in the filament factory; 80 girls are employed at the forming benches, each of whom can form a maximum of 600 filaments a day. At one end a number of motor-generators and air and ammonia compressors have been installed. At the other end of this department is a separate section, where the filaments are weighed and carefully cut off to standard lengths on a gauging and cutting machine. The weighing machines will register up to a tenth of a milligram.

Arrived at this point in the process of manufacture, the filaments are ready for mounting, but this work is carried on in a separate building. The lamp supports are made, the filaments are mounted in position, they are then welded in place, placed in the bulbs, and the final process of exhausting and sealing in is gone through. All these operations are carefully subdivided among the employees into groups which practically form separate departments in themselves.

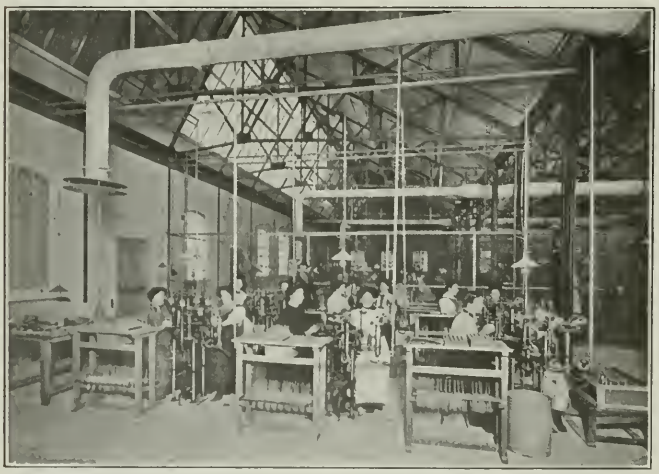


FIG. 3.—SECTION OF GLASS-BLOWING DEPARTMENT.

The foot-making, stem-making, fusing into the glass of the leading-in wires, welding of platinum tips, the fixing of the top and bottom hooks and supports for the filament, are severally delicate operations which are entrusted to each group. As the work proceeds in each department it is passed into a checking section, carefully examined and tested and passed out again, when approved, to the next section.

The leading-in wires are formed of short lengths of copper, platinum and nickel wire welded together, the platinum being a short length to pass through the glass. It is stated, however, that a new material (alloy) is being introduced in place of the platinum, which has, as platinum, the same coefficient of expansion as glass. The leading-in wires are inserted in their glass tube, which is pressed round them after being softened in a blow-pipe flame, the central glass stem is made with its circle of thin and flexible nickel hooks at the other end, and the filaments are placed in position and pasted to the lower nickel wires with a paste of finely divided tungsten paste.

The work is then checked and the filament cage goes to the welding section, where the pasted joint is electrically welded in an atmosphere of hydrogen under a glass bell. In this section the work is again checked and issued, after approval, to the sealing-in department, in which the filament cages are fused into the lamp bulb. This work is carried out on semi-automatic machines which are also attended by girls.

The final process, previous to the actual testing of the lamp, is that of exhausting and sealing off. This is carried out at special benches, each of which is fitted with a unit apparatus comprising a heating box to warm up the bulb, a number of vacuum nozzles, switches for putting the lamps in circuit, and a blow-pipe flame for sealing off.

The initial vacuum is obtained by electrically driven vacuum pumps, which are in operation in a building adjoining the lamp department and the final exhausting is carried out by Gaede rotary mercury pumps which are placed immediately adjacent to the exhausting nozzles. The outfit is the latest of its kind, and it is stated that it ensures a nearly perfect vacuum.

Manufacture of Aluminium.

In an article by Mr. Wm. Clacher in London *Electrical Review* of Jan. 20 some details are given on the manufacture of aluminium as carried out by the British Aluminium Company, Ltd. His notes on the manufacture of carbon electrodes by the same company are dealt with in a separate article in this issue.

From the molten electrolyte, consisting of alumina and its flux cryolite, there is reduction of the oxide at the point of contact of the carbon anodes, the carbon oxidizing and burning away. The metal thus produced sinks to the bottom of the electrolytic furnace, which is carbon-lined, and is drawn off thence.

This metal, from the purity of the material used—the carbon being 99 per cent and the alumina 99.9 per cent purity—is of very high percentage; over 99.5 per cent is very usual.

From the aluminium reduction works the ingot metal is taken to the mills and foundry to be worked into sheet and alloy and also for castings.

The 10-in. square carbon blocks are arranged in sets of 12 in the electric furnaces. The direct current for operating the electrolytic furnaces is obtained from 50-volt dynamos.

The plate for the bottom of the furnace is of cast iron and somewhat of the form of a grid. The interspaces are filled with carbon which is rammed and the whole plate covered. This form makes the contact between the carbon lining of the furnace and the cable much more effective.

The resistance of the carbon blocks is tested at the furnace. The carbon blocks are consumed away, the gases going to the atmosphere; hydrofluoric acid gas is also evolved from the cryolite used.

The metal as it comes from the furnace is afterward melted and the pure metal, free from any mechanical admixture, is molded in ingots. These ingots are bagged, about 20 ingots to the bag, and dispatched to the foundries and mills.

An allocation of the cost of producing the metal is given by Mr. Clacher as follows.

	Per Ton.
Energy	$25.00
Carbon	32.50
Alumina	140.00
General expenses and profit	140.00
	$337.50

This allocation of the cost seems interesting.

The power used is water-power, the most economical source of energy.

The alumina is purified from the bauxite by the Bayer process—solution in caustic soda and precipitation therefrom by addition of some previously precipitated alumina, filtered off and kilned.

The bauxite is first kilned to expel moisture, then finely pulverized; the finely divided material is mixed with caustic soda of 40 per cent strength to a fluid mass and kiered under an internal pressure of 30-60 lb. per square inch, 100 lb. steam jacketing the vessel. After eight hours the mass is blown by its internal pressure to a tank, whence the filter presses are charged; the height of the tank above the level of the presses depends on the fineness of grinding. The insoluble matter is termed red mud.

The mass when blown to the top tank is diluted to a suitable concentration and temperature, so that the wear and tear on the cloths is not heavy.

The filtered aluminate of soda is run to tanks and some alumina, previously precipitated, is added. After some time the alumina is precipitated, a white semi-flocculent substance.

This precipitate is then filtered, washed, partially dried in the press and calcined in a furnace at a cherry-red heat, after which the hard white armorphous alumina is ready for transportation in bags to the electric furnaces.

In analyzing the metal the impurities alone are determined, the difference being assessed aluminium. Of these impurities the most important are silicon, silica, sodium and iron. In practice the items customarily determined are the iron, and silica with silicon.

A New Machine for the Separation of Liquids and Solids by Centrifugal Force.

Centrifugal force is the keyword of the many inventions which have made the name of De Laval famous in the past thirty years. This is as much true for the big De Laval steam turbine which has played such an important part in the modernization of power plants as for the small De Laval cream separator which has completely revolutionized the dairying industry.

The industrial applications of centrifugal force are almost limitless, and it is no wonder that the De Laval concerns are constantly working out new machines for new purposes. In the chemical and metallurgical industries the use of centrifugal force suggests itself naturally for the separation of liquids and solids, and a small centrifugal machine of the De Laval Separator Company which is described and illustrated on page 159 of this issue, has been in successful commercial operation for some time for the clarification of liquids containing a small amount of sediment, etc.

But the bigger problem of separating liquids from solids in general (say mixtures containing from 2 per cent to 20 per cent of solids) by centrifugal force has only recently been solved. The new centrifugal separator, about which the first advance information is given in this article and which has not yet been placed on the market, is the outcome of ten years of continuous work of the engineering experts of the De Laval Separator Company, in the course of which some twenty-nine patents have been taken out for principles and details. But in view of all this, really the most surprising and most characteristic feature of the whole is the simplicity of the design.

The diagram, Fig. 1, gives a section of the machine. An outside view is given in Fig. 2, which shows a machine installed in the experimental plant. This does not represent ideal conditions, so that for continuous operation on a commercial scale some of the arrangements around the machine would be changed. The separating machine itself has so far proven equal to any task, even the most difficult, to which it has been put. Fig. 3 shows the feeding end of the machine and Fig. 4 the opposite end, with cover or cap removed, which shows the differential gear-driving mechanism. All of the moving parts of the machine are carefully encased, the only exposed part being the drive pulley, which is mounted on the shaft shown in Fig. 3.

The working or enclosed cylinder is 30 in. in diameter by 8 ft. long and revolves at a normal speed of 320 r.p.m., equal to a peripheral speed of 2513 lin. ft. per minute. Thus a product delivered into the machine is subjected to a centrifugal pressure, which causes the heavier part (the solids) to settle to the periphery.

The operation is as follows:

The raw product is delivered at *A* (Fig. 1), into a hollow arm of the supporting spider, through which it passes to the revolving cone *B*, thence to *C* and *D*. During the entering process the raw product attains practically the peripheral speed of the machine. The coarser or heavier solids are at once separated from the liquid, and by a series of broken helical scrapers, operating at a differential speed, they are drawn up an incline or cone, freed from the liquid, and discharged by the centrifugal force at *E* into the receiving chamber *F*. The liquid is discharged at the opposite end at *G* into the receiving chamber *H*.

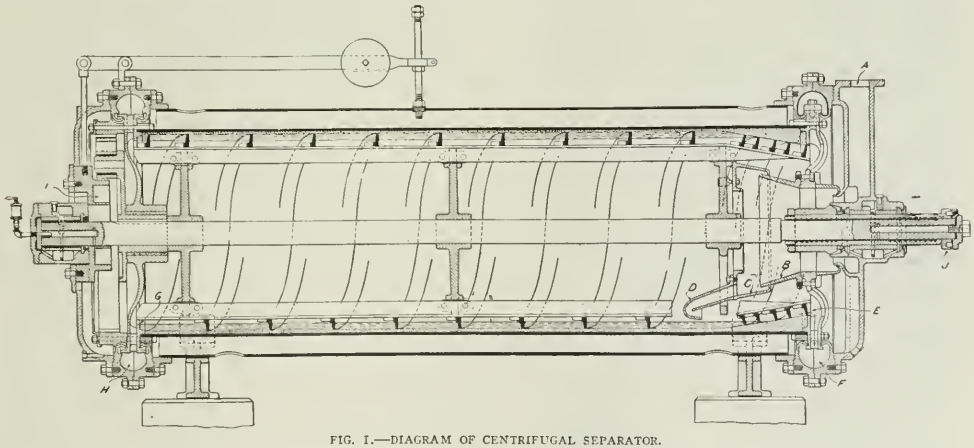


FIG. 1.—DIAGRAM OF CENTRIFUGAL SEPARATOR.

The liquid in passing to the discharge end of the cylinder travels a tortuous course, equal to 27 lin. ft. in distance, which may require one or more minutes to pass through, thus subjecting the lighter, or difficult to settle, solids to the high centrifugal pressure for a much longer period than if it was

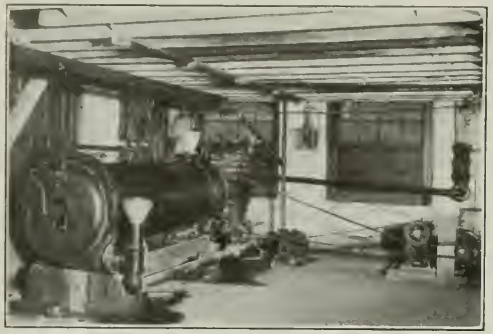


FIG. 2.—OUTSIDE VIEW OF CENTRIFUGAL SEPARATOR.

passed straight through the cylinder, the advantages and importance of which will be fully appreciated. During this period the solids are practically all settled to the periphery of the revolving cylinder, and as there accumulated from each point they are slowly drawn the length of the cylinder by the scrapers to the inlet end and discharged at *E*, as above described.

The capacity of the machine varies according to the kind of products, but unless very difficult the capacity ranges from 2400 to 3000 gal. per hour.

The speed of the machine may be varied from 150 to 350 r.p.m. to accommodate the different kinds of material.

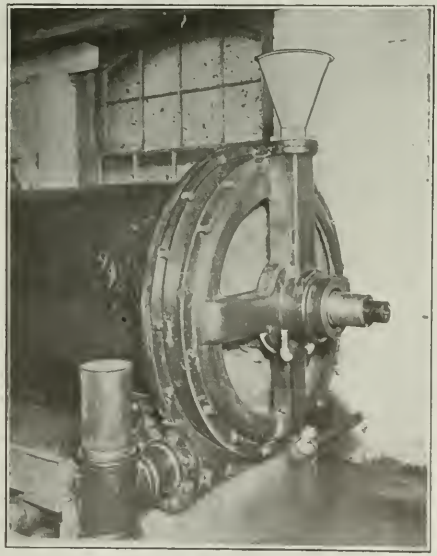


FIG. 3.—FEEDING END OF SEPARATOR.

The lever shown on the top of the machine with the round weight on end (Fig. 1) is the safety device connected to the differential driving gears and which prevents possible damage to the machine in the event of overloading or other cause. The lever is adjusted to permit of a certain pressure or force on the scrapers, and when it exceeds this amount the lever automatically rises, relieves the pressure and throws the differential driving mechanism out of commission, causing the scrapers to run in unison with the cylinder and as one piece. Without going into details, it may be said that this is one of the most ingenious details of the machine.

The machine when once adjusted to a product can be run continuously.

The weight of the machine complete is approximately 7000 lb. The oiling is largely automatic, and one man could manage and care for a number of machines.

This separator has now been working for 2½ months, for nine hours a day, in the experimental shop on a large variety of products, of which starch, sand, clay and whiting may be mentioned. Other materials are to be experimented with; thus the separator will be tried for cyanide work, etc.

From the samples of separated materials which we have seen it is safe to say that the separation obtained with this

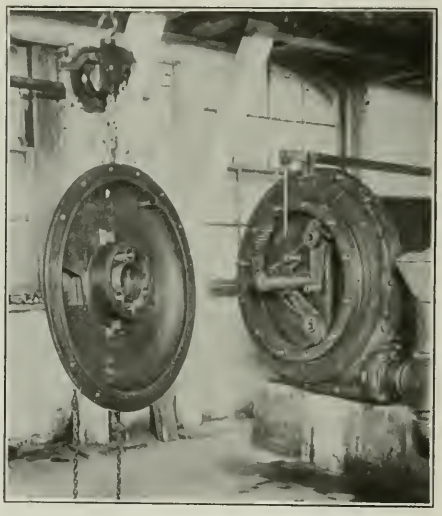


FIG. 4.—DIFFERENTIAL SPEED MECHANISM.

machine is practically perfect. If it is desired to remove the very last traces of sediment, which may have remained in the liquid discharged from the separator, this liquid may be passed through a centrifugal clarifier and filter, as described on page 159 of this issue.

Mixtures combining from 2 or 3 per cent up to 15 or 20 per cent of solids in water may be treated by this machine. The power for driving is very low. On a mixture containing 12½ per cent solids less than 5 hp is required. As mentioned before, the capacity of the machine ranges in general from 2400 to 3000 gal. per hour, according to the kind of material.

In view of the simplicity of design and practical fool-proofness of the machine, the cost of attendance and operation is very low. Its compactness and efficiency of handling quickly large amounts of difficult material should certainly attract the attention of the chemical and metallurgical engineer.

When the machine is placed on the market we expect to give further descriptions, with results of tests, though in the meantime we are sure the manufacturers will be pleased to give interested parties such further information as they may desire.

This new De Laval centrifugal separator is certainly an interesting example of the possibilities of centrifugal force and of the effectiveness of co-operation between the chemical and mechanical engineer.

Some Practical Experience with Corrosion of Metals*

By Rear-Admiral John T. Corner, C.B.

It should be pointed out at the outset that in this paper the writer purposes to limit himself to chiefly giving instances of corrosion which have come under his notice while actively engaged as a marine engineer and to detail some of the practical methods found to be efficacious in dealing with the same.

Some of the causes of corrosion of metals on shipboard are so obscure, and the origin so difficult to trace, that no satisfactory explanation is so far forthcoming. On the other hand, cases have occurred in which the causes operating to produce the effects are clearly recognized and easily dealt with.

The writer has had no personal experience of the trouble which took place through the corrosion of copper and its alloys in the old wooden ships, but that such corrosion did exist is within the knowledge of most of those who have studied the record of such ships. Cases are mentioned of copper sheathing having become corroded in mysterious ways and at unexpected times, and there are reports of the heads of the nails having corroded entirely off, allowing the sheathing to become detached.

There were doubtless at times troubles through corrosion in the bilge and other copper pipes of the old wooden steamships of war, but as a rule these appear to have been of minor importance; and in many ships the chief anxiety the engineers had concerning the pipes was to keep them in a bright and burnished condition so as to pass the captain's weekly inspection, at which the array of copper piping in the bilges was often one of the show sights of the ship.

With the introduction of iron for shipbuilding purposes the conditions are different, for it was soon found necessary to insulate as well as protect the copper pipes in or near the bilges—not, however, for their own welfare so much as for the sake of the adjacent plates and angles, which, when not so treated, were found to be adversely affected by contact, owing to the galvanic action set up between the metal pipes and the iron plating by the bilge water.

There were many instances in which very considerable trouble was caused. Among them may be mentioned the case of the troopship *Megaera*, which vessel had to be beached at St. Paul's Island about 1869 to prevent her sinking. It came out in the subsequent inquiry that among other defects in this ship there was a serious one due to a copper strainer fitted to a bilge section pipe in a remote part of the ship. The action set up by it was sufficient to eat right through the plates and so admit water to such an extent that the ship had to be run ashore to save the crew.

As far as can be ascertained the pipes themselves in iron ships were no more affected than those in wooden ships, probably not so much; and it was not until at or about the time of a great change in the internal economy of men-of-war that serious and persistent troubles through the corrosion of copper pipes began to be manifest and felt. This change was the introduction of the electric light on shipboard.

The writer's first experience of the insidious corrosion of copper pipes occurred in a ship which had recently been fitted with an electrical installation for working searchlights. During a commission of over three years there were, as far as can be remembered, only two pipes which seriously suffered from corrosion; both were water pipes, and both were eaten right through in small holes, which had the appearance of having been punched, and it is noteworthy that each of these pipes was in the immediate vicinity of an electric lead to a searchlight—the one forward and the other aft. The whole business was after all but a minor affair, and consequently very little notice was taken of it; it was only years afterward, when trying to assign a cause for similar trouble in various other ships, that the previous experience was recalled to throw light on an obscure question.

There is no doubt whatever that the time of the construction of the electric light in naval ships synchronized with a considerable increase in the corrosion of the copper and metal pipes and fittings; and as the parts affected were chiefly water fittings, the increased steam pressures which came at or near this period, and which might have been suspected as one of the causes, can be now quite absolved from any share or part of the trouble.

Sometimes pipes would be found eaten nearly, and in some quite, through in a very short period; moreover, the replaced parts often became similarly affected even when different brands or qualities of copper or brass were used. Sometimes the pipes were eaten in holes having sharp, clean-cut edges, as though punched; sometimes parts were pitted and honeycombed over a considerable area of surface; while in other examples the metal was nibbled out in such a way as almost to suggest the presence of rodents or microbes.

As regards the source of this trouble, some engineers for the reasons above stated were inclined to attribute it to electrical action due to the leakage from the mains, which was often so great as to allow electricity to permeate the whole ship. It must be admitted now that some of the earlier electrical installations on shipboard were a long way from perfect, inasmuch as leakage in some cases was sufficient to turn the ships themselves into storage batteries. Others attributed these defects in the copper pipes to induced currents set up in different parts of the ship by the currents in the electrical mains. But nothing definite could be proved, and those then dealing with the electrical plant disclaimed all responsibility for the trouble, and in fact sometimes expressed the opinion that the currents they used could hurt nothing and cause trouble to nobody.

It was also further suggested that the action might be due to mere chemical causes, but here again the responsibility for this was disclaimed by those concerned, the corrosion being put down by them to mechanical rather than chemical action; and there was certainly some ground for the contention, seeing that corrosion in pipes generally occurred at or near bends or branches; also very extensive corrosion often occurred on the bronze propeller blades, especially at the forward side where there was little likelihood of extensive action, electrical or chemical, but abundance of the mechanical kind.

It is very probable that a certain amount of chemical action was first induced by galvanic action, and the erosion caused by the friction of the water, which washed away the results of the chemical action, left the surface exposed to further and similar action. This explanation, however, would not fully apply to cases where clean-cut holes were found in the pipes, condenser tubes, etc.; these results were probably due to want of homogeneity in the composition of the metal, the galvanic and chemical action finding out the weakest places. It would also appear that the presence of air and moisture together largely influences the extent of the corrosion of these metals as of iron, and in a measure helps to explain the pitting on the forward sides of the propellor blades, where the partial vacuum formed by the motion of the screws causes air to be drawn from the surrounding sea water, which assists the corrosive action.

It was also suggested that the corrosion might be due to the use of copper of too pure a quality; and in this connection it may be noted that some locomotive engineers specify for about ½ per cent of arsenic in the copper fire-box plates, as a preventive against corrosion. So there really may have been something in this suggestion, seeing that pure electrolytic copper was beginning to be used at or about the time referred to.

But whatever may have been the causes, the pressing business of the day was to discover the cure of the complaint; the prevention of the corrosion, which affected most of the water pipes, including the condenser tubes of many ships, was what all aimed at. As regards the latter, it is understood there was

*A paper read before the (British) Institute of Metals at the January, 1911, meeting.

not much general corrosion of the tubes in iron condensers. No doubt there have been, as there still are, some instances in which the tubes of iron condensers have given considerable trouble through various causes, some of which were foul circulating water, defective materials, excessive vibration through indirect local pressure, such as the direct impinging of the steam or circulating water on the tubes, etc.; but as regards the corrosion under consideration, it may be stated generally that it is certainly much less in condensers with iron shells than in those with brass shells.

One special case of an iron condenser which came under the writer's notice was interesting and unique, and may be mentioned here: A ship with a cast-iron condenser, with circulating water outside the tubes, had been running for several years with a set of tubes which had given no trouble whatever. The ship unfortunately got ashore on a sandbank, and in getting off stirred up the sand so much as to get a large quantity deposited in the condenser on the circulating sides. On returning to port for overhaul and repairs, the tubes were taken out to get rid of the sand. This afforded the opportunity for examining and attending to the inside of the condenser shells. They were cleaned by being chipped and scraped, and finally given a good coating of red-lead paint. Shortly after the ship resumed service the condenser tubes, which up to then had given no trouble, began to go to such an extent that the chip had to be again laid up for the complete retubing of the condensers. When this was done the red-lead paint was all removed, after which no further trouble with the tube was experienced. This result clearly showed that the brass tubes were being protected when the iron shells were bare, but when covered with lead paint a corrosive action on the tubes manifested itself.

Another case of the protection afforded by iron may be mentioned: In a certain ship one particular pipe conveying circulating water was repeatedly in trouble through corrosion. This pipe would be found badly eaten away at a bend. It was taken down, patched and replaced time after time, but the corrosion still went on and continued. The chief engineer then resorted to the following expedient: He sweated a lump of copper on the outside of the pipe, so as to provide material for a screwed hole, through which a large iron bolt, about 1½ in. diameter, was fitted so that the point protruded; and no further trouble was experienced with this pipe, the bolt being screwed further in as the point became eaten away, and being entirely renewed when necessary.

These two examples appear to indicate that the presence of sufficient bare or unprotected iron in perfect metallic contact with the system strongly counteracts the corrosion of the copper or brass; the chemical or electrical or mechanical action, whichever it may be, solely or jointly, being entirely expended on the iron. It is therefore by the free use of iron or steel that it is considered the best protection of the copper or alloy pipes is provided.

Before resorting, however, to such free use of iron various other means were tried in different ships. One method was to coat the insides of the pipes with marine glue, another to use plumber's solder or tinman's solder as an internal coating. Special alloys containing aluminium were tried. Zinc slabs and blocks were used, as also zinc bars let in among the condenser tubes; and though some of these expedients effected a certain amount of good, it is considered that the present practice has more to recommend it than any of them, and that is the insertion of short iron lengths of pipe in runs of water pipes which can be easily removed and replaced; and further, the use of iron or steel doors for brass condensers with iron rods and plates in and about the tubes is very effective as a protection to the tubes and plates.

In some warships of late years the condenser barrels are made of steel. Of course in this latter case there is the possibility of a certain amount of corrosion in the condensers themselves, but as this is only to be expected on the water-side, it should not be difficult to deal with—the design of the condenser being such as to admit of the easy replacement of parts likely to be affected and provision being made for zinc protectors as in boilers.

It is considered more than probable that the reason why iron is found on the whole to be more efficient as a protective than zinc is that with the former better metallic contact is assured and maintained, and perfect metallic contact is undoubtedly an absolute necessity.

Respecting the corrosion on propeller blades, it is not known to the writer whether iron has been tried as a protection against corrosion; but he would suggest for consideration that if easily renewable plugs of iron could be put at the parts of the propeller blades where corrosion is likely to occur they might lead to a marked difference in the extent of the pitting, for it may possibly be found that so long as there is iron in the immediate vicinity sufficient to absorb the corrosive action the bronze will be entirely protected.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Corrosion of Metals and Alloys.

Mr. G. D. Bengough's lengthy preliminary report to the corrosion committee of the Institute of Metals was discussed at the annual meeting of that body on Jan. 18.

Sir William Tilden was of opinion that iron in the metallic form undoubtedly afforded protection, yet the oxide of iron, if deposited locally, was a cause of corrosion in brass tubes he had examined a quarter of a century ago; and it should be noted that ferric oxide was electronegative. There were great differences of opinion on the subject of corrosion, and it was necessary that the question should be carefully investigated. He emphasized the necessity of assimilating the conditions of the experiments to those of actual practice.

Dr. W. Rosenhain said reference had been made to the dimensions of the microstructure as affecting corrosion and there was evidence that the metal having the corrosion was more liable to pitting. If two pieces of brass were taken, one lightly strained and the second more severely strained, a coarser structure was given to the latter. It was possible that similar differences in the microstructure might result from an unequal treatment of different parts of the same tube during the drawing process.

Mr. A. E. Seaton said that brass tubes were largely used in the navy for ships' boilers 30 years ago, with iron stay-tubes, and there was not any trouble from pitting then. Condenser tubes were formerly of 16-gage thickness, but were now of 18-gage to 20-gage.

Prof. T. Turner said he had conducted some experiments on the green deposit which formed on condenser tubes. The deposit was a non-conductor and was not part of an electric cell, but that fact did not controvert the view that the green deposit might play a part in causing corrosion.

Dr. J. T. Dunn thought that pitting was the result of the presence of local impurities introduced with the water. It had been proved that local corrosion of brass was set up by small pieces of rusty iron when immersed in sea water.

Mr. Arnold Philip (Admiralty) objected that the report had been made rather from the academical than the practical standpoint. A vast amount of useful information was available from manufacturers and users on the subject of corrosion and it was a fundamental omission not to have drawn more largely on their experience. The causes of corrosion were very numerous. The chief of these were the attacks of particles of carbon, etc., hydrochloric acid in the steam, externally applied e.m.f. mechanical abrasion by steam and water, presence of air and gases at bends of tubes, thermo-chemical action, mechanical strain, intrusion of foreign matter at surface of tubes, pres-

ence of two or more phases in the alloy, character and speed of the cooling water, and impurities in the alloy.

Mr. E. L. Rhead mentioned the influence of difference in temperature on the inner and outer walls of condenser tubes as a possible cause of accelerating corrosion. The question of annealing temperatures had an important bearing on the life of the tubes. With regard to crystalline structure, manufacturers employed photographs of crystals in different sizes in order to work within specified limits. He had recently examined some steel tubes which had corroded and found that failure was due to the use of an originally porous material.

Copper-Zinc Alloys.

Another paper read at the annual meeting of the Institute of Metals was on "A New Critical Point in Copper-Zinc Alloys: Its Interpretation, and Influence on Their Properties," by Prof. H. C. H. Carpenter and Mr. C. A. Edwards.

The discussion was opened by Dr. W. Rosenhain, who, while admitting the existence of the critical point, was not able to agree with the authors in their interpretation of the facts.

Mr. G. D. Bengough was not able to coincide with Dr. Rosenhain's view, and, besides accepting the authors' theory, considered that the paper marked a distinct epoch in the history of brass. Recent researches of his own on the mechanical properties of alloys at high temperatures and on delta metal indicated that a remarkable change took place at the critical temperature, which, in his experiments, was 510° C. He also found a corresponding change in the 70:30 brass. His experiments on Muntz metal showed that at 445° C. there was not any B. constituent, which result was in accordance with the authors' conclusions.

Prof. A. K. Huntington remarked that the results of this research seemed to imply that with prolonged lapse of time all copper-zinc alloys would become brittle. He had lately made alloys with 36 per cent of zinc and less, and found that these—even after annealing—contained an appreciable proportion of the beta constituent.

Mr. E. F. Law said it was doubtful whether all inter-metallic compounds retained their identity when in solution, and this was a question which was well worth investigation. His opinion was that inter-metallic compounds were loosely combined and might behave similarly to double salts. The critical point dealt with by the authors was not a new one. Roberts-Austen recorded its existence in 1897, and, although Shepherd, Tafel and Bornemann ascribed his discovery to experimental error because they failed to detect the thermal change in question, the authors had now established beyond question that Roberts-Austen's observations were correct.

Mr. O. F. Hudson was not satisfied that the metal lost its brittleness above a certain temperature, and could subsequently revert to a brittle condition on further cooling. Brass was brittle only within a limited range of temperature. The authors' theories were consistent with the thermal observation, but not with the microscopic evidence.

Mr. E. L. Rhead said that calendering rolls, which were forced onto steel mandrils by heavy pressure, withstood a long period of work at high temperature, but after some months a longitudinal crack developed in consequence of a complete change in the character of the metal resulting from prolonged heating, and there was not much doubt that this structural change was connected with the point of inversion referred to by the authors.

Prof. T. Turner said that the establishment of this arrest point was of great consequence to brass manufacturers. It had long been known to practical men, but had never been mentioned in any text-books.

British Engineering Trade Returns.

The official statistics for 1910 published on Jan. 12 in the *Board of Trade Journal* show increases in both the imports and exports of all the principal engineering materials and manufactures. The imports of iron and steel amounted to £9,093,795, an increase compared with 1909 of £1,122,201, while the exports reached £43,002,937, a rise of £4,810,795. The imports and exports of other metals and manufactures thereof were £24,699,531 and £10,360,159, the respective increases being £353,203 and £1,651,214. The imports of electrical goods increased by £363,960 and the exports by £1,887,641, the totals for the year being £1,686,469 and £4,117,840. The imports of machinery amounted to £4,471,303, an increase of £32,967, and the exports to £29,296,721, a rise of £1,239,078. The value of ships imported was £27,104 and exports were £8,769,331, increases of £3,178 and £2,842,217. The exports of ships decreased £1,798,144 from the values in 1908.

Production of Nitric Oxide by Combustion of Hydrogen.

At high pressures the combustion of hydrogen in a mixture of equal molecular parts of oxygen and nitrogen produces a considerable yield of oxides of nitrogen. A. Walokitin states that, without heating the gases, about three molecules of nitric acid in 100 of water—approximately ten per cent nitric acid—are obtained. Pressure exerts an appreciable effect only above three atmospheres, with a maximum effect at 16 atmospheres With excess of oxygen practically the same results ensue. The yields agree with earlier results with explosion methods; but—probably in consequence of the theoretical temperature not being reached—are rather lower than the calculated values. The yield from the combustion of hydrogen in air is, of course, lower.

The Electric Furnace in Great Britain.

The *Times* of Nov. 30 calls attention to the reluctance of British steel manufacturers to advance with the times in the matter of electric furnaces. The article says that to England, with a rapidly diminishing stock of high-grade ore, the electric furnace is of peculiar value, as offering a means by which high-class steel can be manufactured from such low-grade ores as those from Cleveland, Lincolnshire and Northamptonshire; and proceeds to combat the argument that the absence of water-power is a fatal impediment to the extensive adoption of the electric furnace in England. The writer points out, too, that in the great manufacturing centers waste gases from blast furnaces and coke ovens as well as exhaust steam furnish sources of power which could be utilized for electric steel refining; and, further, that it is believed that there is at least one English manufactory which is buying electric energy at ⅛d per unit. This is an exceptional case, but indicates future possibilities. The article referred to is inspired by the adoption of the Heroult patents by the United States Steel Corporation

The Cost of Electric Energy in England.

An editorial note in the Engineering Supplement of *The Times* of Nov. 30 pointed out the misapprehension existing with regard to especial cheapness of electric energy obtained from water-power abroad as compared with the rates at which current can be supplied in England. In *The Times* of Dec. 7 Mr. A. S. Esslemont stated that he had recently made a tour of Norway, Sweden and southern Europe to study this question and found the average price of electric energy from water-power no lower than can be obtained to-day, for similar loads, from the power companies on the northeast coast of England. He pointed out that the very low prices often mentioned as being prevalent in Norway and Sweden are frequently misleading, the prices varying from $6.25 to $8.75 per electric hp-year. He stated that the Hydroelectric Power Company does not offer such terms; they are quite exceptional and are only occasionally obtainable under especially favorable circumstances. He considers that steel works on the northeast coast are in a better position to adopt the electric furnace so far as a supply of electric energy is concerned than many of their continental competitors, by reason of the huge output of the supply companies and their policy and enterprise in utilizing the waste gases from blast furnaces and coke ovens.

The Physical Society's Exhibition.

The Physical Society's sixth annual exhibition was held at

the Imperial College of Science and Technology on Dec. 20. The exhibits included almost all the newest and best forms of scientific apparatus from some 35 of the leading firms in Britain; and there was a very good attendance. Even to name all the various apparatus well deserving of commendation would extend this notice to an inordinate length, and only those more directly of interest to engineers will be mentioned. The Cambridge Scientific Instrument Company showed Peake's patent compensating leads for thermo-couples, the Brearley recalescence curve tracer, and a new "Bi-meter" CO_2 recorder, which consists of two meters with an intermediate absorption chamber; Messrs. A. C. Cossor had Rose's high-vacuum oil pump, suited to the exhaustion of lamps; Messrs. Crompton & Company pyrometers, thermo-couples and electrical instruments; Messrs. Elliott Brothers, micrometer gages and the Wimperis accelerometer; Messrs. Everett, Edgcumbe & Company photometers and the Trotter combined accelerometer and gradient measurer; Messrs. Evershed & Vignoles, the Digby & Biggs "Dionic water tester" by which, from the conductivity, the degree of hardness, amount of condenser leakage, priming and sewage pollution may be rapidly determined; the Foster Instrument Company, a fixed-focus radiation pyrometer giving direct-temperature readings without focussing or other adjustment, and a base metal thermo-couple of high e.m.f. and low resistance; Messrs. Gallenkamp & Company, an electric furnace designed for experiments in recalescence and radiant heat; Messrs. J. J. Griffin & Sons, improved electric furnaces; the India Rubber, Gutta Percha & Telegraph Works Company, the Bleeck-Love primary battery—a new two-fluid cell giving 2.7 volts with great current capacity and constancy; Messrs. Muirhead & Company, an electromagnetic rectifier which utilizes all the wave and has an efficiency of about 90 per cent; Mr. R. W. Paul, new patterns of thermo-electric and resistance pyrometers and a thermo-electric indicator which can be read from a distance of 20 ft. to 30 ft.; Messrs. Reynolds & Branson, a lamp for testing color-vision; Messrs. Townson & Mercer, the Wanner optical pyrometer, and Somerville's CS_2 and H_2S apparatus for gas testing. The optical apparatus shown by Messrs. R. & J. Beck, Messrs. J. H. Dallmeyer & Company, Messrs. A. Hilger, Messrs. Newton & Company and Messrs. Carl Zeiss was of a very high standard.

New Process of Extraction of Copper and Zinc from Complex Ores.

Dr. W. Hommel and Mr. H. T. Durant subject a mixture of roasted ore and water to the action of cooled sulphurous acid gas, obtained from an ore-roasting furnace. The sulphite solution thus formed is collected and submitted to the action of hot gases containing sulphurous acid from the roasting furnace, the excess of sulphurous acid being thereby driven off to enrich the escaping furnace gases, and the copper or zinc is precipitated as monosulphite. The process has been patented.

A New Scotch Iron Ore Field.

The iron ore recently discovered in the Island of Raasay is of rather low grade, only about 30 per cent, but is quite fit for making foundry iron. The island has an area of 28 sq. miles and there is an excellent site for the construction of a harbor. After the field has been thoroughly proved, Messrs. William Baird & Company propose to erect blast furnaces to smelt the ore on the island and to take the necessary coal from Clyde ports by steamers, which will return laden with pig iron.

Corrosion of Copper and Iron Alloys by Aerated Salt Water

A piece of "delta" metal corroded by sea water was found by Mr. W. P. Jorissen to contain 41.1 per cent of cuprous oxide, no cupric oxide and no zinc. Cuprous oxide was also formed by the action on copper of sea water or sodium chloride solution of similar strength for two years. If copper is partly immersed in distilled water cuprous oxide is produced on the parts submerged and cupric oxide on the surface exposed to the air.

Samples of iron, some corroded underground and some by sea water, contained both ferrous and ferric oxides and from 14 per cent to 45 per cent of substances insoluble in hydrochloric acid and which contained 50 per cent and upward of carbon. This, together with the fact that the specific gravity of the samples varied from 2.0 to 2.53, shows that a large proportion of the iron had been dissolved.

Preparation of Tantalum Compounds from West Australian Fergusonite.

A specimen of fergusonite from West Australia was found by Messrs. E. Wedekind and W. Maas to contain 54.22 per cent of Ta_2O_5, 27.73 per cent Y_2O_3, 1.8 per cent Cb_2O_5, 3.2 per cent Ce_2O_3, and 0.36 per cent ThO_2. After separation of the basic and acid constituents by the Krüss and Nilson method (Abtr. Chem. Soc., 1887, 706), the residue consisted essentially of tantalic, columbic and titanic acids; 7.5 gm of the residue were carefully added to 15 gm of fused sodic carbonate in a platinum crucible, than a further addition of 4 gm of sodic carbonate and 1.6 gm of sodic nitrate was made; after cooling the fused material was extracted with 750 cc cold water, filtered and left till next day, when 6.4 gm of sodic tantalate had crystallized out. The best crystals were removed, the rest was dissolved in much hot water, and, after cooling, tantalic acid was precipitated by carbon dioxide, dried at 100° C., treated with a little fuming nitric acid, evaporated to dryness, and ignited to pentoxide.

Market Prices.

January, 1911.

Copper has shown, on the whole, a downward tendency. Starting at £56.10.0 it had fallen by the 13th to £55.6.0, recovering sharply to £56.2.6 on the 18th, afterwards sagging and closing at £55.50.

Tin has made a very strong upward movement on the whole month. Opened at £179.18.0, and rose in three sharp jerks followed by equally sharp depressions to £185 on the 10th and £190.10.0 on the 18th. It has since been to £192.10.0, but closes at £190.

Lead has been steady with slight downward tendency. Opened at £13.10.0, closed at £13.7.6.

Hematite has been irregular. Opening at 66/6 it rose to 67/- by the 9th, and remained fairly steady till the 17th, afterward dropping suddenly to 66/-, and then rising with irregular movements to 67/2½.

Alum, lump, loose	£5	12	6
Antimony, black sulphite powder, per ton	22	0	0
Borax, British refined crystal, per ton	16	0	0
Sal ammoniac, lump, first delivered U. K., per ton	42	0	0
Copper sulphate, per ton	19	5	0
Caustic soda, ash, 48 per cent, ordinary, per ton	5	10	0
Litharge, flake, per ton	15	10	0
Sulphur, recovered, per ton	5	0	0
Camphor, 1-oz. tablets		1	7
Shellac, standard T. N. orange spots, per cwt.	4	4	0
Hydrochloric acid, per cwt.		5	0
Copper ores, 10 to 25 per cent unit.	8/10½ to	9	10½
Tin ore, 70 per cent, per ton	£116 to £118		
Carbolic acid, liquid, 97/99 per cent, per gal.		1	0
Creosote, ordinary good liquid, per gal.			2
Naphtha, solvent, 90 per cent at 160° C., per gal.			9
Mica, small slab, original cases, per pound	6d to	1	0
India rubber, Para fine, kept in the neighborhood of 5/4 till the 11th, afterwards falling to 4/10, thence recovered sharply, closing at		5	3
Platinum, per oz.	8	0	0
Sulphate of ammonia, f.o.b. Liverpool, per ton	13	7	6

The most marked advance (excluding tin ore) is that of sulphate of ammonia, 10/9 per ton. Copper sulphate is down 5/-, litharge 5/- dearer, sulphur 5/- dearer, shellac up 1/-, copper ore down 6d, tin ore up £12, naphtha solvent 1d less per gallon, and india rubber 1/- cheaper.

Of the metals tin and iron have improved, copper and lead are slightly down.

Synopsis of Metallurgical and Chemical Literature

Iron and Steel.

Tests of Metals by Alternate Stresses.—A. Boudouard has made tests on the influence of alternate stresses on various metals, and has summed up the results in a paper in the *Bulletin de la Société d'Encouragement* of December, 1910. A metal can be fractured, not only by applying once a breaking load, but also by submitting it to often-repeated tensions and compressions of much smaller magnitude. The resistance of metals to alternate stresses is of great importance in industrial application of steels. A metal is not perfectly elastic, and when an effort which has caused a deformation disappears the piece returns approximately to its initial form, but, nevertheless, it suffers a permanent deformation, though a very feeble one. This residual deformation is negligible compared with the elastic deformations, but very frequent repetitions can produce a profound change of the metal, and even a fracture. The first steel examined in these tests was a good commercial steel with 0.3 per cent carbon, first in its natural state, then, after having been annealed at 350° for 10 hours without access of air, and finally hardened by water quenching from 1000° C. The bar of the steel in its natural state broke after 12½ hours continued vibration, or 1,350,000 vibrations. The annealed bar broke after 4½ hours and 486,000 vibrations. The hardened bar broke after 30 minutes, and 54,000 vibrations. The rupture always took place near the spot where the steel bar was clamped, called here the clamping place. The coming rupture indicated itself in advance by a dampening of the vibrations. At the moment of breaking the free part did not fall off at once, but could only be separated from the other part by some effort. Examination of the section with the naked eye showed parallel zones with grains of different sizes, indicating the phenomenon of shearing. Following are the results of the tests with different metals:

1. *Puddled Iron.*—Iron studied in the natural state; annealed at 950° C. and cooled in air. Bar broke after 18 hours 45 minutes, or 1,995,000 vibrations. The break occurred 5 millimeters inside of clamping place, and pieces were separated with great difficulty.

2. *Very soft steel,* annealed at 950°, cooled in air. Bar broke after 11 hours 15 minutes, or 1,215,000 vibrations. Break occurred 7 millimeters inside of clamping place, and pieces were separated with great difficulty.

3. *Steel with* 0.3 *per cent Carbon.*—Studied first in the natural state (annealed at 900°, cooled in air), and, secondly, hardened (heated to 900° C. and quenched in 15° water). The steel in its natural state broke after 13 hours 15 minutes, or 1,431,000 vibrations. Break started several millimeters inside of clamping place. When drawn out of vice jaws, bar remained in one piece and was only separated with great difficulty. Fracture seemed to extend from the interior to the exterior. The hardened steel bar broke after 14 hours, or 1,512,000 vibrations. Fracture began 12 millimeters outside of clamping place. The sections showed results of shearing of metal particles.

4. *Steel with* 0.6 *per cent Carbon.*—Studied first in natural state (annealed at 850°, then at 700°, and cooled in air), and secondly, hardened (quenched from 850° in 15° water). Annealed bar broke after 6 hours, and 648,000 vibrations. Fracture showed 2-3 millimeter outside of clamping place, and the same observations as on 0.3 per cent carbon were made. The hardened bar broke after 1 hour 55 minutes, and 153,000 vibrations. Fracture started several millimeters inside of clamping place. The two sections of the bar separated without much effort. Sections showed results of shearing. Fracture is not perpendicular to longitudinal axis of bar, and found to be very irregular.

5. *Steel with* 1 *per cent Carbon.*—First in natural state (annealed at 850°, then at 650°, and cooled in air); second, hardened (heated to 850° C. and quenched in 15° water); third, heated to 850°, quenched in 15° oil bath, and reheated to 350° without access of air. The natural steel bar broke after 3 hours 55 minutes, and 369,000 vibrations. Fracture was produced several millimeters outside of clamping place, and same phenomena as in natural steel of 0.3 and 0.6 per cent carbon were observed. The hardened bar broke after 5 minutes, and 9000 vibrations, 7-8 millimeter outside of clamping place. This was due, however, to fracture of the bar during mounting for test. The hardened and reheated bar did not break after 26½ hours, and 2,862,000 vibrations. Tests were not continued on this bar.

The general conclusions are summed up as follows: As these tests can be repeated under same conditions several times, they can be used to find the resistance of steels to alternate stresses. Sufficiently prolonged vibrations lead to the ultimate breaking of the metal. The number of vibrations necessary for breaking varies inversely as the carbon percentage of the semi-hard and hard steels. Puddled iron and very soft steels with similar chemical composition, except for manganese, which varies between 0.050 and 0.400 per cent, present a much greater difference from point of view of length of time for vibrations to produce fracture. The puddled iron has a much greater resistance than the soft steel; 0.3 per cent carbon steel, annealed or hardened, shows no appreciable difference. In the case of hard steels hardening decreases the length of time necessary to produce break considerably. Reheating improves the metal, shown by the last case, where a 1 per cent carbon steel did not break after 26½ hours' vibrations, when reheated.

Examination of the fractured sections shows similarity. Lines of separation, along which the fracture gradually takes place, can be seen. The size of the grain varies in the different zones, and appears notably enlarged in hardened steels. Shearing lines are seen perpendicular to the edges of the fractured section. As regards the mechanical properties, the high elastic limit of puddled iron and very soft steel is pointed out, after having been subjected to vibrations leading to break. All the metals tested broke, although tested considerably below the elastic limit. Some observations were also made on the dampening of the vibrations just before metal breaks.

Gold and Silver.

Improved Grinding Pan Discharge.—In the *Monthly Journal* of the Chamber of Mines of Western Australia, Mr. G. C. Klug describes a simple expedient for obtaining a uniform discharge from grinding pans. The design is by C. C. Freeman, who introduced it at the Great Fingal mill, where it has been in successful use for over a year. It consists of a curved 3-in. pipe introduced into the side of a Wheeler 5-ft. pan opposite the feed and about 12.5 in. below the top. As the mullers revolve at 60 r.p.m. the only product which can flow out of the pipe is the —30 mesh, this being sufficiently light to be unaffected by the centrifugal force produced by the revolution of the mullers. This classifying action of centrifugal force is remarkably effective, as the overflow contains only 4 per cent to 6 per cent of coarse material, as against 10 per cent to 15 per cent which formerly was mixed with the —30 mesh when ordinary classifiers were used. The stamp battery product which goes to these Wheeler pans is crushed through a screen having seven holes per linear inch. One pan will take care of the output of five stamps. The stamp duty has been increased from 7.5 tons to 9.2 tons per day as a result of the change.

Revolving Pressure Filter.—A horizontal tubular revolving filter is an innovation installed at El Oro mill, Mexico, where six units are in operation, handling 1000 tons of slime per day at a cost of about 4 cents per ton. The filter is the invention of EDWIN BURT, who describes it in the Jan. 21, 1911, issue of the *Engineering and Mining Journal*. Each filter consists of a cylinder 42 in. in diameter and 40 ft. long, set in a horizontal position. The head end is closed with a casting bolted to the flanges of the shell. The discharge end is closed by a plate operated by a hydraulic cylinder and piston, working under 125 lb. pressure. The filter revolves on trunnion bearings at the head end, and on roller bearings near the discharge.

The inside of the filter is lined with 20 filter cloths, each 26 in. by 10 ft., placed with the long axis corresponding to that of the tube. This allows five cloths to the circle and 10 to the length of the tube. The cloths are held in place by curved angle irons, staggered throughout the interior in such a way that they act as a spiral guide for the discharge of the cake when it is washed. Midway in the length of each cloth is a nipple and screen inserted through the shell of the filter for the discharge of the filtered solution. As there are five cloths in the circle there are five nipple discharges in the circumference of the tube at four points in its length. Each filter cloth is a unit in itself and its discharge may be plugged independently of the others if the effluent is cloudy.

A set of five valves at the head of the filter is used in the manipulation, viz., one 4-in. slime feed, one 2-in. wash solution, one 2-in. wash water, one ¾-in. compressed air and one 3-in. escape for reducing pressure. The filters revolve continuously. When the doors are closed the slime feed is admitted, under a gravity head of 60 ft., until the gage shows 35 lb. pressure. The feed is then closed and compressed air passed in to bring the pressure up to 40 lb. This is maintained until the solids are caked. Wash solution is then introduced and the same procedure followed. Finally wash water is used in the same way. The pressure is then dropped to zero, the discharge door opened and the cake discharged by the action of the curved angle irons after it has fallen from the walls of the tube. An acid bath is given once a month to remove lime salts. The life of the cloths is about four months; cost of renewal, less than 0.4 cents per ton of slime filtered. Time required for complete cycle: Charging slime, 3 minutes; making cake, 23 minutes; washing, 33 minutes; discharging, 3 minutes; total, 62 minutes.

Copper.

Purification of Electrolytes.—Writing in the *Australian Mining Standard* for Dec. 28, 1910, and Jan. 4, 1911, Mr. H. SCHRODER describes work done at Lithgow, N. S. W., on the purification of electrolytes used in electrolytic copper refining.

Crystallization Method.—The old method used was to draw off a portion of the electrolyte each day into lead-lined evaporating tanks having a hard lead coil in the bottom, connected with exhaust steam from the engines. The solution was evaporated to sp. gr. 1.4. The evaporation amounted to about 3 in. of solution every 12 hours from a tank having a surface area of 7 ft. 6 in. x 3 ft. 6 in. While still hot the solution was syphoned into crystallizing tanks in which were suspended strips of lead on which the copper sulphate crystallized. When the solution had cooled the strips were removed and the crystals detached. The solution was then reconcentrated and recrystallized, but a third crystallization could not be advantageously effected on account of the impurity of the crystals obtained. Consequently, the impure mother liquor was passed over scrap iron contained in boxes similar to zinc boxes used in gold precipitation and the remaining copper recovered in this way. The hot mother liquor solution was delivered to the scrap-iron boxes at the rate of 2 gal. per minute and issued from the fifth box free from copper. The waste liquor, being a strong solution of iron sulphate, was discarded. Much sulphuric acid was lost by this method which might have been saved by first saturating the solution with scrap copper. This, however, would have added to the production of bluestone, and as there was only a limited market for this the plan was not adopted.

The cement copper obtained from the scrap-iron boxes was dried and roasted and gradually worked up by charging it in small lots into the refinery with pigs of rough copper. But the elimination of arsenic was not great and eventually most of it got back into the copper anodes by way of flue dust subsequently collected and melted. Roasting the cement copper by itself was ineffective for the removal of the arsenic, for although dense white fumes came off there was less than 30 per cent arsenic driven off.

The entire method could have been improved by first saturating the solution with copper, making three batches of crystals and thereby reducing the amount of impure cement copper to be dealt with later. This latter might have been stored, mixed with flue dust or fine ore and smelted to a rich matte which could have been blown to blister and thus eliminate most of the arsenic; for while mere roasting of the cement copper is ineffectual in removing arsenic, still the reverberatory smelting and converting will remove it to a great degree, as has been shown by authorities on copper smelting.

Experiments with Insoluble Anodes.—These were instigated by G. H. Blakemore, and although never completed, owing to the works being closed, enough data were obtained to show the nature and results of the work. The object was to purify the electrolyte and at the same time produce the least quantity of impure copper. First a portion of the ordinary electrolyte in circulation was shunted off and circulated through a series of tanks fitted with lead anodes to reduce the copper content. The flow was regulated so that the solution leaving the last tank still contained enough copper to precipitate pure metal.

This liquor then flowed into a second set of tanks provided with insoluble anodes and circulation stopped. The current was then applied and in 8 to 12 days all the arsenic and bismuth were deposited. The cathodes were perfectly black and a large portion of the deposit fell to the bottom of the tank. The solution still carried about 2 per cent $CuSO_4 . 5 H_2O$, and was about 50 per cent stronger in sulphuric acid than the regular electrolyte. In one experiment the electrolyte contained 10 per cent $CuSO_4 . 5 H_2O$. After three days the $CuSO_4 . 5 H_2O$ was reduced to 2.08 per cent and remained at that for four days more, by which time all the bismuth was removed and the arsenic reduced to a very small amount. The current density was 7 amp per square foot of cathode surface and the drop across each tank was 2.20 volts.

With a view to reducing the quantity of impure copper produced in the non-circulating tanks an experiment was run to determine how far the copper content of the ordinary electrolyte could be reduced without impurities being deposited. It was found that 7 per cent $CuSO_4 . 5 H_2O$ was about the limit of safety, as below that figure the impurities would precipitate with the copper.

The author draws the following conclusions from the experiments: (1) Using a current density of 7 amp per square foot of cathode surface at 7 per cent $CuSO_4 . 5 H_2O$ and up to 14 per cent free H_2SO_4 (starting with 13 per cent copper sulphate and 12 per cent free acid) the cathodes darken and bismuth and arsenic begin to deposit. On two occasions, when circulation stopped during the night, it was found that at 8 per cent copper sulphate the cathodes were bright, at 7 per cent dull and at 5.5 per cent quite black. (2) Using the same current density in a non-circulating solution with insoluble anodes arsenic was removed at a regular pace. Starting with 0.54 g. As_2O_3 per liter it required 12 days to lower it to 0.15 g., whereas most of the bismuth was removed in the first two days and the copper sulphate never got below 2 per cent. With 14 amp the solution was reduced to 1.7 per cent copper sulphate and the arsenic to 0.015 g. in six days.

The suggestions offered are that the copper content of the

electrolyte be reduced in lead anode tanks to the safety mark before treatment in non-circulating tanks. The whole of the impurities would then be concentrated in a small amount of copper which could be roasted with a small amount of matte and blown to blister separately, thus eliminating the arsenic to a large degree. Or the impure copper could be melted and treated as anodes in a separate tank and the cathodes, so produced, removed as long as they remained pure. The resultant solution could be concentrated and crystallized as long as pure crystals were obtainable, and the mother liquor either discarded or precipitated on scrap iron and the cement copper rich in impurities disposed of to the best advantage. In any case it would be better to discard a small quantity of copper than to return the impurities to the tank house.

Occluded Gases in the Alloys of Copper.—Messrs G. Guillemin and Delachanal have made extensive studies of the occluded gases in copper alloys, and they report on the results of their investigations in the *Revue de Metallurgie* of January, 1911, as follows:

Tin, phosphor-bronzes and brass, as used in the industries and naval construction, were tested for occluded gases. For these tests quartz tubes were used, as the porcelain tubes were found to be too fragile. The opaque quartz tubes were first tested as to their impermeability to gases, both at a temperature of 1100° C. and in the cold state. In the cold they were impermeable. They were then kept under a vacuum at 1100° for ten hours, and only very slight quantities of gas were evolved, consisting of carbon dioxide and some traces of organic matter, which had entered the tubes in spite of all precautions. The error due to this is, however, so slight that it is negligible. The conclusions derived from the test are as follows: Special forgeable brasses retain important volumes of occuluded gases (1 to 30 vol.) composed of hydrogen, carbon dioxide and carbon monoxide. In sound samples the of occluded gases (1 to 30 vol.) composed of hydrogen, carhydrogen. In unsound cast samples the hydrogen is accompanied by a considerable portion of carbon monoxide and a little carbon dioxide. In sound pieces the presence of occluded hydrogen, even in considerable proportions, seems to have no bad influence on the mechanical properties of brasses which can be forged. Phosphor-bronzes have very little occluded gas; this is composed entirely of carbon dioxide and hydrogen. (As in the case of cast iron and steel, phosphorus seems, therefore, to diminish the solubility of hydrogen in the metal.) The occluded gases can only be extracted by employing an increased temperature and at least a partial volume. The article gives a complete table showing the composition of the pieces tested, the volume of gas extracted, composition of the gases, which were CO_2, H, CH, CO, and some of the rare gases, like argon, etc.

Zinc.

Recovery of Zinc in Liquid State Direct from Zinc Blende.—F. Thomas, in *Metallurgie* of Nov. 22, 1910, reports on the results of tests made for the purpose of obtaining zinc in liquid state direct from blende by electrolysis. A very fusible silicate mixture whose melting point is below the boiling point of zinc, 920°, was first thought of, and for this purpose silicates of Mn, Al, Mg, Ca, Sr, Ba, Na, K were picked out. The curve of the melting points of mixtures of Na_4SiO_4 and K_4SiO_4 shows a minimum (743° C.) for 50 per cent Na_4SiO_4 and 50 per cent K_4SiO_4. The further addition of 6 per cent manganese silicate reduced the melting point to 700°. The solubility of ZnS in this silicate mixture was tested up to 1000°. At 900°, the maximum temperature which can be used lest zinc is lost by evaporation, only 0.5 per cent zinc sulphide is dissolved in the silicate mixture. Electrolysis of ZnS in this silicate mixture was then tried in two furnaces of Borchers. The result of all the tests was the same, namely, a fine distribution of zinc globules throughout the mass and a separation of a very small regulus in two tests only. The probable reasons for failure are the formation of a collodial zinc dust from the cathode and the slow speed of solution of ZnS in the silicate mixture at temperatures below 900° C. Electrolysis under pressure was then tried in order to employ higher temperatures, and in this way 52.6 and 67 per cent zinc were recovered, although mostly in form of zinc dust. However, these tests were given up as not promising success. Further tests were then made, trying a solution of ZnS in a Na_2S—FeS mixture. The intended reactions were $Fe_2O_3 + 2\ Na_2SO_4 + 11\ C = 2\ (Fe + Na_2S) + 11\ CO$ and $Fe + Na_2S + ZnS = (FeS + Na_2S) + Zn$. Fe_2O_3 and Na_2SO_4 were reduced at 1100°. Fifty grams of the reduced mixture were mixed with 36.15 grams ZnS and heated at a constant temperature for a long time. At temperatures below 920°-960° a precipitation of zinc by iron dirt did not take place. Tests in which the fusibility was increased by adding $CaFe_2$, and the intention was to increase the speed of reaction by an increase of the Fe content, had a negative success.

Tests were then made to separate zinc under pressure at higher temperatures. The zinc deposited on the cover of the crucible in form of dust. In one test, which lasted 30 minutes and in which the pressure was 2-3 atmospheres, 55 per cent zinc was recovered as dust. The reaction between the Fe and S takes place near the highly heated carbon resistor, and the Zn rises as vapor upward along the carbon resistor, which is probably so hot that at its temperature the partial pressure of the zinc vapor is higher than the pressure in the electric furnace. The vapor rises, therefore, to the surface and condenses on the relatively cool walls of the iron crucible and crucible cover in form of dust.

The last tests were made with electrolysis of ZnS solution in a fused sodium sulphide-iron sulphide mixture, using Fe-anodes. The greatest percentage of metal recovered was 14.7 per cent; there was again the formation of a zinc mist or fog, which extended from the cathode. Although the results are negative, the paper is interesting.

Assaying.

A Portable Assay Outfit for Field Work.—Engineers who have had occasion to prospect in isolated regions where assay offices are few and far between will be interested in a contribution to the January, 1911, *Bulletin* of the Amer. Inst. of Min. Eng. by S. K. Bradford. He gives full details of equipment and procedure and shows the ingenuity which can be exercised when ordinary laboratory facilities are not at hand. The equipment is packed in an ordinary 26-in. valise and suffices for about 100 assays of commercial accuracy.

Equipment.—Small hammer; steel mortar with stand sawed off flat to serve as an anvil; small steel pestle; blow pipe and blow-pipe cupel holder; small and large spatulas; button pliers; light 1-in. cupel mold; two 3.5-in. x 8-in. muffles; 18 "F" crucibles; six crucible covers; crucible tongs; 25-cc. burette; six 0.5-oz. parting flasks; six 0.5-oz. annealing cups; two or three beakers; hand balance, folding in a tin box; set of gram weights; one 6-in., 60-mesh screen; six small glass stirring rods; teaspoon and tablespoon; a micrometer for measuring diameter of buttons. The latter may be a short fixed microscope stand in which has been placed a scale of 0.1 mm—two or three extra object glasses being carried for safety. Chemical reagents in bottles in metal containers: 0.5 lb. KCN; 1 lb. C.P. HNO_3; 1 lb. C.P. HCl and 2 lb. aqua ammonia. A few ounces of sheet copper; 1 oz. C.P. sheet silver; 0.5 lb sheet lead; 2 lb. tenpenny wire nails; 5 lb. bone ash; 5 lb. sodium bicarbonate; 5 lb. litharge; 3 lb. red lead; prepared charcoal for blow-pipe use; candles and old newspapers.

It will be possible to obtain sodium bicarbonate at country stores and bone ash may, on occasion, be made in the field. Sheet lead may be made from bullets and silver can be obtained from coins of known fineness.

The assays can be fused in a blacksmith's forge, which is to be found at every mining camp, and the cupelling can be done at the same place by setting up the small muffle mentioned. The weight of the buttons obtained can be determined by meas-

uring them and referring to a previously prepared scale based on specific gravity multiplied by solid contents. This method of determining weights can be found in the *Transactions* of the A. I. M. E., xxxi, 800 (1901).

Recent Metallurgical Patents.

Gold and Silver.

Treatment of Auriferous Sand.—A process for treating black sand, concentrate and the like has been patented by Carl F. Siegert, of Jeppes, near Johannesburg, Transvaal. The invention consists in roasting the refractory material, or heating it to a dull red heat, and quenching it in a solution of sodium and ammonium chlorides to free the gold from oxides. The solution is then evaporated and sulphuric acid added to the comparatively dry mass. Water is subsequently added and the mass mixed. After the dilute acid has remained in contact with the ore for a suitable time it is drawn off and the mass is washed with water. It is claimed that this treatment will render the precious metal content free for recovery by amalgamation in the usual manner.

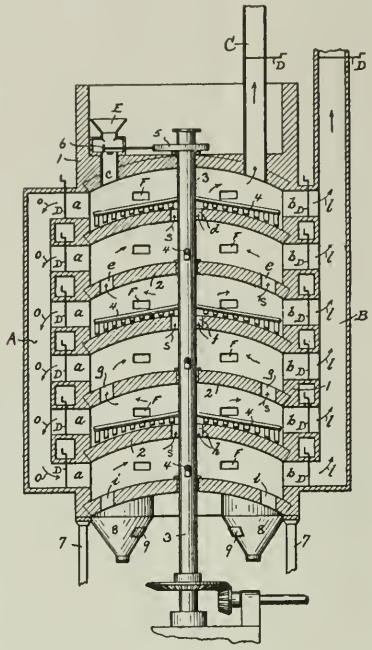
For each ton of material the inventor uses about 42 gal. of water and 10 lb. NaCl, to which is added 2 gal. of strong salammoniac solution made by dissolving 2.5 lb. to 3 lb. salammoniac in 100 gal. of water. This solution is used for the first treatment. For the acid wash he uses 1 lb. of sulphuric acid, more or less, as needed, and adds water after the acid has been poured on the mass. (981,696, Jan. 17, 1911.)

Roasting.

Improved Roasting Process.—For the purpose of improving the quality of gases produced in roasting sulphide ores for the manufacture of sulphuric acid, Charles W. Renwick, of Isabella, Tenn., has patented a furnace of novel construction. In the usual roasting furnace the ore is fed at the top and passes downward over a series of hearths to a discharge beneath, and the air is admitted at the bottom and rises through the several hearths to a stack at the top. Metallic sulphides are thus roasted to oxides and as they are catalytic agents they convert SO_2 into SO_3 in quantities varying with the conditions. In the ordinary procedure described the inventor says that the most limited catalysis is produced. It is his object to reverse the current of gases and cause them to flow down with the ore in order to carry the high temperature found in the upper hearths down to the lower for the better roasting of the ore and in order to have the SO_2 come into more frequent contact with the metallic oxides so that more SO_3 will be formed .

The accompanying drawing, Fig. 1, illustrates his preferred form of apparatus. It consists of an ordinary furnace shell with two vertical flues on opposite sides, one for drawing off all or a part of the gases from any of the upper hearths and entering them into the lower, and one for drawing off all or part of the gases, depending on the amount of conversion desired, and conducting them to the place of use.

In addition to the usual construction of the type of furnace shown, the points of which are generally understood, the improved type embodies flues *A* and *B* placed at opposite sides of the furnace body and having branches *a, b*, provided with dampers *D* which communicate through openings in the furnace shell with the several hearths, so that when the damper *D* in the stack *C* is closed, and the doors *F* properly opened for the admission of air the gases can be taken from any of the upper hearths and directed into those below and passed into the flue *B;* or they can circulate up in the ordinary way after subjecting the desired quantity of SO_2 to catalytic action, the damper in the stack *C* being opened for this purpose. By closing the dampered openings and all the dampers *D* except the one in the lowermost branch flue *b*, the entire draft in the furnace will be downward and through the openings *d, e, f, g, h*, and upward through the flue *B*, air being admitted at the top. All or a part of the gases can be taken from any of the hearths and turned down through flue *A* to any of the lower hearths and carried over to flue *B*, the dampers *D* being open in the branch flues *a, b*, at the point where it is desired that the gases shall pass out of certain of the hearths and into certain of the others and thence on to flue *B* or stack *C* as desired.



ROASTING FURNACE

The inventor further claims that the reversal of the gases will increase the temperature of the gases on the lower hearths and thus break up all sulphates and give a sweet roast, in addition to increasing the conversion of SO_2 to SO_3. (981,880, Jan. 17, 1911.)

Alloys.

Magnesium-Zinc Alloy.—The development of the aeroplane and airship has introduced a demand for light and strong metals or alloys. Walter Rübel, of Berlin, Germany, has recently patented a new alloy of magnesium and zinc which, in his specifications, he claims has a specific gravity of 1.8; when cast, presents a resistance of 15 kilos to 25 kilos; when rolled, from 25 kilos to 35 kilos, with a coefficient of expansion of from 8 per cent to 12 per cent. Satisfactory alloys of these metals cannot be produced from more than 99 per cent nor less than 94 per cent magnesium, and less than 1 per cent nor more than 6 per cent zinc. In case of materials that are to be rolled or pressed the zinc should preferably not exceed 3 per cent. Pure zinc is not essential, as good alloys can be made if the zinc contains copper, aluminium or metals of the iron group, as iron, nickel, chromium or manganese.

The alloy is prepared by carefully melting the magnesium with the exclusion of air and adding the zinc, when a readily fusible material results, which can be cast, wrought, rolled or machined like brass. The alloy presents the fracture of fine steel. (982,022, Jan. 17, 1911.)

The stack at the Great Falls (Mont.) smelter is 506 ft. high and weighs 34,000,000 lb. About 16,000 tons of brick were used in its construction. It will withstand a wind of 120 miles per hour.

New Methods of Obtaining Hydrogen.

In the Linde-Frank-Caro process for the separation of hydrogen from water-gas the basis of the process is the production in the first instance of water-gas. As is well known this is obtained by passing steam over red hot coke whereby the former is disassociated into free hydrogen and oxygen. The oxygen combines with the carbon of the incandescent coke to form carbon monoxide, while the hydrogen remains in a free state. In addition to these two gases, commercial water-gas also contains traces of carbonic acid and nitrogen to the extent of from 3 to 5 per cent of the total volume of the gas. Water-gas consists apart from these impurities of about equal parts of hydrogen and carbon monoxide.

There are a number of ways for the chemical production of hydrogen but by the application of fractional condensation it may also be obtained in the form of a gas by means of a very simple mechanical process. Separation of the carbon monoxide from the water-gas may be brought about by cooling the water-gas to the temperature of liquid air whereby the carbon monoxide is liquified—the conditions of liquification approximating to those of nitrogen—while as is well known much lower temperatures are necessary to liquify hydrogen.

In order to carry out this process on commercial lines, the carbonic acid in the water-gas is in the first instance extracted in some convenient manner, such for example as its absorption by caustic soda. The water-gas is then compressed to a pressure of 60 atmospheres by means of a three-stage compressor. The gas is then dried and passed into an apparatus in which it is subjected to a bath of liquid air. As the result of cooling in this bath of liquid air, the carbon monoxide liquifies and the hydrogen passes out of the apparatus in the form of a gas.

The carbon monoxide may be re-evaporated or otherwise collected as a by-product and utilized in other ways. It may be employed to drive a gas engine or it may be used as fuel under a steam boiler. By using it for power purposes the whole process becomes very economical as in this way the power necessary for the fractionation of the gas can be obtained.

The process is very inexpensive to operate and the apparatus requires very little attention.

The rights in the process for this country have been acquired by The Linde Air Products Company of Buffalo.

* * *

In this connection a paper by Director Richard Blum, of the Berlin-Anhaltische Maschinenbau Company, read before the Aero Club in Berlin is interesting, as the author discusses the different processes now available for producing hydrogen gas, with special reference to the filling of balloons. The paper is printed in full in the *Journal für Gasbeleuchtung* of Jan. 21 and 28.

The oldest methods of making hydrogen gas on a practical scale, used before the eighteenth century, were based either on the action between sulphuric acid on iron or on the passing of steam over iron filings heated to a temperature of red heat. The latter process has recently been taken up again and changed to a cyclic process by the Internationale Wasserstoff Company. They employ steel retorts filled with porous iron oxide. The retorts are placed in a furnace and are heated from the outside. When the temperature has risen to a sufficient degree water gas is passed through the retorts and the iron oxide is thereby reduced to a metallic iron. When all the iron oxide is reduced to a porous mass of iron, the passage of water gas is stopped and steam is passed through the retorts whereby hydrogen is formed and the iron is again oxidized. Then the passage of steam is stopped and water gas is again passed through the retorts and so on. In this way hydrogen is produced and the iron oxidized, and in the next stage the iron oxide is reduced again to metallic iron. It is advantageous to keep the furnaces in continuous operation. The hydrogen thus produced is very pure and is compressed in steel cylinders of 6 cu. m each to 150 or 170 atmospheres.

Numerous plants are in operation in Europe for the decomposition of water into hydrogen and oxygen by electrolysis. In an apparatus for this purpose built by the Schuckert Company the production of 1 cu. m of hydrogen and ½ cu. m of oxygen requires 13 hp-hours. According to the present author the process is expensive and requires a large floor space. But the hydrogen produced is very pure.

Enormous amounts of hydrogen gas are produced in the large works for the production of caustic soda and chlorine by electrolysis of common salt solutions. The hydrogen gas is set free together with the caustic soda at the cathode and the quantity of hydrogen gas produced is chemically equivalent to the quantity of the chlorine produced at the anode. The quantities are so large, compared with the demand which exists at present for hydrogen, that most of the hydrogen gas is passed unused into the air. The Griesheim-Elektron Company in Germany produces daily 20,000 cu. m of hydrogen gas of about 90 per cent to 97 per cent purity. In this case the cost of the gas is practically that of compressing and storing the hydrogen gas. Special railway cars are built in Germany for the transportation of 500 cylinders containing 2750 cu. m of hydrogen gas. Much of the gas used in balloon trips in Germany is procured in this way.

Another method for producing hydrogen gas is by means of certain metals or compounds which in contact with water or alkaline solutions develop hydrogen.[1] Such materials are aluminium, silicon, and calcium hydride.

The production of hydrogen from aluminium and caustic soda solutions was used in practice by the Russian army in their war with Japan. This method is expensive.

More recently the Griesheim-Elektron Company has placed on the market a so-called "activized" aluminium (aktiviertes aluminium), 1.25 kg of which yield 1 cu. m of hydrogen. In this case the aluminium is stirred with water in the gas producer. According to statements of the manufacturing company the cost of producing hydrogen by this method is about 12.5 cents per cubic meter.

The Siemens & Schuckert Company has worked out a process for the production of hydrogen from the reaction between silicon and caustic soda solution. Formerly steam was employed, but now the heat set free during the reaction is utilized for maintaining the proper conditions. The evolution of hydrogen gas takes place when a 25 per cent solution of caustic soda acts on silicon introduced in small quantities. The capacity of a transportable plant (placed on wagons and drawn by two horses) is 60 cu. m to 120 cu. m per hour, while stationary plants are built for capacities up to 300 cu. m per hour. The Italian army has such plants in operation. The process is elegant, but the cost is about 18.75 cents per cubic meter.

For the filling of balloons it is of great importance to have the hydrogen as cheap as possible, say, at a price of 2 cents to 4 cents per cubic meter. This is possible by some newer processes. The first one of these mentioned by Mr. Blum is that of Linde, Frank & Caro for producing hydrogen from water gas with the aid of fractional condensation. This process is described above. In Mr. Blum's paper it is mentioned that under conditions as that exist in Germany the cost of the production of hydrogen gas by this method ranges from 2 cents to 3.25 cents per cubic meter, according to the purity of the gas and the size of the plant.

Another process worked out by the Carbonium Company in Germany employs acetylene gas which is compressed to two atmospheres and exploded by an electric spark. The acetylene thereby dissociates into the elements carbon and hydrogen. The carbon deposits in form of lamp-black. The hydrogen is passed through large washers and stored. The German plant of this company was destroyed last summer by an explosion A condition for getting the hydrogen cheaply by this method is that there is a market for the lamp-black.

[1] The production of hydrogen gas from hydrone (made by the Roessler & Hasslacher Chemical Company) belongs to the same class.

Another method of the Berlin-Anhaltische Maschinenbau Company employs the system of Rincker & Wolter. The starting materials are oils, petroleum, benzine or any heavy hydrocarbons. The process has been first developed as a substitute for illuminating gas plants using bituminous coal. These plants can be so equipped that they produce in general illuminating gas and gas for heating and cooking, and in exceptional cases hydrogen gas.

Plants of this kind are also instilled on railway cars to ship them to any place where the hydrogen may be wanted. There are two gas producers lined with fire-brick. They are filled to three-quarters with coke. By means of a steam turbine and a blower air is blown into the gas producers and the coke blown red hot. The turbo blower is then stopped by means of an oil pump and oil is sprayed into a generator. The oil is gasefied at once with deposition of lamp-black, while the hydrogen gas is passed through a washer and cooler and dryer, etc. The capacity of such plants is about 100 cu. m or more per hour. The time required for putting the plant in operation is two or three hours; it is then able to produce hydrogen gas continuously.

Hydrogen gas plants are furnished from Germany to Austria, Italy, and Russia, so that there seems to be a market for hydrogen in Europe. In the past it has been restricted in this country.

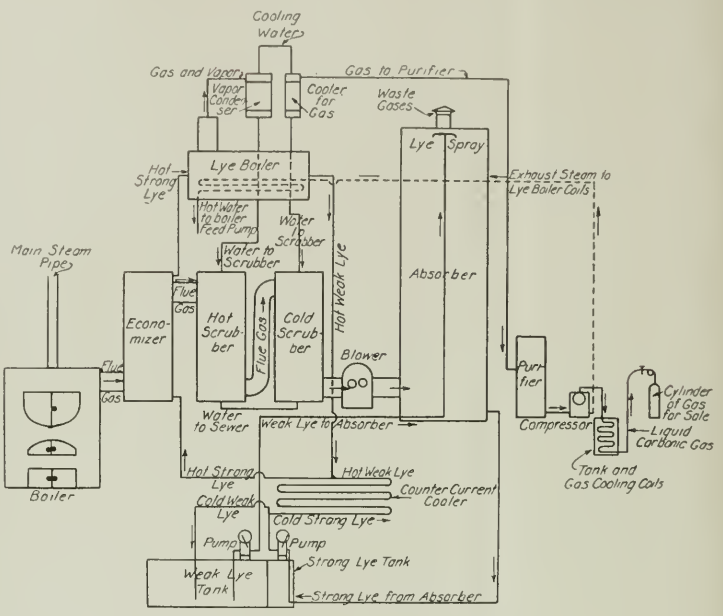


DIAGRAM OF CARBON DIOXIDE MANUFACTURING PLANT.

A Problem in Saving Heat in the Manufacture of Carbon Dioxide.

By Sterling H. Bunnell.

A very large business is done in the manufacture of carbonic gas for use by bottlers and others. This gas, pure carbon dioxide, is obtained from the combustion of coke, washed to take out sulphur and other impurities, separated from the other flue gases, dried, compressed to the liquefying pressure and bottled up in steel drums or cylinders for sale. The problems which must be solved in connection with making an economical plant for the manufacture of liquid carbonic gas entail almost every form of steam plant engineering.

The first operation in manufacturing is to burn the foundry coke under a steam boiler. A deep fire is carried and a forced draft used, so that the flue gases may have the highest possible proportion of carbon dioxide. With a properly designed furnace and good firing the proportion of carbon dioxide will run above 14 per cent and may at times reach above 16 per cent. The flue gases leaving the boiler setting are at a high temperature; this heat must all be disposed of before the gas is in condition for absorption. Here is the first point at which waste heat is to be saved. Accordingly, the flue gases are passed through an economizer, the duty of which will be explained later. The gases leave the economizer at a temperature below the boiling point of water.

The gases next pass through two scrubbers in succession, in which they are washed by showers of water to get rid of the cinders and whatever sulphur may be present. In this operation they are cooled to the temperature of the water supplied and this portion of the heat is necessarily wasted. The gases next pass through an absorber, in which they are brought in contact with lye solution. This solution has the property of absorbing carbon dioxide, so that the flue gases emerge from the absorber with a content of carbon dioxide of 10 per cent.

The lye solution from the absorbing tower is allowed to flow into a tank. This part of the solution is called the "strong" lye, because of its containing a considerable quantity of carbon dioxide. The next step is to pump the strong lye into a lye boiler, where it is heated above the boiling point, a portion of the water is evaporated, and a large part of the absorbed carbon dioxide driven off by the heat. Here again is a point at which heat must be applied continuously to a solution, and as the temperature required is little if any higher than 212°, exhaust steam from the power portion of the plant is used for the purpose of supplying heat. The lye solution returns from the lye boiler to a second tank and as its contents of carbon dioxide have been much reduced this solution is now known as the "weak" lye. In order to permit the weak lye to be again used for absorbing a fresh supply of carbon dioxide it must be cooled from its boiling temperature to perhaps 120°.

Observing that the weak lye from the absorber is cooled and must be heated, and the strong lye from the lye boiler is hot and must be cooled, the natural suggestion is to exchange the heat between the two solutions through a countercurrent cooler. This is of the double-pipe type, the hot weak lye from the lye boiler flowing through the inner pipes and the cold strong lye from the strong lye tank flowing through the outer pipes in the opposite direction. In this way an exchange of heat takes place which returns the weak lye at a temperature nearly as low as the temperature of the strong lye on its way from the tank, and delivers the strong lye at a temperature nearly as high as that of the weak lye coming from the boiler.

The strong lye leaves the countercurrent cooler at a temperature which is still considerably below the boiling point, and here comes in the opportunity to use the waste heat of the flue gases from the steam boiler by passing the strong lye through the economizer before mentioned. The strong lye leaves the economizer at a temperature near to boiling, and in passing

the economizer abstracts a large part of the waste heat in the flue gases.

All the waste heat in flue gases has now been accounted for. The bulk of the heat caused by the combustion of the coke has gone into the steam boiler and been devoted to the work of evaporation. This affords a good supply of steam available for doing the work of the plant. Most of the work required is for the compressors, which take the gas after it has been driven off the hot strong lye in the lye boiler and successively passed through coolers and dryers. Another portion of the steam is required for operating the pumps necessary for circulating the lye and for pumping cooling water about the plant. The exhaust steam from compressor and pumping engine still carries the latent heat which it received in the steam boiler. This large supply of heat is devoted to heating the lye in the lye boiler, evaporating a portion of the water and driving off the gas.

The skill required in designing and operating the entire plant is obviously devoted to arranging the closest possible balance between the heat transfers in all parts of the plant. The capacity of the plant—that is, the amount of carbonic gas that can be put into cylinders for sale—depends first upon the amount which can be driven off the lye in the lye boiler; a step further back, upon the proportion of the gas which can be recovered from the flue gases in the absorber; a step back of this, upon the proportion of the coke which can be burned into carbon dioxide rather than carbon monoxide, and this again depends upon the efficiency of the firing, the thickness of the bed of coke and the regulation of the draft, so that no free air shall pass through the fire. If the absorption of the gas is efficient the amount of coke burned, and therefore the grate surface, can be decreased. This will decrease the amount of steam made by the boiler, the available power of the plant and the amount of heat which can be recovered from the exhaust steam. Evidently, with a very efficient absorption it might be possible to produce enough gas from so little coke that there would not be steam enough to operate the plant, and in that case every possible saving in steam consumption by engines and all waste by radiation by steam pipes should be cut down as far as possible.

The operation of a complete modern plant for manufacturing carbonic gas would be a profitable study for any steam engineer interested in getting the most out of his steam plant. It is safe to say that very few steam plants of sizes in the neighborhood of 100 hp are constructed with any such degree of effort to reduce the usual losses to the minimum as is observed in the design and operation of the modern carbonic acid gas plant here described.

Griscom-Spencer Company,
New York City.

The ball-tube mill is being advocated by cyanide metallurgists as a machine that probably will give good results in preliminary crushing and grinding. These machines have given good results in the cement industry, and it is suggested that they could replace rolls in certain metallurgical operations. Either rolls or ball-tube mills would find preference with American engineers, who regard the use of the stamp as a coarse crusher as inferior to the machines mentioned.

The American Smelting & Refining Company has issued a statement showing a comparison of the condensed balance sheets for Oct. 31, 1910, and April 30, 1910. A similar statement is issued for the American Smelters Securities Company. Both statements show an improvement in earnings as compared with the previous year, and an increase in the quantity of metal held in process. President Guggenheim states that the April, 1911, statement will be more than usually extensive and comprehensive. He further adds that the supply of ore coming to the company's plants is showing a marked improvement, and that they have been encouraged to increase expenditures for improvement and new construction.

Centrifugal Clarifier and Filter

An article published elsewhere in this issue (page 147) contains interesting advance information on a centrifugal machine just perfected after many years' work by the DeLaval Separator Company for the continuous separation of liquids and solids. A smaller machine of the same company, which has solved a special problem of the separation of solids from liquids—namely, the separation of a small amount of sediment, etc., from a solution—has already stood the test of practice in the clarification and filtration of varnishes and allied products, oils, pharmaceuticals, patent medicines, fruit syrups and extracts, in fact, of many liquids containing either a sediment deposit or the finest flock or both. But as this centrifugal machine will undoubtedly prove useful beyond its present already large sphere of work the following notes on its construction and operation should prove welcome:

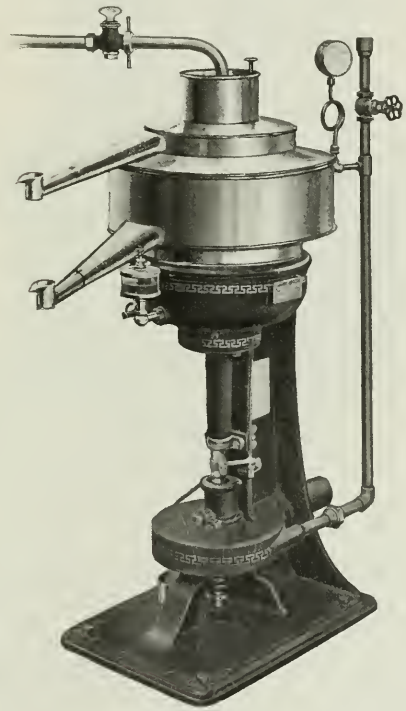


FIG. 1.—STEAM-TURBINE CLARIFIER AND FILTER.

An outside view of this machine is given in Fig. 1, which shows the steam-turbine style for direct steam connection. The largest size of this type has a height of 46 in. and requires a floor space of 18 in. by 24 in.—which indicates a big saving in floor space over other separating methods. In works where shafting is already installed the machine may be used with belt drive, while the smaller sizes of the machine are entirely practicable for running by hand.

Fig. 2 is a section of the clarifier and filter bowl. It contains the two compartments where the work is done, namely, the lower clarifying compartment and the upper filtering chamber. The clarifying compartment consists of a cylinder filled with a series of conical-shaped, pressed-steel disks, one above the other, about 1/32 in. apart and held rigidly in position by a central feeding device. The filtering chamber or upper section is also cylindrical in shape, but of less diameter.

The liquid first enters the bowl at the point *A* through the feeding device in the center and drops to the bottom of the clarifying compartment *B*, where much sediment is retained. It is then forced outward by the great centrifugal force through openings at the bottom to the periphery of the compartment *C*,

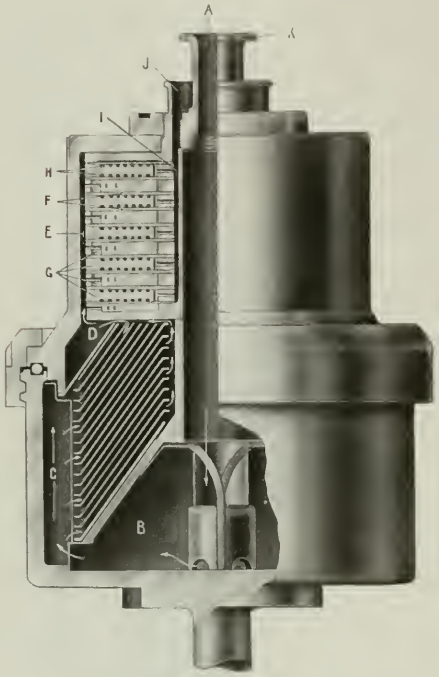


FIG. 2.—BOWL OF CLARIFIER AND FILTER.

where the bulk of heavier sediment is deposited. The liquid is next drawn to the center between the several disks. The purpose of these conical disks is to divide the liquid into thin layers, thus the friction on the disks above and below causes it to revolve rapidly and the heavier part, or balance of the sediment, is gradually forced to the under side of the disk above, whence down and out to the pocket *C*, while the thoroughly clarified liquid passes to the center and is forced upward and out into the second or filtering chamber at *D*. When the liquid enters this compartment fully 99 per cent of the sediment and foreign matter has been removed, thus leaving a small though very important work for the filter to accomplish.

The type used is what is termed the mutiple filter. The filter chamber is filled with a series of horizontal corrugated plates and perforated retaining rings, between which are placed one or more thicknesses of filter material, paper, felt, cloth or whatever may be best adapted to the work in hand. The liquid is forced from compartment *D* upward into the compartment *E*, which is a narrow space between the periphery of the filtering chamber and the filtering disks. This allows a channel for the liquid to pass up, from which it is delivered into the several compartments *F*, where it is forced through the filtering material *G* into compartment *H*. Here it is again forced to the center and upward through chanel *I* to the discharge point *J* and into the large cover from which it flows into the receptacle awaiting it.

The functions of the clarifying and filtering compartments supplement each other very thoroughly. The liquid is so thoroughly clarified by the disk system in the lower clarifying compartment that it leaves but a small amount of work for the filter to perform. Then, too, the filters being placed horizontally are substantially self-cleaning, for as the sediment or flock forms in any volume on the filters the centrifugal force throws it off and it is deposited at the outside or periphery of the compartments. Thus the filter material will do its work very much longer than is possible otherwise and continue to deliver a product of the required fineness.

This machine is built by the DeLaval Separator Company, 165 Broadway, New York City. The manufacturers state that with due care this clarifier and filter will last a lifetime. The average operator finds no trouble whatever in adjusting or cleaning any of the machine's parts and keeping it ever ready to perform its work. The chief advantages claimed for this machine are as follows:

The DeLaval clarifier and filter will take any product to which it is adapted immediately after compounding and deliver continuously (except for occasional cleaning of the machine) a finished article ready for the market. It thus greatly reduces the capital necessarily tied up in settling tanks with other methods.

On an average it will make a saving of 90 per cent in filter paper and cloths and their subsequent washing and cleaning over that required by other processes, also a saving of liquid absorbed by the filter paper and that lost by evaporation or dripping with other methods, a saving of liquid, labor and the

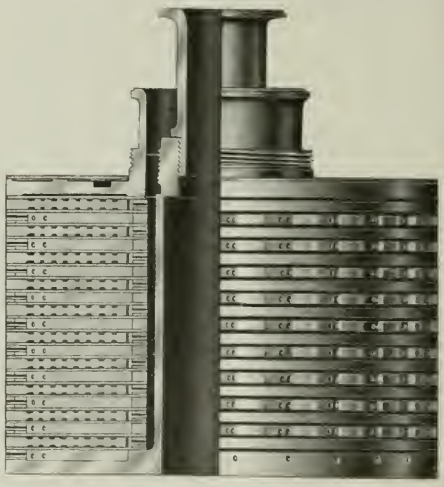


FIG. 3.—FILTERING CHAMBER.

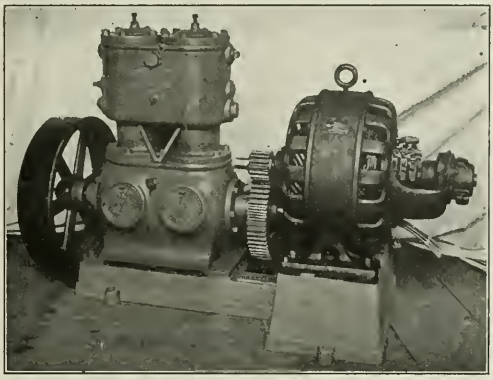
losses that sometimes occur from punctures in paper and a big saving in floor space.

Where it is desirable to aerate the products to be handled the DeLaval machine can be made to automatically aerate the liquids as they pass through it and thus do away entirely with the necessity of air pumps or other equipment and labor necessary for aerating purposes with other filtering processes.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in November, 1910, was 69. The total tonnage milled during that period was 1,843,941 tons of ore; 9746 stamps were in operation with an average duty of 7.32 tons per 24 hours. Tube mills in commission numbered 194. The yield for the month was 642,591 fine ounces of gold. Compared with the preceding month the only change of importance was in the increased number of tube mills in operation.

Sulphuric Acid Handled by Compressed Air.

In the manufacture of certain chemicals considerable danger, difficulty and expense are caused by the rapid deterioration of piping and receptacles with which the acids and fumes come in



MOTOR-DRIVEN AIR COMPRESSOR.

contact. The difficulties which the destructiveness of sulphuric acid, for instance, causes in pumping are well known.

The most satisfactory method of pumping destructive acids is that used by the Nichols Syndicate in their new plant at Bay Point, Cal. At one stage of the process of manufacture the acid is accumulated in a cylindrical tank which is provided with suitable inlet and outlet valves. When it is desired to pump the acid into other retainers the inlet valve is closed and air at from 60 lb. to 100 lb. pressure is admitted, which forces the liquid through the outlet valve to other parts of the plant. The flow of the acid is controlled by suitable valves in the piping system.

To meet the increasing demand for compressed-air apparatus for such work the United Iron Works, of Oakland, Cal., have brought out the motor-compressor shown in the above illustration. The compressor is a 6 x 6 Gardner Rix Duplex; speed, 2000 r.p.m.; pressure, 60 lb. to 100 lb.; capacity, 40 cu. ft. of air per minute. The compressor is driven by a Westinghouse type HF three-phase, 60-cycle, 550-volt motor.

This type of motor is admirably suited to this work. The secondary is phase wound and the three phases are connected to the three collector rings shown. In starting, the controller gradually cuts out the starting resistance which is connected to the secondary through the collector rings. This motor is so designed that an unusually high starting torque is developed, with comparatively low starting current.

The European Spelter Convention has announced that present prices will be maintained through March, but that on April 1, 1911, there will be an advance of 0.25 mark per kilogram. A graduated price list, with concessions for wholesale purchasers, will be adopted.

Determining the Accuracy of Indicator Springs

By E. K. Uebe.

The indicator is unquestionably the most necessary instrument for the operating engineer in a steam plant.

The indicator diagram is the only means by which to ascertain the action of the steam in the engine cylinder, to determine the efficiency of an engine and to detect internal wastes. It is, therefore, of vital importance that the indicator can be relied upon to be absolutely accurate in its performance, which especially applies to the piston spring, the most essential part of the indicator.

Unless the exact tension of the spring is known errors will result from its use and the determinations made with the indicator will be of little or no value. An instrument for testing indicator springs has recently been placed on the market by the Schaeffer & Budenberg Manufacturing Company, of Brooklyn, N. Y.

The instrument, a general view of which is given in Fig. 1, consists essentially of a closed vessel made of cast iron capable of resisting internal steam pressures up to 200 lb. per square inch. The steam pressure in the vessel is measured by a gage of special construction consisting of a piston of ½ sq. in. area and which is free to move in an inclosing cylinder.

The lower portion of the piston is pointed and rests in a yoke which is suspended on the knife-edge of a pair of scales mounted on top of the closed vessel. If the scales are previously balanced, before admitting steam into the vessel, it is evident that the reading of the scales will give the pressure

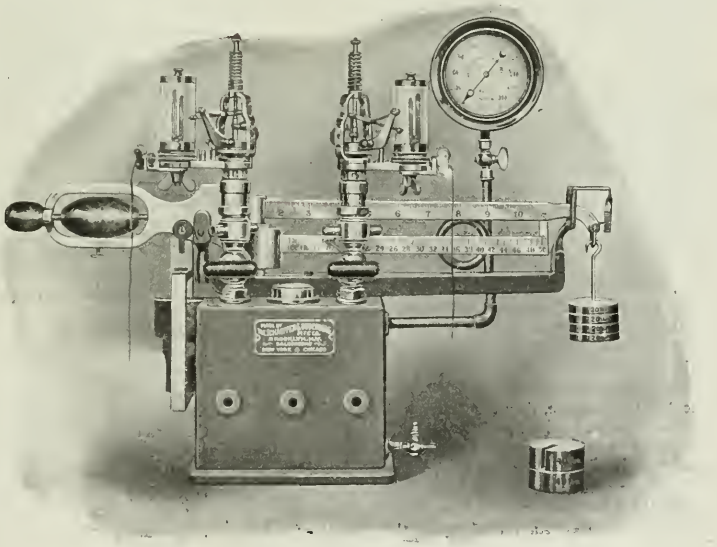


FIG. 1.—APPARATUS FOR DETERMINING THE ACCURACY OF INDICATOR SPRINGS.

acting on each element of the vessel equal in area to that of the piston. The scales are graduated to fiftieths of a pound, which permits of very close readings.

In testing indicator springs steam is turned on into the vessel to gradually warm it and allowed to blow through the connections and indicator cocks to remove any particles of dust or grit which may have accumulated there. Steam is then shut off again and the indicator screwed in its place, putting paper on the indicator drum, on which paper two vertical lines are ruled as 1-2 and 3-4 (see Fig. 2).

The indicator pencil is pressed against the drum and a horizontal line *Q Q* is drawn at the point thus marked. Then

the poise of the scale is set, say, 5 lb. and steam is admitted into the vessel, gradually rising in pressure until the scale floats. When the pressure is exact the line *R R* is drawn, and so on for successive pressures of 5 lb., until the limit of the scale is reached. During this operation the motion of the indicator pencil is continuously kept upward. If at any moment the pressure rises too high it is lowered by manipulating the valves below the required amount and then gradually raised to the desired point.

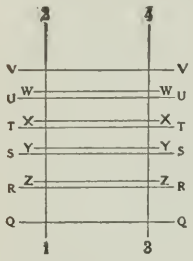


FIG. 2.—RECORDS OF TESTS.

When the indicator pencil has reached its highest position the steam pressure is allowed to fall gradually and a series of similar lines, as *W W*, *X X*, *Y Y* and *Z Z*, drawn during the descent of the piston. The motion being continuously downward. The distance between any two of these lines, as *Z R* or *Y S*, indicates the loss due to friction of the indicator.

The special advantage of this machine, besides its absolute accuracy, is that the indicator springs are tested in the individual indicator in which they are used and under actual steam pressure which is as near to the actual working conditions as possible.

The indicator spring-testing apparatus serves also for testing pressure gages which can be connected to the three openings in the front side of the vessel provided for that purpose, and the readings are verified by the indications on the scales and the test gage shown in the illustration.

A New Recording Gas Meter.

A new meter for indicating, integrating or graphically recording the quantity of flow of gases at any pressure and at any temperature (independent of fluctuations of pressure or temperature) has recently been designed by Prof. Carl C. Thomas, of the University of Wisconsin, and has been placed on the market by the Cutler-Hammer Manufacturing Company, of Milwaukee, Wis.

The operation of the meter is based on a simple and interesting principle which will be understood from the diagram, Fig. 1, which shows a passage *A* through which gas is flowing in the direction of the arrow. Within the passage is an electric heater *B* made of a suitable resistor, and two electric resistance thermometers *E* are provided at the entrance and exit of the passage way. Now suppose that the voltage at the heater terminals remains constant and that the resistance of the heater does not change, then the heater will give off heat to the gas at a constant rate.

If now the rate of flow of the gas remains uniform and constant, the gas is raised by the heater in temperature by some fixed number of degrees, shown by the difference of the readings of the two thermometers *E* and this temperature difference will remain constant so long as the conditions do not change. On the other hand, if the rate of flow of gas increases (while the liberation of heat from the heater remains constant) the larger quantity of gas passing in a given time will be raised less than before in temperature; that is, the temperature difference measured by the two thermometers *E* will be smaller than before. Conversely, if the rate of flow of gas decreases, the temperature difference betwen the two thermometers *E* will increase. In brief, as long as the rate of liberation of heat from the electric heater is kept constant the temperature difference between the two thermometers *E* varies inversely as the rate of flow of gas, and can, therefore, be used as a measure of the rate of flow of gas.

The inherent weakness of this method, as just stated, for commercial service is that it requires the maintenance of an exactly constant rate of liberation of heat from the electric heater. This is a condition difficult to fulfill in commercial service. Therefore, for commercial-service meters the principle of operation has been modified. The change is as simple as it is ingenious.

Not the rate of heat liberation is maintained constant and the temperature difference between the two thermometers *E* measured, but the latter temperature difference is maintained constant artificially by varying the rate of heat liberation from the electric heater—that is, by varying the electrical power supplied to the heater. Hence the temperature difference between the two thermometers *E* remains constant and the varying input of electric power in watts supplied to the heater is measured and serves as a measure of the rate of flow of gas. The watts are measured by recording wattmeter or integrating wattmeter.

The two resistance thermometers are in the form of ruggedly made screens of resistor wire, so disposed as to come in contact with all the gas passing through the meter. They form two arms of a Wheatstone bridge, which by means of the deflection of the galvanometer needle causes the controlling mechanism to cut in or out enough electrical energy so as to keep the temperature difference between the two resistor wires *E* constant. There is no necessity to go further into details as the principle is clearly one of electrical resistance thermometry, which has reached such a high state of excellence in recent years.

The accuracy of these meters is not affected by changes in pressure of the gas or air, since the unit of measurement is that of weight rather than that of volume—that is, the meter takes cognizance of the specific gravity. Also variation of temperature of the incoming gas does not effect the accuracy of the meter because it is a difference of temperature, rather than a fixed temperature, upon which the measurement depends. The meter can, therefore, be used for gas or air at either high

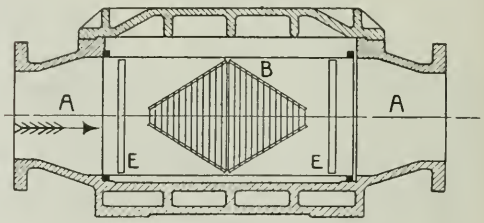


FIG. 1.—DIAGRAM OF GAS METER PRINCIPLE.

or low pressure, and at either high or low temperature, provided the materials used in construction are suited to the conditions.

The meter is especially well adapted for measuring the discharge of gas or air from compressors, blowers, etc., because the small and rapid periodic fluctuations of pressure, due to either suction or discharge conditions, do not interfere with the steady action of the thermometers. The time lag of the latter is sufficient to smooth out the curve of watts input or of temperature variation, as the case may be, and true average results are thus indicated. The fact that the meter is capable of accurately measuring gas under varying pressures and temperatures makes it especially valuable for measuring natural gas, as well as compressed air.

The electrical power required to operate the meter is approximately 1 kw per 75,000 cu. ft. per hour at atmospheric pressure.

The Thomas meter is essentially applicable to the measurement of gas or air which is either dry, saturated or superheated. If water is carried along mechanically as a fog or mist (as may be the case in blast-furnace gases coming from the cleaning apparatus) the evaporation of this by the meter itself, of course, interferes with making accurate measurements. But such fog or mist can readily be transformed into dry vapor by the introduction of heat from a small steam radiator consisting of a coil of pipe in the entrance to the

meter. Steam is usually available, and can be introduced at about atmospheric pressure or above, as may be most convenient. A very small amount of steam suffices for this purpose.

Let us now briefly consider the theory of the meter.

Let G = cubic feet of gas per hour,

E = energy in kilowatts,

then B.t.u. per hour = 3412 E.

T = temperature difference, deg. Fahr.,

S = specific heat per cubic feet.

The GST = heat energy equivalent to E, or

$\frac{GST}{E} = \frac{3412}{S}$ = constant K which depends upon the specific heat of the gas. For a gas of constant specific gravity the value $GT \div E$ is, therefore, a constant.

That this is correct is brought out by experimental tests, the results of which are given in Fig. 2. The abscissas give values of $T \div E$ or degrees of temperature rise per kilowatt of electrical input. The ordinates give values of G or cubic feet of gas per hour. Since according to the above formula $GT \div E$ should be a constant, the curves of Fig. 2 should be rectangular hyperbolas. This is indeed the fact.

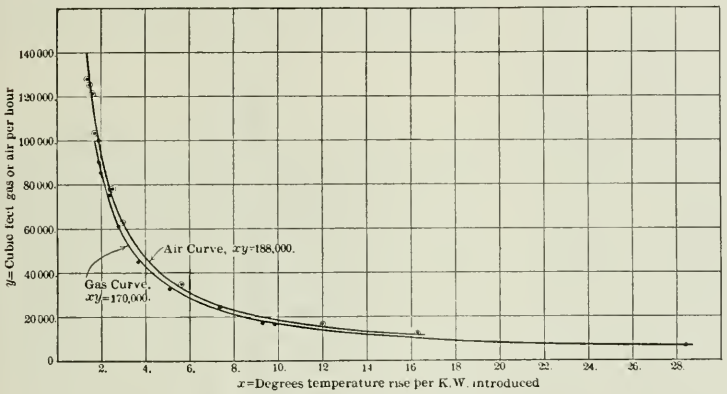


FIG. 2—RESULTS OF TESTS OF GAS METER.

Since the temperature difference T is kept constant it follows that $\frac{K}{T}$ is constant. Let $\frac{K}{T} = C$. Then $G\frac{KE}{T} = CE$.

It can now be shown that if the specific heat of gas made under given conditions be calculated from the customary chemical analysis and the specific heat of the constituents, then this specific heat may be used for determining the constant C. From the gas curve (Fig. 2), which was made with illuminating gas at an average temperature of 59° Fahr. and under an average absolute pressure of 6 in. water and 29.8 in. mercury.

$$K = 170{,}000 = \frac{3412}{S}$$

Therefore, for the condition of the gas when the tests were made the specific heat per cubic foot, $S = \frac{3412}{170{,}000} = 0.0201$. If this be reduced to standard conditions of 32° Fahr. and 29.9" mercury then $S = 0.021$, which is to be compared with the calculated specific heat, giving $S = 0.0211$.

If the standard conditions are taken as 62° Fahr. and 29.9" mercury the specific heat becomes 0.0198 and the constant becomes $K = \frac{3412}{0.0198} = 172{,}500$, nearly.

If the temperature difference is kept constant at 2° Fahr., then $\frac{K}{T} = \frac{172.500}{2} = 86{,}250 = C$, or $G = 86{,}250\,E$.

The cross-section paper on the recording watt-meter is ruled so that 86,250 E is read directly, instead of the watts E. The record is thus read directly in cubic feet of gas.

The regular records of chemical analysis of the gas should be referred to from time to time in order to ascertain what percentage variation, if any, takes place in specific heat. It appears that the elements which vary during the operation of a gas plant are not those whose variation produces serious variation in specific heat. The variation that does take place is apparently well within the limits of accuracy practicable, or generally considered necessary in the operation of gas plants. By making frequent chemical analyses the error can be reduced so as to be quite negligible.

The conditions during the air tests of Fig. 2 were as follows: Barometer, 29.75; pressure, 6.5 in. water; average temperature of air as measured in the wet meter, 60° Fahr. From the air curve obtained under these conditions (Fig. 2). $K = 188{,}000$, and $S = 3412 \div 188{,}000 = 0.0181$.

Reducing this to standard conditions of 32 and 29.9 in. mercury, $S = 0.0191$. This is to be compared with the accepted specific heat of air under these conditions, or 0.0192 b.t.u. per cubic foot.

This is probably excellent evidence of the accuracy of these tests and of the meter.

Aside from its accuracy, based on the correct application of exact specific principles, the Thomas gasometer has some obvious great practical advantages. Compared with a wet gas meter, such as is commonly used for measuring the flow of gas, the Thomas meter is very much smaller for the same capacity. Moreover, its switchboard may be placed at a distance from the pipes through which the gas flows, if this is desired, thus permitting the recording mechanism to be installed in the office, where the records made are under constant observation.

Electric Iron Smelting.

From time to time notices have appeared in our columns on the Grönwall, Lindblad and Stalhane (A. B. "Elektrometall") process of electric iron smelting, and in our last issue (page 111) we mentioned the starting of a furnace of this type at Trollhattan, Sweden, and gave some of the results obtained with it. It should now be of interest to sum up briefly the progress which has recently been made in the introduction of this process, for which the British, United States, Mexican and other rights have been acquired by Electro-Metals, Ltd., 9½ Union Court, Old Broad Street, London, E. C.

At the meeting of the Iron and Steel Institute in May, 1909, a paper describing this process was read by Mr. E. J. Ljungberg, chairman of the Stora Kopparbergs Bergslag's A. B., Sweden, at whose Domnarfvet iron works the experiments had been carried out. Later an exhaustive report on the experiments by Mr. Lars Yngström, managing director of the company, was published (our Vol. VIII, page 11, January, 1910). This report, which gave a complete description of the furnaces, showed that a technically and commercially successful process of electric iron smelting had been developed. We will in the following briefly survey the progress made in the practical application of the process.

In its home country, Sweden, the importance of the process was quickly recognized, more especially because the conditions

of the country are favorable for electric iron smelting. High-grade iron ore is abundant and the numerous waterfalls of the rivers furnish a source of electric power at low cost. In addition the country lacks coal and the iron industry therefore mainly depends on charcoal for fuel which is constantly growing more scarce and costly. The possibility of replacing about two-thirds of the quantity of fuel by electric power together with the prospect of still further improving the quality which has made Swedish iron famous has therefore made the new process a matter of national importance, and it has been taken up as such.

There exists in Sweden an institution called "Jern-Kontoret" which is an association of practically all iron and steel manufacturers in the country. This association is over 150 years old and plays a very important part in the Swidish iron industry. Its object is to suppply technical and financial assistance to the members and generally to promote the iron industry of the country.

After careful consideration of the results of the Domnarfvet experiments the board of directors of "Jern-Kontoret" determined to enter into an agreement with the patentees and make an appropriation of money for a plant to thoroughly investigate the process. The fact that this is the first occasion in its history that the association has embarked on a technical undertaking on its own account proves the importance attached to the process. The Swedish Government also offered to assist in the undertaking by supplying electric energy at a nominal cost from their large power plant at Trollhattan.

This installation has now been in operation since the middle of November. It comprises a furnace for 2,500 hp and is elaborately fitted with the best appliances for exact scientific research. It may be safely predicted that the results of work carried out under such auspices will prove of the greatest value and interest.

It is not to be expected that any official report on this work will be available for a considerable time to come, but we are in a position to state that the results so far are highly satisfactory. Some numerical results were given from absolutely reliable private correspondents on page 111 of our last issue. The experience with the experimental furnace naturally suggested various improvements which have been embodied in the new furnace and have proved of great importance for the smooth and economical working of the process.

Meanwhile private enterprise has not been idle. At the conclusion of the experiments by the inventors the experimental furnace at the Domnarfvet Iron Works was taken over by the owners, the Stora Kopparbergs Bergslag's A. B., who continued to work it for their own purposes. They are now constructing a new furnace for 4000 hp which is the first of a number of similar furnaces planned for a total of about 40,000 hp. The works contemplated will have an annual capacity of over 120,000 tons of pig iron and the cost is estimated at upward of $3,500,000.

Another well known Swedish iron works, the Uddeholm Company, have purchased a license and are about to put down a plant of three furnaces for 3000 hp each. Other Swedish works have also acquired licenses for construction and propose to erect works on similar lines.

The new conditions pertaining to electric iron smelting appear likely to foster an iron industry in some countries which have hitherto not produced iron. An example of this is Norway. The ores of this country are mainly of low grade and no coal is found there. Up to the present time no iron has, therefore, been produced there. Water power is, however, abundant and charcoal can be produced. There has now been formed a company which has acquired the Norwegian patent rights to the "Elektrometall" process and is putting down works at Tyssedal near Hardanger. These works will comprise two furnaces for 3000 hp each. Another company has been formed for putting down works at Bilefoss near Arendal, where about 30,000 hp will be utilized for iron smelting.

Similar conditions bid fair to create an iron industry in more distant countries where the cost of freight would enable a native electric iron industry to compete successfully with imported iron and steel. We are informed that Electro-Metals, Limited, are engaged in a number of important negotiations for electric iron smelting plants in several over-seas countries.

The quality of electrically produced iron has proved decidedly superior to that of ordinary pig iron from a similar grade of ore. It will readily be seen that this must be the case, as the sulphur contents can be very much reduced in the furnace and a very dense and tough material is obtained on account of the absence of gases in the iron.

Considering the developments which have already taken place it is safe to predict that this process is destined to play a very important part in the industrial life of many countries and that we may expect to see iron production established in various parts of the world where cheap electric power rather than an abundance of coal determines the prospects of economical success. Canada and California seem particularly in line for such developments.

NOTES

Society of Chemical Industry.—At the meeting of the New York Section of the Society of Chemical Industry, on Feb. 24, the following papers were read: On "A New Treatment of Water, Preventing Scale in Boilers Without the Use of Chemicals, by Aid of Aluminium Plates," by Mr. Thomas R. Duggan; and on "Boiler Water Purification," by Mr. William M. Booth.

A Congress of Technology will be held in Boston on April 10 and 11 to fittingly mark the fiftieth anniversary of the Massachusetts Institute of Technology. A large number of technology graduates, who have been conspicuously successful in varied lines of engineering, will present papers at the congress discussing the conditions and prospects in specific industries.

The American Mining Congress has recently increased its board of directors from nine to thirteen, and has adopted a plan for the organization of State chapters. The next meeting place of the congress probably will be in Chicago. Secretary Callbreath is now in the East in connection with the work of the congress, and will arrange for the 1911 meeting.

Enameled Sills.—The Stuart & Peterson Company, of Burlington, N. J., has recently completed the job of re-enameling two mammoth chemical stills of Parke, Davis & Company, each weighing 6500 lb. and having a capacity of 650 gal. These stills were originally of German make, but after several years of constant use re-enameling become necessary. To carry it out the Stuart & Peterson Company had to erect a special building and oven. The enamel used is the acid-resisting enamel of the Stuart & Peterson Company.

Refractory Cement.—The H. W. Johns-Manville Company, of New York City, have just placed on the market a line of refractory cements for furnace settings of various types, cupolas, lining brass furnaces, assayers' crucibles, oil burning, tilting and rotary furnaces and for patching and facing brick in the fire zone under various conditions. These cements are rated to resist temperatures as high as 3000° Fahr. (about 1650° C.). The same company has also developed a coating for walls known as brickline cement which prevents clinkers from adhering and to seal the pores of the brick.

Titanium Steel Rails.—The New York Central Lines have ordered 41,500 tons of titanium bessemer rails for 1911, specifying the use of 1 per cent of the 10 per cent ferrotitanium. This alloy is made, according to the patents of Rossi, by the Titanium Alloy Manufacturing Company, of Niagara Falls. This order will, therefore, require more than 400 tons of the 10-per cent alloy, which is believed to represent the largest single order ever placed for alloy steel rails.

Instruments of Precision and Laboratory Apparatus is the title of the handsome new 120-page catalog of Messrs. Wm. Gaertner & Company, of Chicago, giving illustrated descriptions of their numerous instruments of precision (microscopes, telescopes, slide rules, dividing machines) and laboratory apparatus for the demonstration of the phenomena of mechanics, accoustics, heat, light and electricity. Among other instruments Messrs. Wm. Gaertner & Company are the original manufacturers and the co-designers of the apparatus described in Millikan's Molecular Physics and Heat, in Millikan & Mills's Electricity, Sound and Lights, and in Millikan & Gale's Laboratory Course in Physics. The firm has just moved into their new fireproof office and showroom.

Recording Thermometers.—We have received from the Bristol Company, of Waterbury, Conn., their illustrated 40-page catalog of Bristol's class III recording thermometers, including a description of the new compensated gas-filled thermometers which are being used for a great variety of applications for recording temperatures up to 800° Fahr.

PERSONAL

Mr. **Sterling H. Bunnell** has resigned his position as works manager of the Griscom-Spencer Company to open an office at 90 West Street, New York, as a specialist in factory system and consulting engineer. Mr. Bunnell is the author of "Cost Accounting for Manufacturing Plants," about to be issued by D. Appleton & Company. Mr. Bunnell has been retained by Messrs. Clinton H. Scovell & Company, certified public accountants of Boston and Chicago, to take charge of the cost keeping and factory efficiency branches of their work and will also represent them in New York.

Mr. **J. M. Callow**, of the General Engineering Company, Salt Lake City, was in Denver and Leadville, Col., recently on professional business.

Mr. **John Collins Clancy** has been in Denver several times recently conferring regarding the first installation of his process at Victor, Col.

Mr. **J. V. N. Dorr**, metallurgical engineer of Denver, has returned home after spending several months in England.

Mr. **J. R. Finlay**, who has been general manager of the Goldfield Consolidated Mines Company for the past year, has resigned his position.

Mr. **H. W. Hardinge**, of New York, is on a trip through the West on professional business and for the purpose of investigating the subject of ore reduction, in which he is specially interested.

Mr. **Richard Lindenberg** and Prof. **F. R. Eichhoff** are expected to arrive in New York on March 7. Mr. Lindenberg, who has not been in this country since 1904, is the president of the Lindenberg Steel Works, of Remscheid-Hasten, Germany, which were described in detail in our October issue, 1910 (Vol. VIII, page 563), and which are particularly interesting as they were the first German works to introduce on large scale electric steel refining by the Héroult process. Professor Eichhoff was formerly the managing expert of the Héroult-Lindenberg electric furnaces and now occupies the chair of the late Professor Wedding at the Royal School of Mines in Berlin.

Mr. W. H. McFadden has tendered his resignation as vice-president and general manager of Mackintosh, Hemphill & Company, Pittsburgh, to take effect March 1, 1911, and same has been reluctantly accepted by the directors. Mr. McFadden has been closely identified with the rolling mill industry for the past 25 years and has taken an active part in the construction and installation of many of the most important steel mills in this country. Mr. McFadden will divide the next few months between looking after his important gas and oil interests and in traveling for his health.

Mr. **William McLaughlin** has been appointed mill manager of the Alvarado Mining Company.

Dr. **Robert H. Richards** recently made a trip into the zinc district of Missouri for the purpose of inspecting the work of the jigs and classifiers designed by him.

Dr. **L. D. Ricketts**, manager of the Cananea Consolidated Copper Company, Mexico, is taking a vacation in Europe.

Mr. **R. W. Robbins** has been appointed superintendent of the McKinley-Darragh property at Cobalt, Ontario.

Mr. **Dennis Sheedy**, vice-president of the International Smelting Company, recently made a trip of inspection to the company's smelter and other properties.

Mr. **L. C. Triplett** has resigned his position as mill manager of the Alvarado Mining Company, Mexico, and has accepted a similar position at Guanajuato for the Guanajuato Development Company.

OBITUARY

Edward Balbach, Jr., president of the Balbach Smelting & Refining Company, Newark, N. J., died on Dec. 30, 1910, after a short illness. Mr. Balbach was born in Carlsruhe, Germany, in 1839, and came to the United States in 1850. Together with his father, Edward Balbach, Sr., he was a pioneer in lead smelting and refining, and in copper refining by electrolysis. He first introduced improvements in lead refining by desilverizing with the addition of zinc, and recovering about 50 per cent of the zinc by distillation. For many years the Balbach works were one of the few sources of refined lead in this country. In copper metallurgy Mr. Balbach did important work in constructing and operating the first electrolytic refinery in the United States in 1881, which was the forerunner of a great industry in which the United States holds the first position. The history of this pioneer American copper refinery was given in our Vol. II, page 303. In the death of Mr. Balbach the metallurgical profession has lost an able and energetic worker who was worthy the high position accorded him by his associates.

Walter L. Pierce, who for thirty-two years had been connected with the Lidgerwood Manufacturing Company and for twenty-nine years its secretary and general manager, died of heart failure at his winter home in the Hotel St. Andrews, New York City, Dec. 10, 1910. He was a son of John F. Pierce, and was born at Dorchester, Mass., on June 8, 1855. His parents survive him, and he leaves a widow, Jane Hutchins; an only son, Walter L. S. Pierce; a brother and a sister. He was known to a wide circle of personal and business associates. He was remarkable as an organizer, and so perfect was his work that no detail of the great business which grew up under his hand was neglected during his long absences from his desk while seeking health, and the coherent body which he formed is a monument to the efficiency of his work. Besides his connection with the Lidgerwood Manufacturing Company he was treasurer of the Hayward Company and of the Gorton-Lidgerwood Company. One of Mr. Pierce's most intimate and life-long friends said of Mr. Pierce: "His unselfish life is his eulogy, and words in lending themselves to praise his steadfast soul do most honor themselves. Given to mingle in the everyday affairs of life, he dignified them in that he added to the world's store of men's confidence in each other. Tenacious in application, indomitable in courage, he was master of his calling in life, but not before he was master of himself. Experience and knowledge were not, however, for him to hoard, capital for accumulation to himself, but rather he gave of himself generously, and no worthy plea so weakly pressed that did not meet a quick response." It will not be out of place to acknowledge here the friendly sentiment which Mr. Pierce always maintained toward this journal since the early days of its foundation.

Charles F. Shelby, general metallurgical superintendent of the copper smelter of the Cerro de Pasco Mining Company in Peru, was killed in an automobile accident between Lima and Ancon, Peru, on Jan. 25, 1911.

Mr. Shelby was among the prominent and able metallurgists in the copper industry. He was born at Omaha, Neb., in 1874. His first professional employment was as assistant chemist at the Globe smelter, Denver. He left this plant in 1900, going to Aguascalientes, Mexico, where he was chemist and later assistant superintendent. In 1903 he entered the field of copper metallurgy under the able direction of Dr. L. D. Ricketts, at Globe, Ariz., where he acted as superintendent of the Old Dominion Copper Company's reduction works. In 1906 he accepted a more responsible position at Cananea, where he inaugurated many improvements in copper metallurgy and aided in bringing the works of the Cananea Consolidated Copper Company to a high state of efficiency. Early last year he was selected as superintendent of the Cerro de Pasco Mining Company, and in company with Mr. A. B. W. Hodges as general manager, went to Peru.

Mr. Shelby was a constant contributor of valuable notes and articles to the technical press, following in this respect the liberal policy of his employers in giving the profession the benefit of his ingenuity and ability.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore-Treatment.

61,866, Feb. 5, 1867, Julio H. Rae, M. D., of Syracuse, N. Y.

Gold and silver. The crushed ore is placed in the bottom of a glass jar upon a platinum anode plate and around a revoluble platinum-wire cage anode. The dissolved metals are reprecipitated upon a sheet of copper in the solution above the ore.

79,744, July 7, 1868, A. L. Fleury, of New York, N. Y.

Precious metals. Silicious ores, which may be mixed with chemicals, are heated and subjected to chemical gases in a shaft furnace. The treated mass is removed into a closed heated vessel, wherein it is subjected to the action of water, steam and electricity.

90,565, May 25, 1869, William John Lynd, of Golden City, Col.

Removes iron, copper and other discoloring matters from clay by passing an electric current through water containing the clay in suspension.

101,009, March 22, 1870, Nathaniel Haskell, of San Francisco, Cal. Assignor to himself and Joseph F. Steen, of same place.

Copper. The ore is placed in heaps on a hillside where the ground is not porous. A trench is dug around the ore heaps leading to a vessel or vessels below. The ore is alternately drenched with water and exposed to the air for several days. The resulting weak solution of copper sulphate is electrolyzed in the vessels, the copper being preferably electroplated upon metal articles.

239,300, March 22, 1881, Andrew Ryder, Oakland, Cal.

Ore is disintegrated by heating it and dumping it into a liquid carrying an electric current. The apparatus illustrated is a tank containing opposed downwardly converging copper-plate electrodes, the heated ore being dumped between them.

246,201, Aug. 23, 1881, Emile Reynier, of Paris, France. Assignor to Simon Philippart, the son, manager of La Force et la Lumière Société Generale D'Electricite, of Brussels, Belgium.

The ores are used as soluble electrodes in a two-fluid voltaic cell. The anode and cathode compartments are separated by a diaphragm of paper, cloth, skin, membrane or felt. One compartment contains an anode of zinc or lead, or fragments of galena, calamine or other rich conducting ores, a lead or zinc plate conductor being placed on the ores. The anode solution is caustic soda or potash. The other compartment contains a copper solution, e.g., sulphate. The zinc or lead dissolves and is recovered with regeneration of the caustic alkali. The other metals, e.g., silver, remain as residues. Copper is simultaneously deposited in the other compartment. The cell generates an electric current which is externally utilized. The process is applicable to ores containing also nickel, cobalt, mercury and gold.

272,391, Feb. 13, 1883, Antonin Thiollier, of Villeboeuf de Roche la Moliere. Assignor of one-half to Albert Ancel, of Chateau de Bouches D'Uzure, Canton de Craon, Mayenne, France.

The crushed ores are placed around carbon electrodes in a series of parallel vertical receptacles having perforated walls. The electrodes of opposite polarity are placed between the receptacles with their lower edges in gutters to catch deposits. Beneath the receptacles and electrodes is an open space containing steam-heated coils. The receptacles may be of wood, wickerwork, gutta percha, rubber or coated sheet iron. If the ore is very fine they are lined with cloth or felt. If the ore is a poor conductor coke or plumbago is mixed with it, being separated from the residue and re-used. The cells may have a tight cover to retain fumes. When the electrodes in the ore are anodes, the collecting electrodes between the receptacles may be plates of copper. When the electrodes in the ore are cathodes the intermediate electrodes are insoluble, e.g., carbon. For ores of gold, silver and platinum the embedded electrodes are made anodes. For silver ores a solution of sodium sulphate or nitrate, or of hot sodium chloride may be used, or a cyanide solution. For gold or platinum sulphates nitrates, cyanides, sodium chloride or caustic soda or potash are available. Where the ore contains two or more of these ores sodium chloride is preferably used. If the ore contains silver chloride an ammonia, ammonium chloride, hot sodium chloride or cyanide solution is used. For silver-sulphide ores an ammonia or hot sodium chloride solution is used. For gold telluride ores a sodium chloride or cyanide solution, the ore being placed around the anode, or a sodium chloride solution, the ore being placed around the cathode, in which case the gold deposits at the anode as auric acid. Ores containing gold, silver, lead, copper, iron and metalloids are treated with a hot solution of sodium chloride. The gold, silver, copper and lead deposit at the cathode. Iron remains in the vessel as an oxide. Zinc ores are placed around the anode, ammonia, ammonium carbonate, soda and potash solutions, etc., being used. Ores of lead may be treated with a hot solution of sodium or lead chloride, or with a caustic soda solution. Silver may be parted from lead and zinc. For antimony ores a hot potash solution is used; for mercury ores a solution of the sulpbate, nitrate or bichloride of mercury. Copper ores are variously treated, refractory ores being roasted, and ammonia, ammonium carbonate or potash solutions used. Nickel ores are roasted and ammonia or ammonium carbonate solutions used. Ore containing carbonate of iron and calcium may be electrolyzed in a solution of sodium or iron chloride, chemically pure iron being deposited and lime remaining as a residue.

BOOK REVIEW.

Welding.—By **Richard N. Hart**, B.S. 8vo., 181 pages, 93 illustrations. Price $2.50 net. New York: McGraw-Hill Book Company.

The four principal chapters are devoted to ordinary welding, electric welding, hot-flame welding, and thermit welding. The descriptions are good, illustrations and photographs first class, and the whole constitutes a useful and instructive book. In his preliminary remarks the author is sometimes hazy in his statements, and his English is at places faulty, but the practical part is well written, clear, and direct. It is a good practical book.

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History in the Making.

When in the course of human events an ideal is made a reality, history is being made. The opening of the Chemists' Building in New York is a milestone in the history of chemistry in America. In its stately simplicity and dignity it is a monument to the idealism of its builders. In its equipment and facilities it is a great endowment for future service. This ideal was realized without outside help through the combined efforts of American chemists. A few years ago the question might have been raised whether American chemists could do it. They could, they would, they did. And future historians will speak of the opening of the Chemists' Building as the beginning of a new epoch in the history of chemistry in America.

This was the first step—and it is always the first step that counts. Next year an International Congress of Applied Chemistry will meet for the first time on American soil. The American organization has been perfected and tentative rules have been formulated, which will be found on page 173 of this issue. We gladly place our correspondence columns at the disposal of our readers for critical discussion. For all agree that next year's Congress must be made a grand success, worthy of America and American chemistry.

Smelting Fine Ore.

One of the great sources of loss in the smelting of base metals, particularly copper, is the escape of fine ore in the reduction process. It is frequently the case that the fine ore is of high grade and then the loss becomes more serious and the ingenuity of the metallurgist is taxed to provide a suitable remedy. Naturally, in all preliminary treatment, such as concentration, every effort is made to produce as coarse a product as possible and to avoid the production of fine concentrate, but when the mineral is finely disseminated through a large amount of gangue and occurs in small crystal masses, fine grinding and concentrating become a necessity. Then arises the problem of the best disposition of the fine ore.

Various solutions have been proposed, and some adopted. In one instance the fine concentrate is mixed with concentrator slime and furnace flue dust and briquetted. The briquettes are sufficiently strong to stand the subsequent handling and are used as part of the blast-furnace charge. This scheme offers quite a satisfactory method of disposing of these three troublesome products, although the quantity of slime which can be used is not commensurate with that produced.

Another suggestion that has been made and practiced to a limited extent is to mingle the fine ore with low-grade matte or rich copper slag, in such quantity as will be absorbed, with or without solidification of the absorbing medium. This process gives a semi-fused mass suitable for smelting if slag

has been used, or for converting in the case of matte. In either event the fine ore shortly becomes part of a molten bath with no chance for escape as dust.

The most recent and novel of the means proposed is to be used at Cananea for avoiding loss in the treatment of Miami copper concentrate. This material is very fine and high grade. Ordinary methods of smelting would result in high losses and hence some scheme had to be adopted to increase metallurgical recovery. It is proposed to blow the fine concentrate trough the tuyères directly into converters charged with matte and thus cause it to become a part of the molten bath. This will eliminate dust loss and will lower the smelting cost materially.

Expenditures at the Gary Steel Plant.

Four months ago we made some editorial remarks upon "the iron industry's insatiate thirst for capital," and the annual report of the United States Steel Corporation gives figures of its expenditures at its new plant at Gary, Ind., which furnish a concrete illustration of the principle adverted to. The expenditures at Gary would make the iron and steel engineer of half a generation ago stand aghast. To Jan. 1, 1911, they amounted to $69,978,695, of which $16,072,851 was spent in 1910. For work authorized at Gary, but still to be completed, there was the additional sum of $15,500,000, making in round figures a total of $85,000,000 expended, or to be expended, at Gary in connection with the present plans. This expenditure covers the land for both the plant and the town (less credits for lands, lots and houses sold), buildings and other town improvements, an enormous amount of terminal railroad work, as well as the moving of two trunk-line railways, lake harbor and docks, by-product coke ovens, 12 blast furnaces, 42 open-hearth steel furnaces and various rolling and finishing equipment.

There has been appropriated from surplus net income a total of $65,000,000 for Gary, of which $10,000,000 was appropriated in 1910; in addition to this construction work aggregating $9,775,506 was financed by three subsidiary companies, the American Sheet & Tin Plate Company, the American Bridge Company and the Chicago, Lake Shore & Eastern Railway Company, which have been doing construction work at Gary. Thus there is left, of the $85,000,000 expenditure authorized, a trifle over $10,000,000 to be provided, either by future appropriations from surplus earnings, or by subsidiary company financing.

The production of the Gary plant in 1910 was as follows, in gross tons:

Pig iron	729,072
Steel ingots	1,006,252
Rails	435,142
Shapes, bars, etc.	340,993
Total rolled steel	776,135

The blast furnaces are rated nominally at 150,000 tons annually, or 1,200,000 tons for the eight, and the finished rolled steel, when the works are running full, may be estimated at approximately the same total. Thus we see that for 1,200,000 tons of finished steel approximately half of it rails, there is involved an expenditure of $85,000,000, or $72 per ton of annual output, and this does not include coal or ore mines or transportation facilities other than docks and terminal railroad work. Of course, a portion of this may be written off, because town lots or buildings will be sold, or the buildings will furnish revenue, but as the total expenditures in the town, less credits for lands, lots and houses sold, have amounted only to $10,509,325, the expenditures for the plant proper cannot be set at less than $75,000,000, which means $62.50 per ton of annual product.

This allotment per ton of output is very large compared with the standards of half a generation ago, when expenditures ran from $10 to $20 per ton, but it is small compared with the profits which ought to be realized, for 10 per cent to cover interest and depression would involve only $6.25 per ton. Depreciation in the steel industry is written chiefly by what occurs outside the plant, not by what occurs inside; in other words, by the progress of the industry. As the Gary plant is intended to be far ahead of anything yet attempted its depreciation may rightly be estimated as low for the next few years.

The Electrode Problem.

In any electrochemical process, be it electrolytic, electrothermic or a combination of the two, the cost of power is often a very large single item in the cost sheet. Among the other items, such as labor, repairs and renewals, interest and depreciation and general expenses, the second named includes a figure often large and usually uncertain, viz.: The cost of replacement of electrodes. The cost of power, on the other hand, is always large but rarely uncertain. Now in any new business undertaking uncertainty is usually the greatest deterrent. Indeed, the largest fortunes of this country have been gathered for the most part by men whose boldness is founded on the ability to see further and clearer through the haze than can the ordinary man. Darkness generates fear, but the light of day generates courage.

Accordingly, any light that can be shed on the commercial engineering of electrodes is of advantage to the wide and growing circle of metallurgists who are concerned with electrolytic and electric-furnace processes. The former are as a class older than the latter and hence have the advantage of the latter in the question of electrodes. For, in a properly developed electrolytic process, the cost of electrodes is not startling, varying, as it does, from 3 per cent to 10 per cent of the total cost. In the newer and wider field of the commercial electric furnace, especially where the electric furnace is operating on smelting processes or on refining, the cost of electrodes is sometimes as high as 25 per cent of the total cost of treatment of the successful process. And in the case of the unsuccessful process the cost of electrode renewals is so unexpectedly great as to cause often the abandonment of attempts that seemed at the first very hopeful.

Undoubtedly, great progress has been made in the past ten years in the manufacture of electrodes and also in the design of electrodes for their peculiar functions. We are confident that the next five years will witness even greater progress. The process of manufacturing electrodes is everywhere essentially the same. Briefly described, some form of carbonaceous material such as crushed "oil-coke" is mixed with tar and the mixture molded hot to the required form; the molded article

is then subjected to slowly increasing temperature until the tar is either volatilized or carbonized; thereby the solid particles of the original carbon are cemented together in a more or less compact mass; finally, the baked electrodes are annealed by slow cooling.

This process seems quite simple. In point of fact it is both simple and elementary, for a very satisfactory home-made electrode can be made for an intermittent type of electric furnace "in place," by tamping a hot mixture of tar, broken electrodes, and crushed coke in a form, "larding" it with a modern woman's hatpin, driving off the tar slowly with a wood fire and completing the cementing carbonization by auto-generated electric heat. But the process of making regular molded electrodes for a continuous electric furnace is not so simple as it looks at first glance.

In the first place, the chemical and physical properties of both the carbon and the binder are never uniform. Every chemist knows what a complex stuff tar is and also the varying idiosyncrasies of the carbon molecule. Next, to make a compact mass, the structure must have a certain density and at the same time a certain porosity. It seems like a paradox to combine these two diametrically opposed qualities in one article. But, nevertheless, it must be done. To insure firmness, the particles of carbon must be bonded well together, for otherwise the electrode obviously will be mechanically weak. If, on the other hand, the structure be very dense, the evolution of hydrocarbon gases in the baking will cause the development of many small places of intense local pressure with ensuing "checking" and resultant formation of planes of weakness in the finished article. In the third place, the baking must be done exceedingly slowly to reduce the internal pressure of hydrocarbon gases generated to such an extent that they can percolate through the electrode and get out without damaging the electrode. We see plainly that the whole operation is one that needs as much care and conscience as nursing a baby.

Now as the modern tendency is to build electric furnaces of great power requirements, the diameter of the electrodes asked for is, of course, greatly increased. It is apparent that with conditions such as they are the difficulties of manufacture are tremendously increased as the size of the electrode is increased. Carbons for arc lamps are not hard to make. In this case it is largely a question of cheapness gained by a large output. But to produce an electrode 24 in. in diameter and 120 in. in length is a triumph of mechanical and chemical engineering.

As is well known, we use in this country both graphitized and amorphous carbon electrodes. Each has its peculiar advantages and each has its peculiar disadvantages. Both are fairly resistant to the slightly oxidizing influences of the nearly neutral atmosphere of the average electric furnace. The graphitized article costs considerably more than the amorphous carbon electrode but it is a much better conductor of electricity. On the other hand, it is a better conductor of heat. The significance of these two facts has been discussed at such length in this journal a year ago in connection with the laws of electrode losses that there is no necessity to discuss this matter again at the present time. The graphitized electrode can be machined and so a composite electrode can be made, as is done at South Chicago and at Worcester. This is practically denied its rival. Both can be fashioned so that a new electrode can be "butt-connected" to the stub end of an old electrode and so the electrode is for all practical purposes fed, as needed, continuously into the furnace. The greatest point of superiority of the graphite electrode is usually unnoticed. This is its great power of resistance to sudden temperature changes. In the electric furnace industry, especially as developed in the United States, the workmen have never the skill in the art at first that is needed and developed later. The natural result is uneven working and its concomitant, bad temperature control. This affects the amorphous carbon electrode severely and the result is a high proportion of electrode scrap, even when the carbon electrodes have been made under good conditions.

It would appear that the process of baking the electrodes in electric furnaces of the Acheson type is inherently superior to the operation as conducted in the open-hearth furnace with its removable top and the covering of sand over the molded electrodes. The flow of heat from the innumerable points of local resistance is much more uniform in this style of electric furnace than in the gas-fired furnace, where the heat flows down through the sand covering to the entire mass of electrodes. As in a roasting furnace, the question of surface is of most importance. In the Acheson electric furnace the heat is generated in proximity to the place it is needed.

To cheapen the cost of production, it might be well to consider the manufacture of electrodes built up of standard elements of say 3 in. x 8 in. x 72 in. These elements could be produced far cheaper than the larger dimensioned electrodes. Practical experience seems to show that the built-up electrode fills the bill. The elements could be produced cheaply because the heat would only have to penetrate 1½ in. and the hydrocarbons would have just this short distance to percolate outward.

It may also be well to consider a very slow and careful annealing when once the maximum temperature has been reached. A slow annealing greatly toughens the carbon and a slow adjustment of the internal strains makes for the desired quality of resistance to sudden temperature changes—a condition of the American climate that even affects the atmosphere of the American electric furnace.

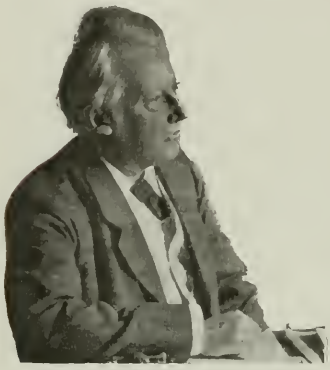
Undoubtedly some of the large concerns using electrodes in quantity will in the future make their electrodes in their own plants. Ground "butts" and ground scrap electrodes are suitable ingredients of the mix and cheapen the cost of manufacture. A very profitable field of study is the application of the microscope to the microstructure of electrodes. The methods of attack in research which have developed the special sciences, known as metallography and petrography, could be well used to garner some solid scientific facts about electrode manufacture.

Indeed, the broad field of electric furnaces, their necessaries and accessories, such as voltage-regulators, pyrometers, refractories, and electrodes, is at the present outlook an ever-expanding one and in it progress is demanded. As the use of the electric furnace increases "the essential details" become better developed and easier of application. The smaller and apparently insignificant points of electrode manufacture which have been pointed out above, when incorporated in the "skill of the art," will render much easier the fruition of many electric-furnace processes.

The Death of Professor van't Hoff.

We regret to report the death of Professor J. H. van't Hoff on March 1, at Steglitz, near Berlin, Germany, at the age of 59 years. Professor van't Hoff was taken ill during the autumn of 1906 with pleurisy, which turned into tuberculosis, and he had not been well since. During the summer of 1907 he was critically ill for three months, but partially recovered, never, however, throwing off the insidious disease.

Jacobus Henricus van't Hoff was born in 1852 at Rotterdam, Holland, as the son of a physician, and was educated in the public school of that city. In 1869 he entered the Polytechnicum (Institute of Technology) at Delft and in 1871 the University of Leyden; subsequently he studied at the Universities of Bonn, under Kekule; of Paris, under Wurtz, and of Utrecht, where he received the degree of doctor of philosophy in 1874 and where he began teaching in 1876. He was called to Amsterdam in 1877, where he was soon made a professor. In 1894 he was called as professor of physical chemistry to the University of Berlin.



J. H. van't Hoff

Professor van't Hoff's name will be connected forever with the history of modern physical chemistry, which owes to him directly some of the most fruitful and important broad generalizations and indirectly the impetus to an enormous amount of experimental work. "Wenn die Könige baun, haben die Kärrner zu thun."

In 1874, while still a student at Utrecht, he published his paper on the necessity of representing structural chemical formulas in a space of three dimensions, instead of a plane of two dimensions, and on the application of this new stereochemical method to the explanation of optical isomerism.

Van't Hoff's most important contributions to science, however, were made in two publications, issued in Amsterdam in 1884 and 1885 under the title "Etudes de Dynamique Chimique" and "Lois de l'Equilibre Chimique dens l'Etat Dilué, Gaseux ou Dissous." In these two papers those broad generalizations are contained which were so effective in determining the future course of the development of physical chemistry. First, van't Hoff's "principe de l'équilibre mobile" which replaced Berthelot's "principe du travail maximum." According to van't Hoff, when two chemical systems (represented by the two sides of a chemical equilibrium equation) are in equilibrium at a certain temperature, then a rise of temperature causes a shifting of the equilibrium toward that system (or toward that side of the equation of equilibrium) the formation of which is combined with absorption of heat.

Further, van't Hoff showed here how we have in the electrical energy of a galvanic cell a measure of the chemical affinity of the reactions taking place in the cell.

But perhaps most important of all was the generalization by which van't Hoff developed the parallelism between dilute solutions and gases. The laws for the osmotic pressure in dilute solutions and for the gas pressure in gases are identical. How the apparent discrepancies of this new rule from the facts in the case of electrolytes led directly to the electrolytic dissociation theory is too well known to be further recorded here.

Van't Hoff later took up the study of solid solutions, and the depth of his scientific imagination is indicated by his applications of physical chemistry to geology. In this connection his patient work on double salts concerning the Strassfurt salt beds must be mentioned, to which he devoted the last ten years of his life.

Program of New York Meeting of American Electrochemical Society.

The first convention of a national society to be held in the new Chemists' Building at 50-54 East 41st street, New York, is that of the American Electrochemical Society, the dates being April 6 to 8 (Thursday to Saturday).

There will be professional sessions for the reading and discussion of papers on the three mornings. All will be held in the Chemists' Club. The Friday and Saturday sessions will start at 9:30, the Thursday session at 10 o'clock, when the annual business meeting will be held. At a special evening session on Friday the retiring president, Professor W. H. Walker, will present his address.

For the afternoons the following **excursions** have been arranged: On *Thursday* afternoon the Laurel Hill works of the Nicholas Copper Company in Brooklyn will be visited, which is the largest copper refinery in the world.

On *Friday* afternoon the following optional visits have been arranged: (1) The new and splendidly equipped plant of the Hanson & Van Winkle Company in Newark, manufacturers of electroplating material and machinery, with a thoroughly modern demonstrating electroplating plant; or (2) electric furnace laboratory of Dr. F. J. Maywald and the photoengraving and electrotyping shops of the Moss Engraving Company in New York City; or (3) New York Electrical Testing Laboratories, the Chemical Laboratory of the College of New York and the Laboratories of Columbia University.

On *Saturday* afternoon an excursion will be made to the Perth Amboy plant of the American Smelting & Refining Company. It comprises ore smelters, lead refining plant, electrolytic copper refining, electrolytic silver refining and the bessemerizing of leady copper matter in basic-lined converters. This immense plant refines, per month, 7,500 tons of copper, 8,000 tons of lead, 3,000,000 ounces of silver and 20,000 ounces of gold. In doing this it smelts 15,000 tons of ore and bessemerizes 1500 tons of leady copper matter. Attention is particularly directed to the basic-lined converters, the latest improvements in the metallurgy of copper.

Social Functions.—On Thursday evening a subscription dinner will be held at the Chemists' Club. On the evening of Saturday, April 8, the New York section of the American Electrochemical Society tenders an invitation smoker to the visiting members and guests of the society and to the Chemists' Club.

The program of **papers** is as follows:

Thursday morning after the annual business session, which begins at 10 o'clock.

Report of the Committee on Standard Tests for Dry Cells: C. F. Burgess.

The Testing of Dry Cells: W. B. Pretz.

Electrically Fused Alumina as a Refractory: L. E. Saunders.

A New Electric Resistance Furnace: F. A. J. Fitzgerald.

A New Laboratory Electric Furnace: M. M. Kohn.

An Electric Furnace for Heating Bars and Billets: Thaddeus F. Baily.

The Johnson Electric Zinc Furnace: J. W. Richards.

The Condensation of Zinc from Electric Furnaces: F. T. Snyder.

Reliability of Electric Furnaces for Commercial Work: F. T. Snyder.

Friday morning, 9:30.

Electric Cyanamide Process for Gold Ores: J. F. Clancy.

The Electrolytic System of Amalgamating Gold Ores: E. E. Carey.

The Electrolytic Corrosion of Electroplated Objects: W. S. Landis.

Electrolytic Refining as a Step in the Production of Steel: C. F. Burgess.

Some Problems of the Electroplater: G. B. Hogaboom.

The Haas-Oettel Electrolytic Cell: E. E. Niswonger.

Electrical Heating for Preventing Superheating of Liquids: J. Howard Mathews.

A Mercurous Perchlorate Coulometer—Preliminary Experiments: F. C. Mathers.

Some Observations on the Laboratory Production of Aluminum: H. K. Richardson.

Exhibition of the Schoenherr Arc for Fixatives of Atmospheric Nitrogen: J. W. Turrentine.

Electric Discharges in Gases: L. A. Parsons.

The Mechanism of Electrolytic Transfer: J. W. Richards.

Polarization in Its Relation to the Decomposition Pressure of Electrolytes: O. P. Watts.

Friday evening, 8:15. Joint meeting with the New York section of the American Chemical Society.

Presidential Address: Dr. W. H. Walker.

Lecture by Dr. Harrison E. Patten on "The Relation of Surface Tension and Electrochemical Action."

Saturday morning, 9:30.

Electric Furnaces for the Manufacture of Steel: James Lyman.

Some Experiences with the Electric Furnace at South Chicago: C. G. Osborne.

Electric Steel Processes as Competitors of the Bessemer and Open Hearth: A. E. Greene.

Design of a Fifty-Ton Induction Electric Steel Furnace: A. Hiorth.

A New Type of Electric Furnace: Carl Hering.

Electric Furnace Treatment of Nickel Ore: W. L. Morrison.

An Electrolytic Furnace Method for Producing the Rare Metals: J. W. Beckman.

The Advantages of Keokuk, Iowa, for Electrochemical Industries: F. P. H. Knight.

The Advantages of Maine for Electrochemical Industries: C. Vey Holman.

The Western Metallurgical Field.

Magnetic Roasting.

The treatment of those Western ores which contain a mixture of iron and zinc sulphides, pyrite and blende is in a large measure dependent on magnetic roasting and separation. The magnetic roast is a delicate operation and the devices used for producing it are not always sufficiently under the control of the operator to turn out a uniform product. Overroasting produces oxide of iron, which frequently is seen in roasted ore as a reddish substance, and which reduces the efficiency of the subsequent magnetic separation.

The magnetic roast usually is produced in furnaces not unlike those used for desulphurizing roasting. They have certain inherent drawbacks which, in a measure, make them unfit, or at least not the best, for magnetic work. As a rule the ore is exposed to heat for too long a time, and all of the particles are not evenly exposed, so that a variable product results. By rabbling a deep bed of fine ore on a hearth one cannot be certain that all particles receive the same degree of heat for the same time, although the purpose of the rabbling is to accomplish this result as well as to advance the ore through the furnace.

A new idea in magnetic roasting has been applied by Mr. A. R. Wilfley, known for his invention of the concentrating table of that name. Mr. Wilfley has tried to evolve a furnace that would give to all ore particles as nearly the same treatment as it is possible to do in a commercial roasting furnace. The furnace consists of a circular, rotating roasting hearth, with an auxiliary fire box, feeding device and means for removing the ore from the hearth when roasted. By an adaptation of the roller feed, such as is used on some magnetic and electrostatic separators, the ore is fed onto the hearth in a very shallow bed, approximately one grain deep. The hearth rotates in the opposite direction to the path of the hot-furnace gases, so that the ore gradually advances through an atmosphere of increasing temperature until it is finally removed from the hearth after one revolution. The points of feed and removal of the ore are in close proximity between the inlet and outlet of the furnace gases. The device for removing the ore from the hearth is a specially constructed suction nozzle, which is connected with a suitable receptacle.

A number of advantages will be apparent in this construction. There will be little tendency for the ore particles to become agglomerated; one particle will have the same treatment as another; the rate of speed of the hearth can be regulated to give exactly the right time of exposure according to the temperature and the roast is immediately arrested when the ore is cooled in the suction nozzle. Naturally the time of exposure of the ore to heat is very short, one to three minutes sufficing to produce the same result that is accomplished in an ordinary roaster in the same number of hours. The capacity of an experimental machine has shown that 7.5 sq. ft. of hearth area per minute will give a magnetic roast to one ton of ore in 24 hours.

Legislation.

This is the biennial period when Western Legislatures consider, among other things, ways and means to control the metallurgical and mining industries in the interest of the producer of ore, as well as methods for obtaining more State revenue from both industries. Two Colorado bills show the trend of legislation this year. Their subject matter is not exactly new and it is doubtful if either of them will pass, but they are interesting as reflecting the attitude of legislators.

The Joyce bill for regulating metallurgical plants seeks to prevent the imposition of unjust or unreasonable rates on ore treatment, the giving or receiving of rebates, or any other form of discrimination, and empowers the State commissioner of mines to enforce the provisions of the bill. Some years ago when the smelting of ores was the foremost metallurgical industry in Colorado there was considerable complaint against the smelter monopoly on account of treatment rates. It was alleged that large shippers were favored at the expense of the small, and that no uniform tariff existed for all shippers of the same kind of ore. The difficulty then, as now, was to make workable provisions for enforcing regulation of the industry.

The Joyce bill seeks to empower the State commissioner of mines to enforce the provisions of the bill, but such a thing would be almost impossible, even if conditions urgently demanded rate regulation. Any corporation strong enough to control the metallurgical situation would, under our present system of politics, first see that an acceptable commissioner of mines was appointed, after which the provisions of the law would be of little effect.

The McPolin bill for the regulation of the practice of assaying, and requiring the registration and examination of all assayers in the State, embodies another idea that frequently has been put forward. It undoubtedly springs from the abuses that have been perpetrated in the name of assaying by men who have taken advantage of a respectable calling to engage in criminal practices. Every rich mining camp in the West has had to fight the "high graders" who run alleged assay offices and buy small lots of stolen high-grade ore. In the absence of adequate means of coping with them the leading companies frequently have taken matters in their own hands

and have forcibly stopped the practice of the high-graders.

The proposed bill for the licensing of assayers is apparently another step in the elimination of the high grader as well as the incompetent workman. As with most bills of good intent the carrying out of its provisions might prove difficult. The greatest care would have to be exercised in the selection of the proposed board of examiners, for they have the power to accept or reject applicants, to supervise their work in practice and revoke licenses in case of unprofessional conduct or incompetency.

The bill would have several beneficial effects if passed and properly enforced. The public would be protected from unscrupulous assayers who sometimes falsify reports for financial considerations. Legitimate assayers would be accepted by the public on the face of their standing with the board of examiners.

No hardship would thereby be worked, and as all assayers in metal mining districts are engaged in the same line of work there would not be the objection to licensing them which is quoted against the licensing of civil engineers, for example, in which profession all members are not engaged in the same class of work and ought not to be required to show proficiency in all branches of the profession.

Utah.

A sequel to the smoke suits brought by the farmers of the Salt Lake Valley against the local smelters several years ago is the recent decision of the Bingham Mines Company, owner of the old Bingham Consolidated Smelter, to raze that plant and dispose of the equipment. This smelter was put in operation in 1901 and operated successfully until the close of 1907, when it was closed by order of the court on account of the damages caused by its fumes to surrounding agricultural land. The plant was well equipped, having five blast furnaces, one reverberatory, four McDougall roasters and the complementary converting department. When the plant was closed there were plans on foot to enlarge and improve it, but these were abandoned and the closing of the plant followed.

A recent transaction of importance was the leasing of the Ontario property at Park City, Utah, to Salt Lake men. For two years past Mr. Henry M. Crowther has represented a company which has been experimenting on the old stope fillings of the Ontario, ascertaining their value and amenability to modern methods of metallurgy. Originally the Ontario produced very rich ore which was treated by the Russell process. The average grade was over $100 per ton, and the low-grade ore was used as filling.

It is said that there are about 1,000,000 tons of this material, and it is the intention of the leasing company to remove and treat it.

The experimental work has shown that the methods of fine grinding, agitating in cyanide solution, filter pressing and precipitating by zinc dust developed in recent years will permit an extraction of over 95 per cent of the gold and about 80 per cent of the silver. With such an installation in a remodeled mill it is expected that 300 tons of ore will be treated daily. The improvements will cost in the neighborhood of $200,000.

The New Copper Mills.

The first unit of the large Ray Consolidated mill in Arizona has been put in operation and from this time on the company will be classed among the producers. The Miami also is expected to be in operation by the time this report reaches our readers.

The Chino mill in New Mexico is not expected to be in operation until July, but when it commences it will have 30 years' ore supply at the present rated capacity of the mill. The latest reports from Chino state that 45,000,000 tons of copper ore have been developed, and that 70 per cent of this will be mined by steam shovel.

The Non-Ferrous Metal Market.

The general improvement in the spelter market and the decline in prices for spot tin have been the principal features of the metal market since our last report. In copper and lead the volume of business is fair, but only for immediate needs of manufacturers.

Copper.—The March market has been without special feature, both domestic and foreign demands being light. It is generally understood that the largest transactions were at concessions in price, as the sellers seemed eager to get business. Lake copper is quoted at 12⅜ to 12⅝ cents and electrolytic at 12.15 to 12.20.

Lead.—No improvement has been noted in this market, despite the expectation that the usual demand would occur with the coming of spring. Considerable Missouri lead has been sold and the St. Louis market has dropped to 4.22½ to 4.25 cents. The New York market quotes 4.37½ to 4.40 cents.

Spelter.—Improvement in the galvanizing trade has stimulated the spelter market somewhat, with the result that prices are a little higher than last month. St. Louis spelter is quoted at 5.45 to 5.50 and the New York market at 5.60 to 5.65 cents.

Tin.—The quotations on this metal have been erratic during March. In the beginning of the month the metal was expected to show advances owing to the operations of the syndicate which was supposed to have taken advantage of the weakness developed in February. Foreign markets showed a steady decline, however, and the domestic corner was broken. Future tin has held its own and spot has suffered the decline. March tin was quoted at about 40⅜ cents.

The Iron and Steel Market.

March showed a definite decline in the rate of buying of finished steel products, which had increased so rapidly late in January and early in February. The decrease is not traceable altogether to a decrease in general business activity, although it does appear that general business is hardly as active as it was early in the year. Nor is it traceable to the famous rate decisions of the Interstate Commerce Commission, made Feb. 22. It appears rather that during December and the fore part of January a considerable tonnage of requirements backed up, partly because buyers at that time were fearful of a decline in the market, and partly because they wished to avoid incurring obligations or increasing stocks until after the inventory date. The release of this business, at a time when requirements were increasing, effected a rapid increase in the rate of booking orders. To use figures for conciseness rather than precise accuracy, bookings of orders for finished steel products averaged 35 to 40 per cent of capacity in December and the early part of January, 75 to 85 per cent late in January and in February and 55 to 65 per cent during March.

The iron and steel trade regarded rather seriously the claims made by the railroads during the pendency of the rate cases before the Interstate Commerce Commission that an adverse decision would greatly curtail their buying power, and was disappointed at the decisions rendered. However, no direct results have been traced. No orders for railroad material have been canceled, or enquiries withdrawn, while along certain lines there seems to have been a slight increase in the volume of railroad buying. In the last analysis, there is good reason to expect the throttling of the rate advancing tendency to prove favorable to the iron industry. The railroads had reached a point where large extensions were very improbable. Their existing equipment seems to be fairly adequate to the conduct of the business offered, and large consumption of iron and steel for extensions to facilities was not to be expected. On the other hand, the railroads have reached a point where they are wearing out material very rapidly, their ton-mileage of freight having doubled every 10 or 12 years. The replacement of worn-

out material, whether in brake shoes, entire cars, or rails, does not involve fresh borrowing of capital, being an ordinary item of expense, and whatever tends to increase the volume of traffic, as the curbing of the rate advancing tendency does, makes for more wearing out of material and more demand along the only line in which any large demand was to have been expected.

While bookings in the finished steel trade decreased somewhat in March, production increased. It required time for the mills to limber up and the heavy bookings in February resulted in heavy operations in March. The mills ran at about 50 per cent of capacity in December and January, about 60 per cent in February and about 75 per cent in March. It is evident that they will not be able to maintain the present pace of production unless there is an increase in buying. This is to be expected as spring weather comes.

There is no disposition to buy ahead, and current orders are for current requirements. By far the bulk of the bookings in March were in the form of specifications against old contracts, there being relatively little strictly new buying.

Prices of finished steel products are being held by the mills all along the line, there being apparently a firm conviction that at least as large a tonnage of business will result from present prices as would be brought out by lower prices, and there is evidently also a firm conviction that any departures from existing prices by individual producers would quickly inaugurate a general reduction, in which the original cutters would suffer as much as their neighbors.

Effective at the opening of business Monday, March 6, wire products were advanced $1 per ton, this following an advance of a dollar a ton made effective Jan. 23. The mills are adhering strictly to the advanced prices, but the same cannot be said of jobbers, who are cutting the market more or less generally by a dollar a ton. They did a little cutting following the first advance, and the practice is now more general. The trade custom is to book jobbers with contracts prior to an advance, these contracts being good for specifications during the ensuing 60 days. After the first advance jobbers specified liberally on their old contracts, but retailers and large consumers did not take hold in like manner, with the result that jobbers were left with large stocks, and at the moment they are buying rather sparingly.

On March 10 the United States Steel Corporation reported its unfilled orders at the close of business Feb. 28 at 3,400,543 tons, representing a gain of 289,624 tons during February, against a gain of 436,162 tons in January, and a continued loss during 1910 aggregating 3,252,274 tons for the year.

Pig Iron.

The total turnover in pig iron in March was perhaps a trifle larger than in February, that month having shown a good gain over December or January. There was a barely perceptible tendency toward higher prices in March. At Buffalo and Cleveland the asking prices of furnaces were advanced about 25 cents, but only after considerable sales had been made at the low point. In eastern Pennsylvania a sharp rise occurred in basic iron, from $14.50 to $15.25, delivered Philadelphia. In the South the $11, Birmingham, basis was simply maintained, with $11.25 to $11.50 demanded for deliveries after July 1. In the valley market nothing of interest occurred except the development of a price on foundry iron for delivery over the second half, at 25 cents advance over the prompt and second quarter market. Basic iron was practically dormant, the sales noted in last report seeming to have filled consumers' requirements. Bessemer moved in small lots at the regular price. We quote at valley furnaces, 90 cents higher delivered Pittsburgh: Bessemer, $15; basic, $13.75 to $14; No. 2 foundry, $14 to $14.25; malleable, $13.75.

Steel.

Shipments of semi-finished steel in March showed a large increase over February, but almost entirely by reason of heavier specifications on existing contracts, there being very little fresh buying. Bessemer and open-hearth billets remain quotable at $23, Pittsburgh or Youngstown, and sheet bars at $24. Several of the wire mills advanced their price on rods from $29 to $30 at the time of the advance in wire products, but it seems probable that material can be picked up at the old figure.

Finished Steel.

Finished steel prices are held firmly all along the line, with the exception of shading on wire products by some jobbers ,as already noted. Prices f. o. b. Pittsburgh, except where otherwise noted, are as follows:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3 in. to 15 in. inclusive, zees and angles, 2 x 3 and larger.

Steel bars, 1.40 cents, base; iron bars, 1.35 cents, Pittsburgh; 1.25 cents, Chicago.

Wire nails, $1.80, base, per keg; plain wire, 1.60 cents, base; galvanized barb wire, 2.10 cents; painted barb wire, 1.80 cents.

Black sheets, 28 gage, 2.20 cents; galvanized, 3.20 cents; blue annealed, 10 gage, 1.65 cents; painted corrugated roofing, $1.55 per square; galvanized, $2.75.

International Congress of Applied Chemistry, 1912

A preliminary announcement has just been issued giving the present status of the organization of the congress to be held in this country from Sept. 4 to 13, 1912.

This same announcement contains tentative rules on the publication of papers and on sectional procedure and the suggestion of a special interests' registration bureau. As these are matters of greatest immediate importance, they are given here slightly abbreviated. We will be glad to place our correspondence columns at the disposal of our readers for discussion and criticism. These tentative rules should be made final as soon as possible, but before this is done they should be clearly understood and discussed.

Tentative Rules Governing the Receipt of Papers for Presentation or Publication.

All papers should be in the hands of the American committee on or before July 1, 1912. All such papers should be presented in *duplicate,* legibly written, but preferably typewritten. *Each paper must be accompanied by an abstract* thereof also in duplicate, legibly written, but preferably typewritten.

All papers received prior to July 1, 1912, and accepted for publication will be printed prior to the meeting of the congress and grouped according to the sections to which they are assigned; papers received after July 1, 1912, and accepted for publication will be printed prior to the meeting of the congress if practicable, but it cannot be guaranteed that they will be placed in the section to which they belong, but they may appear in an appendix.

The American committee will neither revise nor edit any papers or abstracts; papers received for publication will be printed *in extenso* as offered, or only the abstract accompanying the full paper will be printed, or the paper will be printed by title only, together with the name and post-office address of its author, or the paper will not be printed at all, as may be decided in each case by the committee on papers and publications.

Authors will not receive printer's proofs of papers or abstracts; authors must do their proofreading on the manuscript; whatever is printed by the American committee will be printed in exact accordance with whichever of the authors' manuscripts is selected for publication.

Discussions will be recorded in the official language in which they are uttered, and participants in the discussions will have an opportunity of editing the manuscript report of such discussion; the American committees will print from such edited

manuscript report of the discussion, and printer's proofs will not be sent to participants.

No paper which has previously been published shall be read at the eighth congress nor printed in its final report without the consent of the sectional executive committee, the committee on papers and publications and the executive committees of the eighth congress having first been obtained.

The American committee will proceed to print the papers during the first half of July, 1912. The size of the edition printed will be determined by the number of membership fees received on or before July 1, 1912.

Persons contemplating membership in the congress should have their membership completed prior to July 1, 1912, in order that they may be sure of receiving a copy of the report of the congress. Membership fees can be received after July 1, 1912, only as subject to the condition that copies of the report of the congress cannot be guaranteed and will be supplied only until the number of extra copies printed is exhausted.

Tentative Rules on Sectional Procedure.

The presentation of the subject matter of any paper before any section or subsection shall not exceed ten minutes; if the original paper requires more than ten minutes for reading, it must be suitably condensed.

All participants in discussions must speak from the rostrum and not from the floor of the house; where practicable, discussions should be handed in to the sectional secretary in writing and in advance.

Discussion is limited to five minutes for each participant unless consent of those in attendance be obtained for an extension of that time.

According to the proposed order of business, the reading of minutes of the preceeding meeting is to be omitted except in so far as they may contain resolutions or motions calling for action by the section or subsection at that meeting.

After the reading of the minutes, if any be read, the order of the day shall be taken up.

The sessions of sections and subsections will begin at 10 in the forenoon and last until 12 noon; the afternooon sessions will begin at 2 and close at 4 o'clock, except on those afternoons on which there shall be public or general lectures, when the afternoon sessions will begin at 1 o'clock and end at 3. Any papers on the order of the day not reached during the session to which they are allotted will be taken up at the meeting next succeeding in which the order of the day is completed prior to the hour of adjournment, and not otherwise. No paper is to be taken up for presentation within 15 minutes of the adjournment hour, except by unanimous vote of all members present.

Special Interests Registration Bureau.

It is proposed to establish at the eighth congress a *Special Interests Registration Bureau.* This will be distinct from the Central Registration Bureau. Members concerned with any one of the very special phases of chemical industry, and prepared to exchange information with their professional brethren, may register under such specialties. The list of registrants in each specialty will be made accessible to all, either in a pamphlet printed and distributed daily or by exhibiting copies of the typewritten list on bulletin boards. Is such a Special Interests Registration Bureau desirable? If such a special registration is established, what are the titles of the different special phases which should appear at the head of such registration lists in order to be sure of meeting reasonable demands for a minutely specialized classification of chemical interests?

These two questions should be discussed and answered by chemical and allied societies as well as by individual chemists in order to enable the American committee to decide the advisability of such a bureau.

The opening meeting of the congress will be held at Washington, D. C., on September 4, 1912. The other meetings, business and scientific, will all be held in New York. This will begin on Friday, September 6, 1912, and will end on Friday, September 13, 1912. The president of the United States is the patron of the congress, Dr. E. W. Morley the honorary president, Dr. W. H. Nichols the president.

Dr. B. C. Hesse, 25 Broad Street, New York City, is the secretary of the congress.

CORRESPONDENCE

Separation of Oxygen by Cold.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In the January issue of your journal there is an article with the above title which constituted the address of Mr. James Swinburne before the Faraday Society in London.

We are accustomed to receive as gospel in the scientific world anything which bears the hall mark of the Faraday Society, but some of the statements in that paper are so at variance with other statements made before equally august bodies of scientists that it seems worth while to call attention to the discrepancies to the end that the truth may be brought out more clearly and definitely.

One of the statements which constitutes a discrepancy from the accepted facts involves a purely theoretical consideration—the other a purely practical one. The former is contained in the sentence:

"The mixture of two mutually inert gases thus involves increase of entropy. . . . As this depends solely on the logarithms of the free paths it depends in the case of two gases at the same temperature and pressure only on the relative volumes of the gases before they are mixed."

The immortal Clausius invented entropy in the broadest sense—that is, he discovered the function and took the name from the Greek. His definition was, roughly, that the change in entropy of a given body between two temperatures was the integral of the infinitesimal quantities of heat passing at each temperature divided by that temperature. The whole of thermo-dynamics is founded on this definition. There may be mathematical laws which give a relation between the entropy of a gas and the length of the free path of its molecules, but it is not evident to this deponent how there can be any increase in entropy by mixing volumes of different gases at the same temperature and pressure. It would seem, in fact, that it were demonstrable that this were a mathematical impossibility.

If your journal is to give the weight of its countenance to articles containing statements so at variance with commonly accepted physical definitions it owes to its readers some explanation of the differences, or a disclaimer of responsibility for the ideas set forth.

The other discrepancy involves a difference in absolute facts between the statements of Mr. Swinburne and those made by M. L. Marchis before the International Congress of Refrigerative Industries, held, I think, in Paris in 1909. The title of M. Marchis' paper is "The Production of Low Temperature and Refrigeration." It is republished in the report of the Smithsonian Institution for 1910, recently issued. All the articles republished in this report are taken from the very highest authorities and publication therein implies the approval of one of the leading scientific bodies of America.

On the subject of separating oxygen from nitrogen Mr. Swinburne declares that to produce a ton (of oxygen) would need 120,000 kw-hours. This result he reaches by a theoretical reasoning which the present writer is unfortunately incapable of understanding, but the statement is clear and distinct.

On the other hand, M. Marchis states that the Société L'Air Liquide has built apparatus which actually do deliver 100 cu. m of pure oxygen per hour, and that these apparatus yield 1.19 cu. m of pure oxygen per hp-hour, or 3.67 lb. If Mr. Swinburne were using the English ton of 2240 lb., 3.67 lb. would go into this 610 times, and, therefore, to produce one ton of oxygen would require 610 hp-hours, or 455 kw-hours.

This is about 1/300 part of the power declared by Mr. Swinburne to be necessary for this operation.

The writer personally has a letter from the Société L'Air Liquide in which they offer to furnish an apparatus which will deliver 500 cu. m of pure oxygen per hour, of 1545 lb., with only the power required to compress 2500 cu. m of air per hour to a pressure of 10 atmospheres. The power required for this operation in a good compressor would not exceed 300 hp, so that that for a ton would not exceed 435 hp, probably not over 400 hp. This is just 1/400 part of the power claimed to be required by Mr. Swinburne.

The discrepancy between the power required for the apparatus of 500 cu. m and that of 100 cu. m capacity is easily accounted for by the greater efficiency to be expected from the larger apparatus.

It is well to remember that the first steam-propelled vessel which came from England to the shores of the United States carried copies of a pamphlet which proved mathematically the impossibility of a steamship ever crossing the Atlantic. Even in science the prediction of the impossible is dangerous.

Think what a fool anyone would have been thought 15 years ago who declared that a substance would eventually be found which gave out heat in large quantities without combustion and kept it up for an almost infinite time! Yet this impossibility of that recent date is now serving as the key to unlock many of Nature's doors hitherto sealed to us.

This letter is written in the hope that someone capable of doing so will explain these frightful instances of the "doctors disagreeing" for the benefit of an unfortunate engineer, himself unable to read the riddle.

John J. Smith.

[We trust that our correspondent's interesting letter will bring forward an instructive discussion and the desired information.

With respect to the theoretical point raised by him in the first place, we are unable to see how any discrepancy can be claimed between the classical theory of thermodynamics and Mr. Swinburne's statement that when two mutually inert gases mix (so that the mixture occupies a volume equal to the sum of the volumes of the two gases before the mixture) the entropy is increased. According to Clausius' principle that for any irreversible process in nature the entropy increases, Mr. Swinburne's statement means simply that the mixture of two gases in the way described is irreversible, i.e., that it is an impossibility to separate the two gases from the mixture again so as to bring them back into their original separate condition, without causing some other change as a matter of compensation. Mr. Swinburne's statement means that a separation of the gases, as just sketched, is impossible whatever kind of cyclically operating machine one may try for the purpose, the only condition being that when the gases are separated, everything else, including the condition of the cyclic machines, should be exactly in the same condition as at the beginning of the process. On closer analysis the impossibility of the imagined process will be found to lie in the fulfilment of the last-named condition.

To calculate numerically the increase of entropy involved in the mixture of the two gases we must devise an imaginary process by which (for instance, with the aid of imaginary semi-permeable diaphragms) the mixture of gases is separated into its constituents, every infinitesimal step of this process being a reversible equilibrium-step (i.i., if heat is given off to or absorbed from an outside heat reservoir there must be temperature equilibrium between our system of gases and the heat reservoir). The Clausius integral referred to by our correspondent, of the ratio of the infinitely small heat quantity to the absolute temperature, extended over our whole imaginary process, must then be calculated. In this way, which cannot now be sketched in greater detail, the numerical value of the increase of entropy due to the mixture of the two gases is found. The entropy increase, derived in a slightly different way, may be found, for instance, in Planck's Vorlesungen über Thermodynamik, §238, where it is shown to have an essentially positive value.

We may call attention here to an excellent and highly instructive little book by Mr. James Swinburne on "Entropy or Thermodynamics from an Engineer's Standpoint" (London, 1904). In this book some of the fundamental questions about entropy are clearly explained in accordance with classical thermodynamics; that is, in contradistinction to the vague statements often found in engineering textbooks—which would seem to be at the bottom of the theoretical scruples of our correspondent. As to the discrepancy between Mr. Swinburne's theoretical results and the practical results to which our correspondent refers, we trust an explanation will be forthcoming. —Editor.]

Heat Losses Through Furnace Walls.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Mr. Carl Hering, in your March number, page 125, makes some criticisms of remarks which I made in Chicago before the American Electrochemical Society in discussing Mr. Snyder's paper on the "Heat Losses Through Furnace Walls." My remarks are in Volume XVIII of the *Transactions* of this society, on pages 245 and 247. After carefully re-reading them and considering Mr. Hering's criticisms I feel convinced that the best answer to the latter is simply to repeat the former. My words were as follows:

"I think it is extremely important for electric furnace manufacturers, particularly, to study the question of the amount of heat transferred through the furnace walls. If they are experimenting with a given thickness of wall with a given furnace, there is one particular which it is important to bear in mind. Given an electric furnace with a wall of a given thickness, and experimenting with different materials in the walls to find the one which conducts the least heat from the furnace, you have a very good way of comparing the different materials by measuring the temperature of the outside of the furnace shell. The total heat loss from the furnace will vary with the temperature of its outside. I found a very convenient means of doing this by taking an iron-nickel thermo-couple and holding the junction against the furnace wall. In that way you can measure temperatures up to 500° C. With a given external surface the higher the temperature outside, the more heat you are losing; and the more efficient the heat insulation the lower the outside wall temperature will be." * * *

"I meant to say that for a given thickness of furnace wall, comparing different materials in that same thickness of furnace wall, you can measure the temperature of the outside and thus get a comparison of different materials composing a wall of given thickness."

And now, Mr. Editor, allow me to admit that I have repeated these words not for the pleasure there might be in refuting Mr. Hering's criticisms, but because of their real importance to all electric furnace constructors. If so keen a scientist as Mr. Hering is so slow in seeing their truth, and even rushes into print to confute them, it is surely worth while repeating them and commending them again to the serious attention of all interested, with the emphatic assurance that, as will be evident on proper reflection, they are important and true.

Lehigh University, March 12. Jos. W. Richards.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In the above letter of Professor Richards I notice that he does not quote all that he said in that original discussion, but that he has now omitted some of the very parts which gave rise to the present discussion; the portions now quoted in his letter, when read by themselves, would have called for a different comment than the one I made in my previous letter. I think that almost anyone reading all that he originally said, especially on page 246 of the *Transactions* (which page I notice he omits referring to in his letter), and realizing that the last paragraph quoted above beginning with

"I *meant* to say" certainly seems to signify that he did not mean what he had said before and was given as a *correction* of his former statements, will agree with me that, taken as a whole, it was quite confusing and that it was difficult to find out what he really did mean to say.

One of the confusing statements omitted from his present quotation is: "The higher the external temperature the less the heat gradient from the inside to the outside and, therefore, the less heat is being taken out of the furnace." But in the part quoted above he says that a higher outside temperature (hence a *less heat gradient*) means a *greater* loss, hence just the contrary; why should a less heat gradient sometimes mean a less loss and at other times a greater loss; he neither then nor now offers an explanation of this discrepancy or apparent contradiction. And in his apparent correction of this confusion (on page 247, namely, the last paragraph quoted by him above) he omits entirely to say whether he then meant that it is a higher or a lower outside temperature that indicates the greater loss. This certainly seems to indicate that he was not then sure which it was, and did not wish to commit himself.

It was because I thought others, like myself, might be confused by this apparent contradiction that I endeavored to show in the March issue that in the general case, barring the minor influences, the drop of temperature by itself, or the heat gradient, is not necessarily a measure of the relative losses, although it is in some specific cases.

Whatever Professor Richards meant to say, the main feature of his remarks certainly was that the temperature of the outside of the wall, and therefore the heat gradient when the thickness is the same, was a true measure of the loss, and was even "a *very good* way of comparing the different materials" in the walls. It is also a fact that in his final corrected statement he refrains from saying whether a greater heat gradient means a greater or a less loss.

I have endeavored to explain the true facts in the case in an article published elsewhere in this issue, the general conclusions in which are that the drop of temperature in the walls, as a general rule, is by no means a criterion of the loss, but that it is so in stated specific cases; that under certain specified conditions the higher the outside voltage (that is, the less the heat gradient) the less the loss, and under others, the greater the loss; but that in the latter case (which appears to be the one he now wants it understood that he meant) the outside temperature would be a very poor and in fact dangerously bad method of measuring the goodness of the material of the walls, and would therefore surely not be "a very good way of comparing different materials" and should be condemned rather than encouraged. It was therefore to his credit that in my first reply I did not assume that he wanted to be understood that he was referring to this particular case.

If, for instance, he were to measure the same furnace in summer and again in winter without knowing it was the same one he would have to erroneously conclude that the one measured in winter had the better wall material. Or if the supposedly constant inside temperatures in two different tests of two comparative furnaces happened to differ somewhat in the two cases, which is very likely to be the case, the relative positions of the two wall materials as to their insulating power might actually be reversed! I have shown by actual examples in that article that the outside temperatures, even under the conditions most favorable to Professor Richards' present apparent argument, which conditions it would be practically impossible to realize in practice, would vary so little with very widely differing materials that it would be very inadvisable to attempt to draw such conclusions from them as he recommends, or to place the reliance that he does on the results.

If Professor Richards now means what I understood the latter part of his original discussion to infer that he did not mean to say, namely, that the greater the heat gradient the less the loss, I can hardly assume all the blame for having at first misunderstood what he meant by the words he then used, and for being "so slow in seeing their truth" (sic), and I would respectfully suggest that he might have explained himself more clearly if he understood it clearly at that time.

My reply to his letter, therefore, is, that I regret that I cannot attach the same "real importance" which he does to either the "words" he used or to his method, and that I cannot endorse his "commending them (both) again to the serious attention of all interested, with the repeated assurance that, as will be evident on proper reflection, they are important." Whether my "reflections" on this subject are "proper" or improper must be judged from my views as expressed in the article on another page of this issue. If the statements in that article are true and correct, as I believe they are, and if they have assisted somewhat in clarifying the subject, I have no regret that I "even rushed into print," which under such circumstances I consider a professional duty to do.

Philadelphia, Pa., March 15. CARL HERING.

* * *

To the Editor of Metallurgical and Chemical Engineering:

SIR:—Referring to above communication from Mr. Hering I am sorry that he is apparently so slow in appreciating the correctness of my original remarks which he criticises.

My remarks were directed to bringing out two principles:

(1) If you keep all other conditions the same, and substitute one lining material for another in a furnace, then the better heat insulator said lining is the lower will be the temperature of the outside surface of the furnace and vice versa. Therefore, when experimenting with linings and substituting one material for another in the wall, the measurement of the outside surface temperature is a reliable index of the heat insulating quality of the wall material, the lower outside surface temperature indicating a better heat insulator and vice versa

(2) If you keep all other conditions the same, and substitute one kind of outside surface for another on a furnace, then the less heat radiating and conducting power said surface has the less heat will be lost from the furnace, although the outside surface temperature will be *higher* than it was before, and *per contra* the greater the heat radiating and conducting power said surface has the more heat will be lost from the furnace, although the outside surface temperature will be *lower* than it was before.

Fearing that the latter generalization was too difficult to be seen at once I expatiated upon it, explaining that the paradoxical statement would be better understood if it was realized that, in case (2), when varying the outside surface only, the higher the external surface temperature the less the heat gradient through the furnace wall, and therefore the less heat could pass from the furnace.

Mr. Hering has apparently not been able to see through these generalizations. He applies what I said about the heat gradient in the walls when varying the outside surface (case 2) to the conditions when varying the material in the walls and keeping the outside surface the same (case 1)—to which, of course, they do not apply.

In closing, I would say that I consider my original remarks, criticised by Mr. Hering, as clear and correct. They are not easy to understand, it is true, and if they had been merely self-evident platitudes there would have been no justification for my saying them. Mr. Hering has apparently, in good faith, been unable to understand them. If any reader of this journal is inclined to think the criticisms made upon them correct I would ask only one favor of him, viz., that he *study* carefully my original remarks, in Vol. XVIII of the *Transactions* of the American Electrochemical Society, pages 245, 246 and 247, and in doing so avoid the mistake which Mr. Hering made of confounding the two aspects of the experimentation there clearly distinguished from each other.

Lehigh University, March 16. JOSEPH W. RICHARDS.

The Opening of the Chemists' Building.

On a fine crisp spring day, full of glorious sunshine, in the presence of representatives of chemical science and industry, assembled from many parts of this country, the new Chemists' Building at 50-54 East Forty-first Street, New York City, was formally dedicated on the afternoon of March 17.

The building and equipment has been described in our issue of last December (vol. VIII, page 701). The purpose for which it was erected and the ideals which it represents are admirably stated in the opening address of Dr. Morris Loeb to whom more than any other individual the thanks of the profession are due for the transformation of the idealistic conception of a Chemists' Building into an actual and beautiful reality.

Dedication Exercises

Dr. Morris Loeb, as president of the Chemists' Building Company, made the following address:

"I rise to welcome you on behalf of the directors and stock-

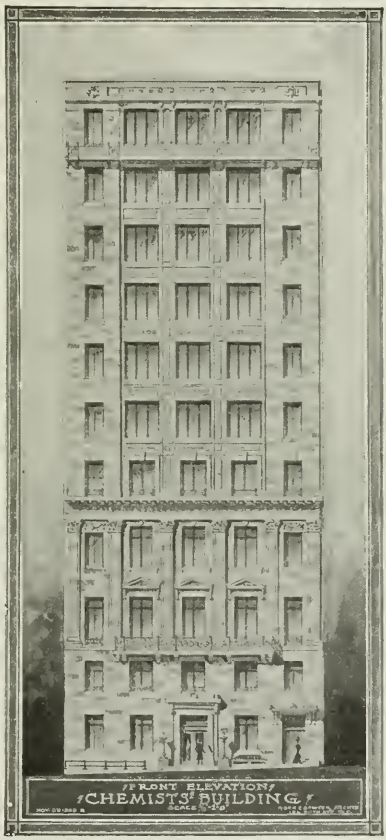
MORRIS LOEB

holders of the Chemists' Building Company and to thank you for the interest which your presence indicates in the formal opening of a building which we believe to be the first of its kind, not only in this country, but on earth. It is true that Berlin possesses in the Hofmann-Haus a home for the German Chemical Society and that London owes to the munificence of the late Ludwig Mond its Faraday Laboratories for Chemical and Physical Research. But this new building in which you find yourselves is planned to serve under one roof the social, intellectual, and practical needs of the chemical profession not of New York alone, but of our whole beloved country.

"The means for its construction have been furnished by men, many of whom can expect to share to but a slight degree in the benefits of the Chemists' Club, which occupies so much of its floor space. These shareholders see in it the incarnation of some of the ideals that led them to the pursuit of chemistry, pure or applied, as their lifework.

"For, strange as it may seem to the layman, who has seen the ugliest blots on a landscape designated as chemical factories, who has sniffed with disgust a chemical odor, has been urged to believe that the chemist's shadow contaminates pure foods, and has been taught in school that alchemy spelled fraud and sorcery, our science is one calculated to develop the ideal side of human nature, and the chemist, more perhaps than the votary of natural science or the devotee of the so-called humanities, is led to an intense interest in human development.

"Our science aspires not only to know, but also to do. On the one hand, it leads us to delve into the secrets of nature, in the minute atom as well as in the far distant stars, in the living cell as well as in the crystallized relics of the convulsions from which this earth was born; on the other, it leads us to apply this knowledge to the immediate needs of man, be it in safeguarding his health, in ministering to his material or esthteic



CHEMISTS' BUILDING

wants or in regulating his commerce and in facilitating his utilization of the earth's resources. These many points of contact with nature and with human interest will be the theme of the eminent men whom I shall have the privilege of introducing this afternoon and will be illustrated at greater length in the scientific meetings which are to follow; it is enough that we recognize at this moment that this versatility of method and of purpose must necessarily enlarge the viewpoint of the chemist, and to seek therein the motives for the ready cooperation in the present enterprise.

"Our shareholders content themselves with a moderate return for their money and have agreed that surplus profits shall accure to the benefit of chemical science. They hope that the facilities afforded within these walls will redound to the benefit of mankind and the prosperity of the country.

"May I emphasize the word 'facilities'? There are two ways

of aiding a man or a cause: by addition to the income or reduction of the expense. The pecuniary result to the beneficiary may be the same, but the moral one is far different; it is not only the beggar who is pauperized by the cash gift and uplifted by the aid which enables him to earn his own livelihood. Arts and sciences may be stimulated by prizes and scholarships beyond a doubt, but the relation between donor and recipient is not free from restraint, and the probability of human error in the selection of the right incumbent makes the method a wasteful one at best.

"Far better is it to remove those obstacles which hamper all work equally and are felt more severely by those whose means are restricted and who have not yet earned the recognition of the world at large. The laboratory student is encumbered by certain restrictions in America more than elsewhere. The higher price of commodities affects him in many ways besides the higher cost of living, rent of laboratory, installation of equipment, purchase of supplies, salary of assistant, acquisition of books. The advanced cost of labor militates against the construction of certain grades of apparatus and the preparation of the rarer chemicals in this country; and a curiously unscientific tariff law, while pretending to lift the duty from articles required for educational purposes, practically forces the colleges to make their purchases abroad and prevents American dealers from carrying an adequate selection of imported material. It is no exaggeration to state that this duty-free importation clause, as interpreted by the United States Treasury Department, forces the American chemist to wait from two to three months before making an experiment for which he could obtain his material in two or three days if he were working abroad.

"To remove these disadvantages in time and cost, to provide easy access to books and apparatus, to make room for the independent scientific worker are the ideals which hovered before the eyes of those who planned this present enterprise. Time will show whether they can all be realized, but whatever is done in this beautiful building, which we are about to dedicate, must open free opportunities to all and show favoritism toward none if the trust imposed upon its management be administered in the spirit of those who have contributed toward its erection.

"A library of the highest scientific importance and a museum of chemical substances will be available for every reputable chemist; laboratories for temporary as well as permanent use will be at the disposal of the earnest student; help and advice will be extended to the struggling beginner; good comradeship and hearty co-operation will characterize the Chemists' Club, and this auditorium, soon to be named after the first great chemist of American birth, will ever minister to his ideal of the application of science to the useful arts."

Dr. Loeb concluded with words of appreciation for the work of Messrs. York & Sawyer, the architects, of Mr. Ralph C. Crow and his associates in the construction company, of Messrs. Marks & Woodwell, the consulting engineers and of Mr. Chas. Eidlitz and Messrs. Black & Boyd, the contractors for the electrical and lighting equipment. "And now, ladies and gentlemen, I have given you an account of the empty shell. The spirit which is to animate it will be brought to your minds by the speakers whom I have the privilege to introduce—master minds of our science and guides of the future of its leading organizations."

Dr. Loeb was followed by Dr. Edward S. Morley, who spoke as the honorary president of the Eighth International Congress of Applied Chemistry, to be held in September, 1912, in this country. While the organization of the congress so far consists of officers only and no crew, yet the congress rejoices in the opening of the Chemists' Building, which will be its headquarters, and the congress transmits its hearty congratulations.

Prof. Alexander Smith spoke as president of the American Chemical Society on research work. Two requirements are absolutely necessary in an investigator—first, a proper training, and, second, a true spirit of research or thirst of discovery. Both belong together. Professor Smith explained how the Chemical Society had helped possible investigators to find themselves and expressed the conviction that the new Chemists' Club Building would offer magnificent opportunities in the same direction.

A letter from Prof. Ira Remsen, president of Johns Hopkins University, was read, expressing his regrets of being forced to be absent. As past president of the Society of Chemical Industry Prof. Remsen referred to the relations between science and industry.

A letter from Prof. Edgar F. Smith, provost of the University of Pennsylvania, also expressed the writer's sincere regrets of not being able to be present.

Prof. William H. Walker, of the Massachusetts Institute, then spoke as president of the American Electrochemical Society. The field of chemical science is now so large that no one person can expect to cultivate more than a small portion of it. Electrochemistry deals with the chemical changes brought about through the application of electrical energy, and within this limitation has grown up a strong, virile organization, international in its scope, the American Electrochemical Society.

But, as Dr. Walker pointed out, "the specialization of to-day tends to carry us apart—tends to a divergence of interest. This new chemists' building will function as a center to which every chemist, however diverse his interest, will be drawn. There was a time when the advice to a young man was 'Paddle your own canoe.' Each chemist did this to so great an extent that he worked for himself, by himself, without thought of others. The national chemical societies have already done a grand, good work in bringing men engaged in chemistry together, but this Chemists' Building will be an additional and most potent factor in placing the profession of chemistry where it by right of its importance belongs. No longer will the cry be 'Paddle your own canoe,' but, rather, 'Get the stroke and pull together.' It is this sentiment which is exemplified in the Chemists' Building. Here we can present a united front, and strive for the further uplift of the chemical profession and the community."

To Dr. Loeb and the men who have stood by him Professor Walker tendered the grateful thanks of the American Electrochemical Society. "And again I would greet the Chemists' Club, and wish for it many long years of usefulness in which to enjoy the splendid new building which to-day is theirs."

Prof. Wilder D. Bancroft, of Cornell University, then presented in his characteristic humorous manner a brief yet suggestive address on chemical education. He said that the trend of the many discussions in recent years had been how to teach a man so that he becomes a technical chemist, so that he knows all about machinery, about the handling of materials on a large and small scale, about the handling of men, about questions of costs, etc.

All these, Professor Bancroft said, are valuable and interesting problems. But people who want to instruct students in these subjects overlook several very important points. Firstly, that it would require a prohibitive expenditure of time. Secondly, by giving a man a proper drill, can you really make him a works manager? Why is a works manager paid a big salary? Because he is relied upon to pull the plant out of unforeseen difficulties. An ideal plant, running with mathematical accuracy, would not need a manager.

Professor Bancroft thinks there is something wrong with our methods of education, but he finds the trouble to be rather in another direction. Every young boy asks in a few minutes a great many more times "why"? than his parents can ever answer. But when the same boy has passed ten years through a school and four years through a university he no longer asks "why"? Why this is so—this would be a good subject of future discussions tending toward an improvement in chemical education.

Then followed the unveiling of the tablet of Count Rumford, after whom, as the first great American chemist, the Auditorium is named. Dr. Frank W. Clarke, of the United States Geological Survey, gave a very witty history of Count Rumford's life. He spoke of Rumford as one of the real great Americans and dwelt on his statesmanship. He emphasized that Rumford, an American, founded the Royal Institution in London. Yet his greatest discovery was Humphry Davy, and the latter discovered Faraday.

Dr. Loeb then explained the building, its design and equipment, and handed the keys of the clubrooms of the Chemists' Club over to its president, Dr. Russell W. Moore, who replied in a felicitous little speech.

This concluded the formal dedication exercises, which were followed by a reception in the clubrooms.

Scientific Sessions and Banquet.

The evening of Friday, March 17, and the morning and afternoon of Saturday, March 18, were devoted to scientific sessions held under the auspices of the New York sections of the national chemical societies.

The meeting of the New York section of the *American Chemical Society* began with an elaborate historical and philosophical address by the chairman, Prof. Charles Baskerville, of the College of the City of New York.

Prof. Jacques Loeb, of the Rockefeller Institute for Medical Research, followed with a suggestive lecture on the characteristics of living matter from the physico-chemical point of view.

Prof. R. B. Moore, of Butler College, Indianapolis, gave a lecture on the rare gases of the atmosphere. The lecture was illustrated by a series of highly interesting experiments, in which the lecturer isolated with great skill and success the rare gases from atmospheric air.

While the session of the American Chemical Society was thus pitched in a highly scientific key, the proceedings of the other two societies on the next day broadened in their effects, bringing out the human not less than the technical and scientific sides of engineering.

The session of the New York section of the *American Electrochemical Society* started with an address of the chairman, Mr. Elmer A. Sperry, who emphasized that of all branches of applied chemistry electrochemistry is most specifically American.

Dr. W. R. Whitney, chief of the Research Laboratory of the General Electric Company, followed with a lecture on "mental catalysis." This admirable and inspiring address on idealism in science and engineering is printed in full at the conclusion of this report.

Prof. Wilder D. Bancroft then lectured in a very interesting manner on the chemistry of phosphorescence, showing how it is possible to distinguish the chemical reactions which play a part in phosphorescence. After the subject of phosphorescence has been studied in great detail from a purely physical point of view, Dr. Bancroft's way of looking at it from the chemist's standpoint is decidedly novel and promises important results.

The session of the New York section of the *Society of Chemical Industry* in the afternoon began with an interesting address by the chairman, Dr. Charles F. McKenna, who in conclusion read a letter from Hon. James Bryce expressing his regrets of not being able to be with the society on this occasion.

Prof. Wm. P. Mason, of the Rensselaer Polytechnic Institute, followed with a very instructive address on the contributions of chemistry to sanitation, especially water purification.

The session was concluded by an address of the doyen of the chemical professions in the United States, Prof. Chas. F. Chandler, who gave interesting personal reminiscences of the history of chemistry in New York City.

As in classic Greece a serious dramatic trilogy was always followed by a satyrplay, thus the trilogy of serious scientific sessions was followed on Saturday night by the first banquet of the Chemists' Club in the new building.

It was a decided success, in spite of the fact that it was almost too well attended. Frolicsome merriment reigned supreme, and the toastmaster, President Russell W. Moore, and the master of the revels, Mr. Elwood Hendrick, must be highly complimented on their success. Dr. Moore's chief speech was in the presentation of a large silver loving cup to Dr. Loeb—as Dr. Chandler said, only a token of our affection, not a measure. Mr. Hendrick contributed delightful newspaper stories and remarkable telegrams and letters.

Other speakers were Dr. Loeb, Dr. Schweitzer, Dr. McKenna, Mr. Plaut, Prof. Alexander Smith, Dr. Rosengarten, Mr. Parker, Dr. Chandler and Mr. Jacob Hasslacher, whose concluding words were that while the English language had no exact equivalent for the German Gemütlichkeit, the Chemists' Club was the nearest he could think of.

The celebration was fittingly concluded by a concert of the Olive Mead Quartet, on Sunday afternoon, with a program of Mozart, Tschaikowsky, Mendelssohn, and Grieg.

* * *

Mental Catalysis.*

By Dr. W. R. Whitney.

Perhaps you have asked yourself the question, what has this subject of mental catalysis to do with electrochemistry? Therefore, I want to explain.

One might justly expect from me at this time a paper embodying the results of some specific electrochemical research, but when I was asked by your committee to prepare a paper my first impulse was to go back to first principles, and as the title had to be given before the paper was started, I chose a title which might cover a line of thought along which I had already quietly enjoyed some study. You know that in most extended chemical researches a great deal depends on a study of what in Germany we called our "Ausgangsmaterial." A long line of interconnected reactions, of properties and of products are evolved from a certain "Ausgangsmaterial," and we are probably all familiar with the way in which this whole line may be altered by only a small change in the purity of our "Ausgangsmaterial." So, for our science in general, I am going back a few minutes to consider our Ausgangsmaterial.

I suppose that every chemist who is deeply interested in his work finds himself sometimes wondering: Am I doing this in the biggest way, or am I a small machine, working only with my hands? Perhaps I am only a little more complex machine, using a stock of knowledge and certain trained habits merely as a means of livelihood, just as some engine uses steam and eccentrics. Or, finally, is it possible to be what is called a free moral agent, using knowledge, training, talents, and all to their ideal efficiency?

Those of you who have read Maeterlinck's "Life of the Bee" know that it seems as though the bee-hive represented the acme of communistic perfection. Yet we are apparently more highly endowed than bees, and much more is expected of us There seems to be something in us which calls for a continually widening horizon. We must do more than work.

This is an attempt to consider for a moment some of the laws of forces which may underlie successful work and advances in its broad, its only satisfying sense. Of course there are such forces and such laws. We all feel the force of a great, far-seeing mind. We can easily point to our leaders of the past. Our history itself is largely the product of time into some mental forces. I want to imagine that the force which determines advance in any line is *idealism*, and I call it the catalyser. I want to point out that this force in its growth is subject to two entirely different influences, which may be looked into and studied separately. These I may call the *conceptive* and *receptive*. In each of these we all take part. A

*An address made at the extraordinary meeting of the American Electrochemical Society, held at the opening exercises of the new Chemists' Building in New York City on March 18, 1911.

club like this is a place where both the conceptive and receptive influences should be very prevalent.

A number of years ago I overheard a statement which was not intended as complimentary, in which a young chemist was referred to as an idealist. I think the intention of the speaker was to use the word as an antonym of the word "practical." This has often been recalled to my mind, and the results of some thinking on the subject form the matter of this brief talk on Metal Catalysis.

There is no profession where idealism can play a more important part than in our own. Why? Well, because we possess all the elements for material realizations. The exact bearing of idealism is fairly clear to me, and I want to make it clear to you.

A methodical chemist who is solely trained to observe and to record may *enormously contribute* to the science without *advancing* it appreciably. By advance, I mean to infer an increase in our breadth of horizon. I may be faithful, industrious and well educated, and contribute largely to the work of my subject, but there is something which will tell more certainly upon my relation to my work than my erudition. I think this criterion is idealism.

Far be it from me to compare the work of one kind of chemist with that of another, for until we shall have learned just exactly *why* and how we are, we are not in a logical position to compare one another's contributions. I rather want to interest you for a few minutes in the peculiar *attractiveness* of idealism.

"But is this a real thing? Is it practical"? I hear you ask. I answer, "It is as real as we are." Consider its growth. It is safe to assume that our earliest ancestors were gifted with as much as or more physical strength than we are. As organic entities, apparently tending to increase the entropy of the universe, they did not differ much from ourselves. In the stone age a dietetic study would have shown nearly the present calorific change and metabolism. I do not mean to raise the question whether we are better than they or not, though that should be permitted to an idealist. But I expect you to admit that we are different and are growing more so every day.

Our horizon is broader. Now I want to point out that it is our idealism that accounts for the difference. I call idealism that developed power of *prevision* of the more perfect, that mental elasticity which sees improved things before they arrive, which may even see some that never arrive, and which power has hope and optimism for pressure indicators. This is the catalyser.

The doctors of medicine and the specialists in diseases have shown us how, under certain conditions, contagions appear and spread; how they are carried from one plant or animal to another, under what conditions they thrive and how specific immunity may be procured. Such discoveries (relating to the realities of life) apply directly to the ideals. It can be seen in the histories of our mental growth. For example, the spores of plastic art were evinced in the stone age, but the idealism did not develop very rapidly until much later. But we find it almost as widespread and advanced in Egyptian and Greek sculpture before the time of Christ as it is to-day. There was a period and a place in which it made most rapid strides.

Idealism, as expressed in marble, which first had mental form in the minds of a few idealists, spread by their contact with others and soon reached a splendid state of advancement. Many centuries later idealists from Giotto, de Vinci, Michael Angelo and the painters of the fifteenth century spread the idealism of painting so that in their eras the world very rapidly made its greatest advances in painting and in architectural art.

We are so accustomed to the laws governing idealism that we often fail to realize them. A few idealists partly realized their visions in some such ways, and world-advances were made. That is, permanent additions were made to the diameter of the world's mental horizon. Through all these epochs the realist lived. The cost of food, expressed in labor, remained about the same. The question whether a purely mental enjoyment was a utility had long been put, and answered as now. Among the things which kept on acting was idealism.

In the chemist's field we see the value of idealism in matters both small and large. Usually the first expression of an ideal is so unrealistic as to meet opposition or at least friction. I think we can consider some of our hypotheses as ideals. Consider for a moment the benzene ring. We all know its value to us as an idea, but it was once an ideal. The even more idealistic theories of disease immunity, such as that proposed by Paul Ehrlich, with its fantastic receptors and side chains, while leading to very important practical results, still appeal to many as too speculative, too idealistic. But they are more than valuable. They hasten our advance, they catalyse.

I do not care to raise the question of the exact accuracy of the conceptions of the idealist. This we need never know. But as trees are known by their fruits, we should measure the value to us of idealism by the products. We may admit that what idealism accomplishes for us might be reached in time without it. This is the earmark of a catalyser. In catalysis the velocity of a reaction is increased by agents which still exist, unaltered, after the reaction. In many cases the catalysis is plainly one of contact. I have known several men who worked in Edison's laboratory. They are still a definite type of idealist.

I also recall how, just nine years ago, a small group of 11 men actuated by an ideal in the mind of two or three started in Philadelphia the American Electrochemical Society. My reason for referring to it is to point out that it was a *lonesome mental conception* before it became a group of men. I am trying to point out the value of such ideas and their fertilization. I want you to realize their frequent frailty, their need of nurture and their final possible power.

I find an application in this occasion to-day. An epoch-making event is just transpiring. In nearly the largest city in the world the first clubhouse and guild hall devoted to our science is being inaugurated. It was first an ideal, a vision in the mind of a single man. He could have slighted it. He could have said to himself: "We have always gotten along without this thing. Why disturb the equilibrium of the day?" But he persisted, and started a thread of history which will never break. The ideal preceded the real, the mental vision preceded the physical vision. That is the law. The ideal catalised the reaction and we have the product. This clubhouse will do much for idealism, and all I can do now is to point out ways along which it will be done.

The deep student of chemistry, confined entirely to his books and laboratory, has ideals, but contact with others of his calling will contribute enormously to his information and incentives and must, therefore, direct and influence his ideals. This is so well known as to need no enlargement.

The teacher to whom we owe so much and on whom such a great responsibility rests can be aided, through this organization, with increased efficiency. Here he will come into contact with others of his profession and can gain from them encouragement for his ideals. He will also learn of the ideals and advances of the industries and increase his power of imparting ideals to his students.

As I still count myself among the teachers of chemistry and am so counted in Institute catalogs I may be permitted to say that in the teaching of science there is still very much room for improvement. I would go further and say that we have scarcely learned how to teach idealism *at all* and that we are continually in danger of destroying what idealism comes under our observation by the very methods of this system we employ. Most teachers have probably had occasion to notice the loss of what he calls enthusiasm in some student as the latter has been taught more. Of course, his enthusiasm and optimism ought always to grow greater with added powers instead of becoming less.

One thing needed by us as Americans is more idealism. In our times, I think I am safe in saying, idealism is not greatly influenced by climatic conditions, but it certainly is by environment. I don't thnk there is proportionally as much idealism in the English as in the French or German of to-day. I know more about us Americans. We are generally defective in this respect. We are aggressive. We are realists and often, basing our energies upon foreign visions, we can do the rest. This is most marked in the industries, and we know it well in physics and chemistry; but in art and literature we are still copyists. In medicine and surgery we show to better advantage, though I think most of the pioneer conceptions are still of foreign make.

In other words, I believe we have, as Americans, plenty of latent energy, but are still deficient in idealism. We can and do master the problems with which we come into contact, but largely for themselves and possibly for the pleasure of doing something useful. This alone may be commendable, but it can be improved.

A would-be idealist cannot be expected to produce useful ideals if he is out of contact with his fellow-man. We need the idealist as a guide, but a guide who is so far ahead as to be out of sight is a poor guide. This means that the contact catalyser must really be in contact to catalyse.

Huxley, in an address as the rector at Aberdeen, says: "The man who is all morality and intellect, although he may be good and even great, is, after all, only half a man. There is beauty in the moral world and in the intellectual world, but there is also beauty which is neither moral nor intellectual—the beauty of the world of art. There are men who are devoid of the power of seeing it, as there are men born deaf and blind, and the loss of those, as of these, is simply infinite. There are others in whom it is an overpowering passion, happy men born with the productive (or at the lowest) the appreciative genius of the artist. But in the mass of mankind the aesthetic faculty, like the reasoning power and the moral sense, needs to be roused, directed and cultivated."

I consider idealism a name for this faculty to which he refers. It is the faculty of seeing beyond and over the physical dimensions of things. It is the power of being the parent of plans. It is that something which makes us call certain men artists in their profession.

The start we all get in the physical sciences by being taught in laboratories especially equipped to insure contact of student with the thing, rather than with the book or with the man, is an idealism realized in our own day. It was only a vision seen by a few physicists less than a hundred years ago, and this led to chemical and physical laboratories equipped to give experimental work to students. I only mention this to show how we are realizing the benefits of a condition which a short time ago was only an idealistic dream.

I am quite sure that within the lifetime of some of us here he would have been smilingly and pityingly called an idealist who had dared predict an 11-story building for a chemists' *club*. It seems realistic enough to us now, but half a century ago there were hardly chemists enough in the country to keep a chemical society alive, and they fined themselves for absences from meetings, too. Such advances are not exactly like "Topsy" in "Uncle Tom's Cabin." They don't just grow. They can be traced to some ideal in every case.

In a hearing before the Massachusetts Legislature the other day, Mr. F. P. Fish, formerly president of the Bell Telephone Company, made the following remarks, which are idealistic, but warranted, and are of interest to us:

"I should like to prophesy that we are really just at the beginning of this industrial development. During the last generation we have seen the extraordinary development of electricity, which is not by any manner of means completed. I am satisfied that during the next generation—the next two generations—there is going to be a development in chemistry which will far surpass in its importance and value to the human race that of electricity in the last few years. A development in chemistry which is going to revolutionize methods of manufacture, and, more than that, going to revolutionize methods of agriculture. I personally believe that in a generation or two we shall see the deserted farms of New England, if there are any, blooming with vegetation because of the work that is done by chemists. * * *"

Idealism? Certainly, but practical to the core. Why? Because it helps produce. It catalyses. This spirit is almost invaluable in assisting in realization. Bare mechanical work in any field is a heartless thing. Its tendency is to habit and unaltering self-perpetuation, but the idealist interferes and keeps pointing to something more perfect, more interesting.

Henry Van Dyke, in writing on creative education, said:

"The right ideal of education in a democracy is the creative ideal. It does not seek to adorn men with certain rare accomplishments which shall be the marks of a Braminical caste. It does not seek to train men for certain practical pursuits with an eye single to their own advantage. It seeks, by a vital culture, to create new men and new kinds of men who shall be of ever-increasing worth to the republic and to mankind." And, again: "It has four definite marks: the power to see clearly, the power to imagine vividly, the power to think independently and the power to will nobly. These are the objects that the creative ideal sets before us."

Herbert Spencer also saw the operation of idealism when he said that society was a growth and not a manufacture and has its laws of evolution. He says: "Our immense manufacturing system, with its multitudinous inventions supplying both home and foreign consumers, * * * have been naturally and not artificially originated. * * * Knowledge developing into science (which has become so vast in mass that no one can grasp a tithe of it and which now guides productive activities at large) has resulted from the working of individuals prompted not by the ruling agency, but by their own inclinations." It is the direction given to these inclinations that determines idealism. This is a flexible factor. It depends upon us in both the efferent and afferent, our production of ideals and our reception of them.

One can hardly look into the history of any leader of a profession without seeing the effect of idealism, and yet we are more apt to think first of the individual's accumulated knowledge or his products. Think, for a moment, of the idealism of a Faraday. It is really more to be admired than his tangible work. It continues to operate long after he has passed away. The way he undertook his work we call ideal. Everyone admires his approach to his subject, his deference. The ions he idealized we know by name and by weight. The lines of the magnetic field, which were an ideal to him, have the bearing of working realities to us to-day.

I did not start with the thought that I would be able to perfectly declare the genesis of idealism, or mental catalysis, any more than I might explain chemical catalysis. But just as we know a few of the laws governing the latter, so we may cite a few principles of the former. Idealism grows better in a warm, receptive atmosphere than in a chilly or doubtful one. In other words, while I do not claim at this point that idealism can be made autogenetic or that one can will himself an idealist, I know that one may increase the fertility of the soil in which the plant grows.

I cannot help thinking at this moment of the late S. P. Langley, of the Smithsonian Institute. As an American citizen I am ashamed of myself at the mention of his name. He was called an idealist. One of his ideals had to do with aeronautics. I believe he did more to advance the knowledge of the subject than anyone prior to his day. He did not capitalize his idealism, either. He built a machine which was almost identical with our present successful types, and it flew half a mile under its own power. It was as healthy a set of experiments as we American people have ever paid for, but because his ideals were not completely realized in the first test we laughed

him to death. We may know enough as a practical people, our morals may accord with any standard erected, but our appreciation of idealism needs a lot of fertilizer.

It is in such a place as this club that idealism in chemistry will be fostered. The atmosphere can be made to favor its growth. Instead of shunning some idealist as a crank (and the pet of his idealism as a freak) we can show him we have learned that most essentially new things have been called freaks ever since the word was known. What is better, perhaps, we can learn that it greatly helps in the realization of ideals to assist them.

We recognize the effect of idealism when we see it in such art as that of Michael Angelo. We see, too, that there must have been a great deal of idealism back of the wingless Victory of the Acropolis or the winged Victory of Samothrace, because even now they are perfect to us, but we ought also to find it nearer home. We say it is in poetry, but it may also lie in prose. If it belongs to art, it also belongs to science. If it has existed, it still exists. Perhaps the law of conservation may be applied to it, as it is to mass and energy.

It certainly does not grow as a result of mere work, however hard. It is not measured in watt-hours or foot-pounds. It cannot be forced. Dr. Gulick has said: "Compulsion fails to account for the greatest things in the world. One cannot imagine, for example, that those poems which bless us with their beauty and strength, with their vision and inspiration, were written under a compelling sense of duty. Such poems as Mrs. Browning's "Portuguese Sonnets" spring from sources other than those of necessity. It is so with the world's great statues. There was a vision of beauty, an ideal within the souls of the men who made them, and this ideal was so compelling that they worked it out, oftentimes at great personal sacrifice." He also says, "When we say to a man, 'Play the game,' we mean treat the situation as an ideal." It means to drop all selfish individual considerations and to meet the real situation by ideal means.

There are times when the most of us feel that we know more about mental paralysis than about mental catalysis. At the end of a day's work, when the practical difficulties seem to have been particularly devised and peculiarly adapted to meet our special insufficiencies, we are apt to feel ourselves far removed from the ideal of anything. A clubhouse such as this ought to help dispel such feelings. When we are unable to bring forth pet flowers of idealism for the sake of the fertilization that others can give them, we can at least carry a little watering-pot for the opening bud of some other fellow, and that's a use for a club.

Research Laboratory,
General Electric Company.

The Arizona Copper Company has issued its annual report for the year ending Sept. 30, 1910. During the year 599,250 dry tons of sulphide ore were concentrated, the ratio of concentration being 5.14 to 1. Improvements are under way which will result in combining the No. 5 concentrator into the No. 6, greatly enlarging the capacity of the latter and permitting the treatment of a lower grade of ore than formerly. It is expected that the changes will be completed by the middle of this year. The oxide concentrator treated 115,223 dry tons of ore and leached 102,381 dry tons of tailings. The copper recovered amounted to 11.3 per cent of the total copper produced during the year. The sulphuric acid plant produced 2971.5 tons of acid. In the smelting department an average of 3.8 furnaces were in blast, smelting ore and concentrate amounting to 158,532 tons. The average yield of copper from this material was 10.14 per cent. The company has been carrying on extensive experiments in metallurgical treatment, which will be of benefit in future changes or improvements in the plant. The output of copper for February, 1911, was 1430 tons.

The Evolution of Portland Cement Processes.*

By Charles F. McKenna, Ph.D.

In the Portland cement industry there is a perpetual but healthy unrest. No sooner is one feature of marked improvement successfully brought forward producing a suggestion of finality than several minor progessive steps are seen to tend together toward such economic advantages as to constitute practically again a new process. On reviewing the subject I have felt that the history has the elements of a romance, if it were not for the overpowering newness of it all, and I have felt that the near future still has surprises in store.

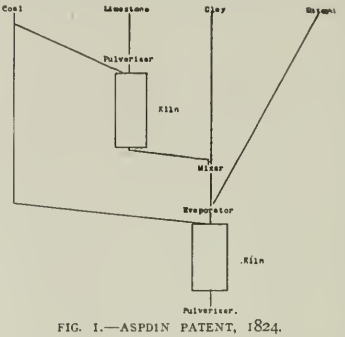


FIG. 1.—ASPDIN PATENT, 1824.

I hope I may leave this impression upon you, and also in rehearsing it to show properly that while praise is due to the mechanical engineer, it must also be given to the chemist. Indeed, it is generally felt to-day that it is to chemical engineering that the industry must appeal most strongly in the future, and that progress and economy is promised more surely because the leaders have realized that the trained and experienced chemical engineer is best fitted to cope with the new problems and avoid some errors similar to those of the past.

We have seen a product obtained in a most primitive fashion only 50 years ago now produced throughout the world in millions of tons annually, and with such economy that for 1 cent 12 lb. of raw material will be taken from the earth, dried, ground, calcined and vitrified, and the product ground again. It would seem as if the use of fuel, machinery, human hands and human brains could not go further. Yet there is ample scope for improvement and economies as you will easily perceive.

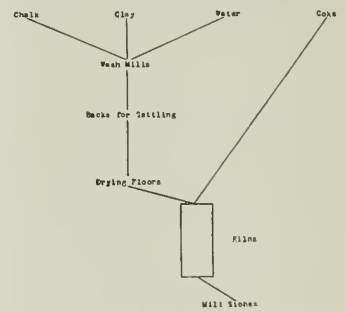


FIG. 2.—OLD ENGLISH PROCESS.

Aspdin's invention seemed a simple one (Fig. 1). He pulverized limestone, burned it, mixed clay with it and water to a plastic mass, after which this was dried and burned in a kiln and the product pulverized.

Much experimenting had previously been done by himself and by many engineers throughout the world to try to discover the cause for hydraulicity, but beyond perceiving the value of the argillaceous elements in the limestone, none of them discerned the important part played by high temperatures in the burning. Aspdin was the first to combine these elements properly, and burn much of it properly.

It is very probable that he did not know that the vitrified pieces of his clinker made the only true Portland cement, and it is also very probable that other experimenters had obtained vitrified Portland clinker when burning natural argillaceous

*Presidential address delivered before the American Institute of Chemical Engineers in December, 1910.

limestone and did not recognize its special value at the time.

The success of Aspdin's work was noted in other countrys. but for the first 25 years its greatest vogue was in England. Many modifications had in the mean time been introduced there and the typical English process of the day was one (Fig. 2) in which chalk and clay were introduced with an excess of water into wash mills—large circular basins with revolving harrows—and after incorporating here, transferred to large backs or settling reservoirs from which the surplus water was drawn off. It took about six weeks to settle.

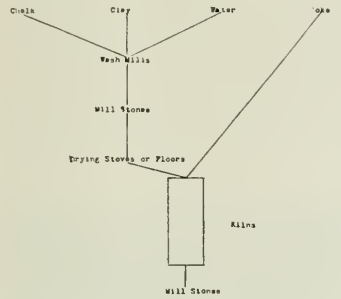


FIG. 3.—LATER ENGLISH AND CONTINENTAL PROCESS, SEMI-WET, AFTER 1870.

The stiff mud from these was dried on floors heated from below and then charged into the kilns in which coke or coke dust was the fuel. The product was ground by burr stones.

The juxtaposition of chalk deposits near the Thames and the Medway and the mud banks of these rivers, together with a plentiful supply of coke from the gas works of London caused a large development of this industry near the world's greatest city.

About the middle of the last century the industry was fairly started in Germany, and was destined there to take a development characteristic of that country, and in it the skill of the chemist was successfully given free scope.

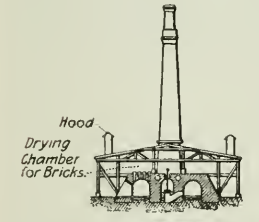


FIG. 4.—HOFFMAN CONTINUOUS KILN.

The favorite method ultimately developing in Europe (Fig. 3), sometimes called the semi-wet process, was one in which less water was used in the beginning, and in which air incorporating mill was used on the stiff mixture. This saved the time previously allowed for the very wet mixture to settle in the backs.

When you add good chemical control of the original proportions and this feature of incorporation by grinding you see how progress was being made. This is what the Germans particularly did.

The kilns first used were the ordinary lime kilns, later increased in height for the purpose of increasing the heat.

Later in some of the greatest of the German works the Hoffman continuous kiln (Fig. 4) was much used in which compartments are charged and discharged and the flat roof used for drying.

In Germany the Dietzsch became the favorite in later years. In this kiln (Fig. 5) the material is preheated in the upper part as usual, and in the central part into which it is drawn by the workmen operating through the fire door it is particularly exposed to high heat by reason of the construction of the furnace here. The clinker gradually cools as it sinks below.

The Schöfer or Aalborg kiln is an upright one like the Dietzsch and has a zone of high heat.

Now investigators in this country in the decade after the close of the Civil War noting that the Portland cement imported from Germany and England was selling for $5 a barrel sought eagerly for materials from which to make it.

It was thought that a soft chalk must be found before any economical works could be thought of. This was the view of all English cement makers who were visiting this country at one time with a view of starting business.

But David F. Saylor, who was making natural cement from one kind of rock at Siegfried's Bridge, near Allentown, Pa., began experimenting in the early seventies upon a mixture of this rock with a purer limestone. He produced in effect a true Portland cement (Fig. 6) and he was destined to be the first in

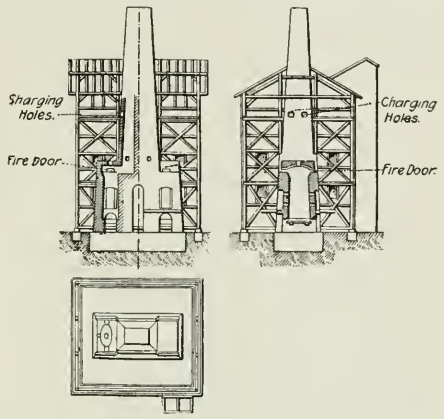


FIG. 5.—DIETZSCH KILN

a long line of American inventors, each of whom can be said to have modified the industry so fundamentally and markedly as to the world in each step forward.

The first kiln used to produce Portland cement in America was a bottle kiln of the type used in England. Subsequently at Coplay, Pa., the seat of the industry, the Schofer kiln was built.

Saylor's American Portland was the recognized and only important American Portland cement for a few years after this till Antonio Navarro, a capitalist, entered the business. Ransome had patented a rotary kiln in England in the seventies, but it was not practically adopted there. In the early eighties Navarro secured the American rights, and what is now the present Atlas Portland Cement Company was started at Northampton, Pa. Whittaker also used a similar kiln, and both of these used oil as a fuel.

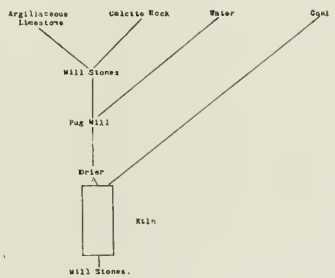


FIG. 6.—FIRST AMERICAN PROCESS. (SAYLOR. ABOUT 1875.)

This was entirely a dry process (Fig. 7). It is sometimes thought that in no country but America was it the practice in those days to eliminate the wash mill and to incoroporate dry. This is not so; for in Europe from 1885 to 1895 there were several plants operating on the process of: First, crushing and pulverizing dry; second, mixing and moistening with about 15 per cent. water; third, moulding into bricks; fourth, burning; fifth, grinding clinker.

In 1895 Hurry and Seaman were amongst the first to operate

in this country by a process which was destined to be of great influence the world over in the industry. This was the process for using as the fuel powdered coal blown into the kilns. The product was so well burned and so much in volume was continuously produced that the method rapidly extended. The clinker thus produced made a cement which was very quick setting. If a change of the composition of the raw materials was made to avoid this the cement was unsound, particularly upon trial by the quick method of being exposed to boiling water or steam. Then it became the custom to do here what had for some time been practiced in Europe, namely, to add plaster of Paris or fine ground gypsum to the extent of 1½ per cent. to 2 per cent. to the ground cement, which modified this phenomenon materially (Fig. 8) and produced a hydraulic substance of very desirable qualities.

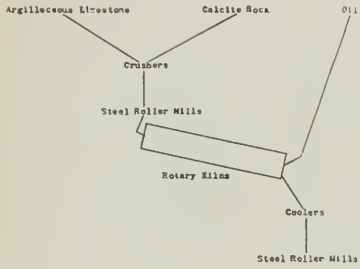


FIG. 7.—EARLY AMERICAN DRY PROCESS (ABOUT 1890.)

A little later, almost within the last twelve years, it was discovered that the raw materials must be very finely ground if you want to succeed in getting a perfect clinkering.

The wet method of the European works did good incorporating, and if you want that in the dry process and the quick-acting rotary kiln you must grind to an exceedingly great fineness. Also, if the cement is coarse it will hydrate slowly, with disastrous results of expansion. This was the heyday of the mechanical engineer.

These details of fine grinding had been neglected by the European practitioners of the art, but really explained their troubles. Their raw materials had been all right, because they had chalk and clay and it facilitated wet grinding. The incorporation was good, and the clinkering consequently was good, but their cement was frequently bad because it was coarse. Immediately a great variety of milling apparatus was invented and the American mechanical engineers worked hard upon this problem and with great success, and really the inventing and improving of this class of apparatus is still going on. The one kind of mill, modified, will serve for both raw and clinker grinding.

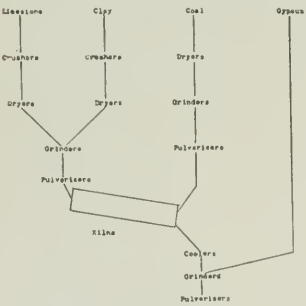


FIG. 8.—MODERN WORLD-WIDE PROCESS, DRY MATERIAL.

Two important classes of milling apparatus have been brought into use in this industry.

The development of flour milling when away from stone mills was towards steel roller mills, but in the cement industry the development of two important classes of comminuting apparatus resulted from the abandonment of stone mills. (It is true, stone mills are not yet entirely antiquated, for a new mill at Norfolk has just been equipped with everything of most modern form except the mills, which are burr-stones and which on experiment have proven to give satisfaction for the material operated on—marl—An interesting reversion!)

Of the new classes of mills referred to we have those of that type to which belong the Griffen, the Fuller and the Kent mills, in which revolving steel shoes or balls press against a steel die or channel. The Fuller, using balls, is highly efficient, particularly in producing a high percentage of the finest flour of cement.

The other type requires two separate installations, namely, a large drum lined with steel plates and partially filled with steel balls, to do the comminuting of rocks coming from the coarse grinders, and a long horizontal drum partly filled with large quartz balls, to do the pulverizing.

But the genius of Edison was now introduced upon the stage and in the first years of the century, after improving the large crushing machinery, he turned his attention to the improvement of the kilns. The prevailing length at this time was 60 ft., with an output of about 180 barrels per day and a

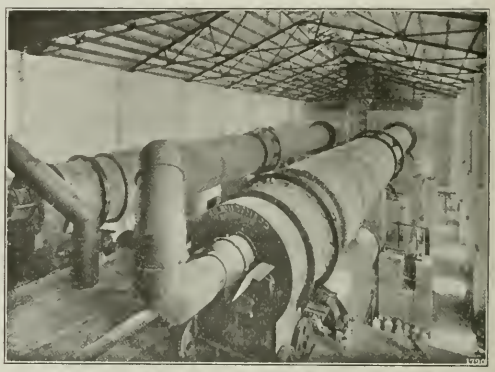


FIG. 8A.—ROTARY KILNS WITH ENLARGED BURNING ZONE AND COMPRESSED AIR FIRING AT THE WORKS OF NOERRESUNDBY PORTLAND CEMENT-FABRIK, NOERRESUNDBY, DENMARK.

coal consumption of 125 lb. per bbl. Some kilns 100 ft. long and 7 ft. diameter had been used in Europe on wet materials. Edison believed that if the kiln were 100 ft. long the product would be larger, the cement better and the coal consumption less. He has proven these points. But even with this economy the fuel consumption was three times the theoretical and twice what it should be practically. Immediately interest was centered upon the economics of fuel consumption in the Portland cement kiln.

It is admitted that the greatest avoidable loss is in the stack gases; next, the loss of radiation, and lastly, the heat carried off by the clinker. The last named has been most often attacked. Losses by radiation have been lately considered by inventors, but not yet successfully obviated, principally on account of the dilemma or choice between overheating the lining or overheating the shell. The losses in the stack gases have been diminished by careful attention to the details of burning, and lately it is believed by still further lengthening the kiln. One or two kilns here and some abroad, are said to be now operated experimentally on a length of 235 ft., more or less, but results are not divulged.

Strange to say, it is my belief that economies in coal, taken as a whole; that is, the coal for power and the coal for burning, will be improved by returning to the wet process. Before describing this I will speak of another process that marks an important epoch.

One of the greatest developments in Portland cement manufacture in this country has emanated from the steel works. Blast furnace slag, particularly when treated by water when hot, and then granulated, has many virtues for the aluminosilicate element in a hydraulic compound with lime. This was the old slag cement, but deserves no further mention in that

the composition and the reactions of that product are entirely different from Portland cement, belonging rather to the class puzzolana cement, with which Roman engineering accomplished much.

But investigators both here and abroad labored to produce clinker from a mixture of slag and limestone, and with the high heat of the rotary kiln they have been able to do it. It differs little from the standard process except that the slag is the clay element and is at once ready for pulverization, and except that Kent mills, together with air separators, are used for finishing. They have a thermal advantage, both in the fact that part of the lime has been calcined previous to their receiving it, and part of it is in favorable form of combination with the silica and alumina. Other advantages possessed by a steel works in making slag Portland cement are the credit for removing what is otherwise a costly nuisance and the important advantage of having blast furnace gas to utilize for the production of power.

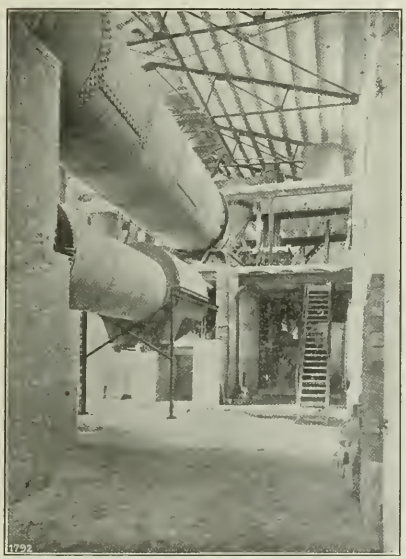


FIG. 8B.—ROTARY KILN OF NOERRESUNDBY PORTLAND CEMENT FABRIK, NOERRESUNDBY, DENMARK.

The Rotary Kilns are equipped with enlarged burning zone, compressed air firing and under coolers.

The manufacture of cement under these conditions is so favorable a proposition for the United States Steel Company that it is rapidly becoming the largest factor in the business. The environs of Chicago are producing almost as much cement as the whole of the Lehigh Valley.

Besides mixtures of clay and limestone, argillaceous limestone and pure limestone or calcite, slag and limestone, we have also as materials shale and limestone and clay and marl.

The shale and limestone introduce no complexity or important change from the type modern process except in the drying and grinding.

The marl, or what might be called *bog calcite*, has always been a different proposition. One of the important events which I have not previously mentioined, in the history of American Portland cement industry was the work of Millen and others in Central New York State, where such marls are plentiful. They struggled in the same way as Saylor and at the same time, and really manufactured on a commercial scale, but one by one succumbed to the competition of the Lehigh Vallley material. The latest to close, the Empire at Warners, N. Y., was a very important factor for some years, producing a good cement in rotary kilns, but under severe economic difficulties. Michigan suffered most from the false hopes held out of profit from her marl beds.

Now, it would seem strange to admit that wet marl costs too much to dry or to put in the kiln in a slurry, and then to claim that grinding into a semi-wet mix hard and almost dry rock and clay, or rock and shale, is a good economical proposition. Yet the truth of this is coming to be recognized.

A company operating in one of the Eastern States built a wet plant for limestone and clay quite some years ago, and although their handicaps were great they have succeeded by

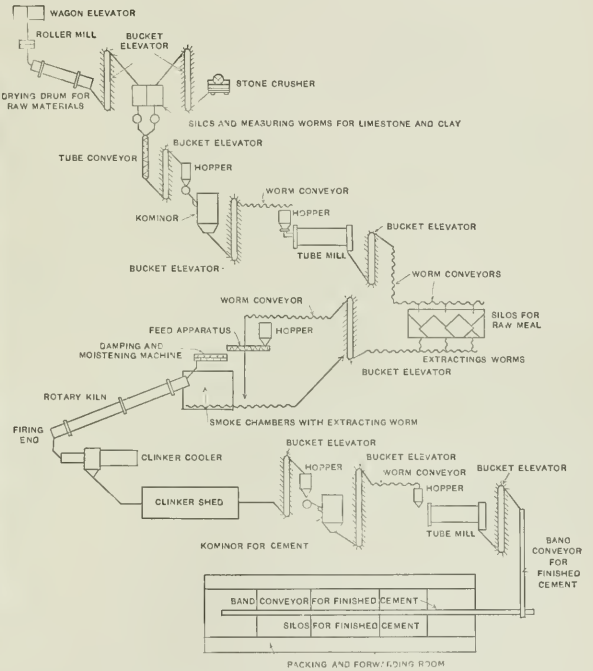


FIG. 9.—DIAGRAM OF MODERN CEMENT MANUFACTURE.

persistent effort at improvement and are about to obtain great aid from the introduction of the long kiln.

American cement engineering advanced beyond Europeon ten years ago—to-day European practice is in advance, and it is on account largely of improving in details on the American ways that most is promised from current practice in the wet way.

The explanation is this: The physical character of marl prevents the easy removal of water and also causes a slurry to carry too much water—almost 60 per cent.

The reason why wet processes failed was because no account was taken of this excess water even in the so-callled semi-wet slurry. When it was realized that this proportion of water should be cut down it was found that slurry with 40 per cent water could be pumped. But it was also found that a great deal depends upon the physical character of the raw constituents, and there are some, notably from hard materials, as limestone and shale, wherein 35 per cent to 32 per cent and even a little less may yet permit the free movement and carriage of the slurry. Great attention must be paid to

the fineness to succeed in attaining this minimum water. Thus you see the consumption of 200 lb. of coal formerly recorded against the rotary kiln in the wet method must be materially reduced. Then adopting the advantages of the long kiln for full utilization of heat and for large capacity whereby you also avoid coal for the dryer you come to a situation where a

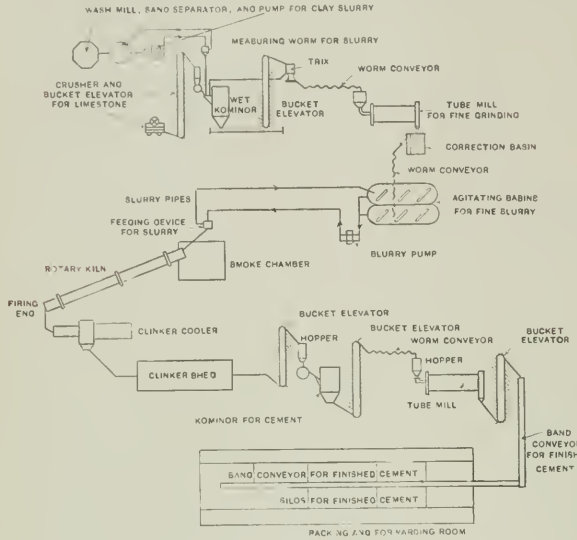


FIG. 10.—DIAGRAM OF MODERN CEMENT MANUFACTURE.

wet process for hard materials is not only feasible but highly desirable.

In most modern European mills there is provision for drying in rotary driers between the kiln and the stack and also for saving clinker heat by under coolers taking the air for combustion first over the clinker and closing up air leaks in the front of the kiln. (Figs. 8a and 8b.)

Then we have other factors which are important, and in these I believe a well-designed wet process introduces promise of saving. The first is raw grinding. With greater capacity of mills for equal horse-power we shall have made a gain.

It was admitted, when marl or chalk or clay were used, that the wet grinding was done at an advantage. It is so with the hard stone: after crushing, the pulverizing wet will be done more economically. In an experiment with the pebble mill of F. H. Smidth & Company the following results were obtained:

Fineness.

Residue on Sieve No. 180

13.0 per cent

8.0 per cent

Product.

Wet Grinding.	Dry Grinding.
Pebble weight	73 per cent of pebble weight
70 per cent of pebble weight	54 per cent of pebble weight

Secondly, it is safe to say that the fuel requirement in the kiln will be less by reason of the intimate contact of the particles of stone and clay—the rearrangement of the molecules in the clinkering stage will require less heat. This clinker also grinds more easily than that from less well compounded dry materials requiring high vitrification. Then there is no drier installation and no coal to be burned for it. There is no dust at the driers. The dust nuisance, in fact, is more than half eliminated in a wet mill. This is of considerable importance in reducing losses to machinery and the actual loss of the dust itself which carries with it the cost of royalty, quarry, drier and grinder and the nuisance to the neighborhood. The dusty appearance of the air around a group of cement mills is a joy to all interested in the industry as a sign of prosperity, but it is not pretty. We ought to pay dividends in esthetic currency whenever we can spare some—for outsiders.

This process has been placed on an economical basis by the advances made in the design and construction of rotary kilns which have been brought to such a point of perfection that it is possible to have wet mixtures containing between 30 per cent and 35 per cent of moisture with the same economy and to obtain the *same capacity* as can be had from modern rotary kilns on the dry process.

The following is the record of the performance of a kiln in a modern mill in Russia working the wet process on a slurry of 38 per cent water made from raw materials:

The kiln was 150 ft. long with enlarged burning zone from 6.9 ft. in diameter to 7.9 ft. in diameter. In 26 working days it produced 14,500 barrels of 380 lb. each, or about 560 barrels per day, with a coal consumption of 24.2 per cent, or about 92 lb. per barrel. This coal contained 1.42 per cent water, and on the dry basis had a heat value of 11,500 b.t.u.

Much advance is being made in the study of the heat economics of the kiln, and the researches of Newberry, Meade, Richardson and Landis are carrying us forward by placing more accurate data before us.

In good present dry practice, in kilns 135 ft. to 150 ft. long, we are producing 750 barrels of cement daily with a kiln coal

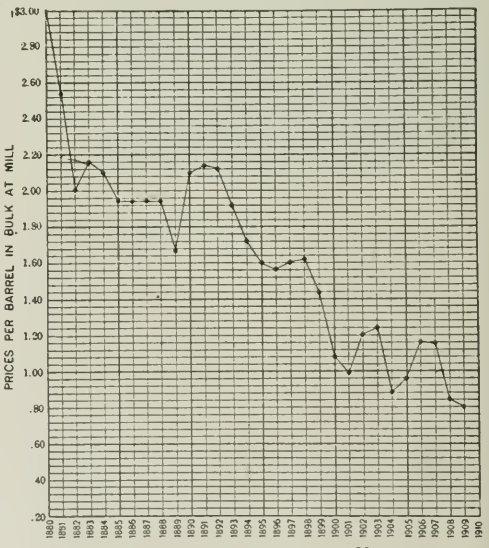


FIG. 11.—PRICE CURVE OF CEMENT FROM 1880 TO 1909.

consumption of 85 lb. The product is ground to 80 per cent fineness on the No. 200 sieve. Some mills are probably doing this at a mill cost of 60 cents. I have seen figures of some which indicate that 50 cents has been reached in the best

situated and best engineered plants, and that the given good installation could do this in a given good situation for coal and raw material. But the coal and the power draft are still the items promising improvement.

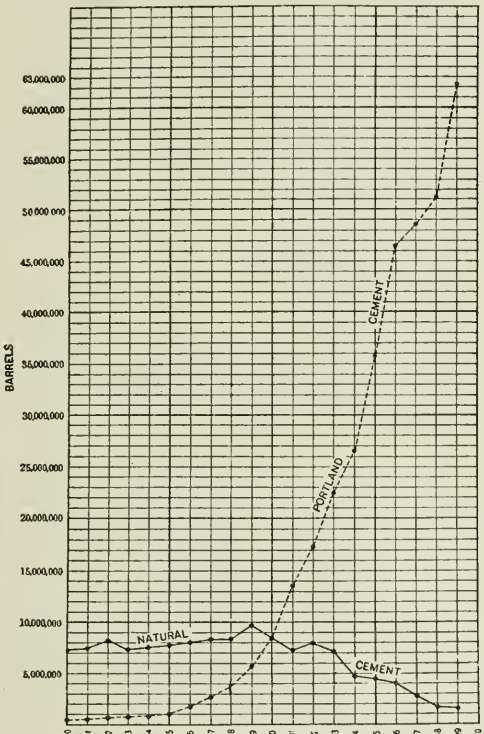


FIG. 12.—PRODUCTION CURVE OF PORTLAND AND NATURAL CEMENT.

As said before, power should be saved by wet grinding; it should be saved in grinding so-called wet-process clinker. Again, there is something promised in the way of a new discovery in pulverizing which will increase mill capacity very materially, it is claimed. This is the substitution in place of part of the usual charge of pebbles in the discharge end of the tube mill of some hard metal cylinders which, being separated from the pebbles by a screen, complete the pulverizing more perfectly and with less power.

The saving of heat from the clinker will improve in details of apparatus and considerably in efficiency. Professor Landis says that 10 lb. of coal are lost by heat of clinker, and that from 5 lb. to 7 lb. can be saved by efficient undercoolers. He has also made the important discovery that much more heat than we had suposed is generated by the formation of clinker. It is so large that in a theoretically perfect condition, with no losses, the reactions of the kiln would require only 15.7 lb. per barrel. With reasonable losses he thinks it ought to be run with four times this amount and maybe some day in some places with something like threefold that amount instead of sixfold, as we most often have it now.

Some are advocating dividing up the process again so that one kiln—perhaps the upright, which is so favorable for it in fuel economy—will be used for calcining and the more highly heated rotary for clinkering.

I have known this to succeed at one plant where the works were run economically on set kilns and rotaries. This was because the company had an excellent plant of old kilnsidle, and at a place where anthracite culm was cheap. The anterior burning of a mix of high and low limestone permitted them also to introduce the correction before entering the rotary.

I have said little of the power house and power transmission in speaking of these processes, but they have also had their natural stages of rapid evolution and improvement, as you all know. Electric driving, so much feared at first in the cement mill, has come to be universal.

By courtesy of Messrs. F. L. Smidth & Company, of Copenhagen and New York, I append sketch outlines (Figs. 9 and 10) which show a modern wet and a modern dry installation also which will give an excellent idea of the mechanical equipment and installation. It needs only the conception of the power house to complete the picture of modern cement manufacture in outline.

It is interesting now to notice from the statistics of the industry the influence of this evolutional development upon the prices of Portland cement.

As pointed out by Eckel, the important epochs in the history are plainly marked in the downward curve of price (Fig. 11), and the constant improvement in the economy of production is suggested in the form of the upward curve of production (Fig. 12).

It would be a very impressive reversion if a new drop in cost would be the result of studies of engineers leading to the use of water and a return to the wash-mill.

Metallurgical Notes.

Filter leaves are usually cleaned by soaking in a 10 per cent HCl solution, but this operation can be accomplished with a much weaker solution—2 or 3 per cent—if the solution is hot and delivered under a slight pressure into the center of the leaves. By forcing the hot solution through the pores of the cloth the lime is quickly dissolved.

Canvas slime tables have been built in various forms for the purpose of saving fine mineral escaping from concentrating works. One of the latest types is that devised by Mr. E. C. Porter at the Arizona Copper Company's concentrator at Morenci, Ariz. It consists of a dozen canvas tables arranged one above the other. The slime feed is allowed to run onto the tables until a coating of slime is formed, when the feed is shut off and a stream of water applied simultaneously to each deck by a traveling water pipe with suitable extensions over the several decks. When the accumulated slime is washed off the operation is repeated. The adaptation saves floor space and reduces the cost of attendance.

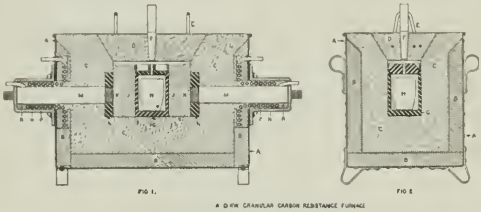
Grading Analyses.—In his reply to the critics of his paper on "Grading Analyses and Their Application," published in the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, Mr. H. Stadler states that an account of the laws of ore crushing and the experiments executed in that connection may be found in a work by F. Kick, entitled "Das Gesetz der proportionalen Widerstände und seine Anwendung," Leipzig, 1885. In further answer to those who object to the application of Kick's law in determining the mechanical efficiency of crushing machines, Mr. Stadler points out that, while perfectly homogeneous materials do not exist, nevertheless the definition of "equal technological state" is broad enough to include irregularities of the material so long as they are similarly distributed so that the average physical peculiarities of particles in different stages are identical. If these irregularities are, on the whole, similarly distributed, the calculations are not affected. He believes that Rand ore complies unusually well with the term "equal technological state" for the reason that it is composed of quartz pebbles of all sizes, embodied in a matrix which gives the same resistance to grinding as the pebbles

Granular Carbon Resistance Furnace.

By J. H. Goodwin.

Several workers have described laboratory furnaces using granular carbon resistors. O. P. Watts[1] suggested embedding in the granular carbon a graphite container for a working or removable crucible. The furnace here shown makes use of that scheme and it has some other details of construction that may be of interest.

As shown in section in Figs. 1 and 2 the furnace consists of a riveted sheet-steel shell, *A*, 17½-in. x 12¾-in. x 10⅝-in x No. 14 B. & S. gage, with feet and lugs of 1-in. x ⅛-in. band iron. Two 30-in. lengths of ¾-in. pipe slipped through the latter serve as convenient handles when it is desired to move the furnace. The shell is lined with magnesia-asbestos plates, *B*, 1 5/16 in. thick, inside of which is a thicker refractory lining, *C*, made by pouring in place a mortar composed of 8 parts of carborundum fire sand and 1 part of portland cement. The heating space is 7 in. x 5 in. x 3 in. and it is covered by a



FIGS. 1 AND 2.—DESIGN OF 10-KW GRANULAR CARBON RESISTANCE FURNACE.

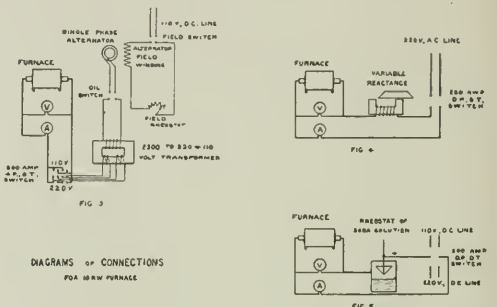
tapered block, *D*, of the same composition as the lining, containing an iron rod, *E*, for reinforcement and lifting and a hole, closed by a taper graphite plug, *F*, for watching through a cobalt glass the charge in the inner crucible. A 4¾-in. x 3-in. crucible, *G*, and cover, turned from Acheson graphite, occupy the center of the heating space and within this covered crucible a smaller graphite, magnesia or alumina crucible, *H*, containing the substance to be heated, is placed.

The electrodes are also of Acheson graphite. The plates, *L*, forming the ends of the heating space, are 4 in. x 3 in. x ¾ in. in size and they are firmly screwed on the 8¾-in. x 1½-in. rods, *M*, which fit loosely into the cylindrical part of spirals, *N*, of 5/16-in. copper tubing. The spirals in turn fit loosely into the brass terminals, *P*, attached with insulated bolts to the ends of the sheet-steel furnace shell. Good electrical connection between the brass terminals and graphite rods is obtained by sweating the latter in place with soft solder, *R*, which is kept cool during furnace runs by the circulation of water through the copper spirals connected by rubber tubing to the water supply and drain. Within the furnace shell the copper tubing is coiled in flat spirals in order to cool and protect the bolts and micanite insulation.

The resistor is composed of tightly tamped graphite, granular carbon or coke, *K*, ground to pass a 20-mesh sieve, and loosely added charcoal, *J*, crushed and collected between 4-mesh and 8-mesh sieves. For the first run some of the fine coke or carbon must be added to the charcoal, but after one heating this can be sifted out and the charcoal will then be of reduced resistance and the furnace can be started from 220 volts without any addition of carbon to the charcoal portion of the resistor.

With this type of furnace no electrode adjustment is possible, and, as the furnace resistance falls rapidly with increasing temperature a wide range of voltage control is essential. This can be obtained by a water rheostat, using sodium carbonate or hydrate to give sufficient conductivity, as shown in the diagram of circuits in Fig. 5, but such a method is every wasteful of energy and should not be employed if the furnace is to be used frequently. It is better to operate the furnace by alternating current and control the voltage by a variable reactance, as shown in Fig. 4, or, better still, to operate the furnace from a separate alternator of which one has the field control, as shown in Fig. 3.

The refractory lining of the furnace deteriorates and requires repairing from time to time. The resistor, outer crucible and cover burn slowly while the furnace is open to change inner crucibles, but making this time as short as possible a long life for these parts can be assured and they are easily replenished or replaced. The electrodes are not so easily replaced, but they do not need to be because they show very little deterioration, being always covered by carbon, which protects them from oxidation.



FIGS. 3, 4 AND 5.—DIAGRAMS OF CONNECTIONS FOR 10-KW FURNACE.

The furnace may be heated rapidly without injury. In one run, with a sample of platinum in the inner crucible, it was started cold with 200 amp at 200 volts. In three minutes the current had become 350 amp at 90 volts and the switch was thrown over to the 110-volt connection. Four minutes later the current was 250 amp at 70 volts and the sample of platinum was melted. Ten kw will maintain this temperature and crucibles may be heated and removed for pouring without interrupting the current or disturbing the outer crucible. The terminal cooling water carries off about 5 kw and the efficiency of the furnace is low, but for laboratory work this is compensated by the advantage of a durable, portable, compact, high-temperature unit that is not dismantled after each heat.

Fremont, Ohio.

[1] Electrochem. and Met. Ind., IV, 273.

Bakelite as an Insulator in Influence Machines.—Interesting information on the possibilities of bakelite and bakelized paper as an electric insulator is found in a recent German patent of Dr. H. Wommelsdorf for improvements in influence machines. In the past the discs, posts and other insulating portions of influence machines have always been made of glass or hard rubber. Both these materials have distinct disadvantages, the glass being breakable and the hard rubber becoming inefficient in time through electrical and chemical action. Dr. Wommelsdorf has now discovered that bakelite is suitable in a surprisingly effective manner for the manufacture of those parts of influence machines which are subject to the effect of the sparks and silent discharges and to the chemical action of the mixture of ozone and nitrogen oxide formed during the operation of the machine. Bakelite is very resistive against these influences. Moreover, by comparative tests the unexpected fact was established that influence machines, the inducing parts of which (for instance, disks) are made of this material, give an appreciably greater and almost double the quantity of electricity (coulombs) than influence machines of the same size and the same design with hard rubber disks. Bakelized paste-board is also suitable.

Heat Insulation of Furnace Walls.

By Carl Hering.

Although the fundamental principles concerning the losses of heat through the walls of furnaces are simple, yet it seems there still exists some doubt and uncertainty concerning rather elementary features, which in turn may lead to confusion, misleading statements and possibly even to positive errors.

The purpose of the present article is to endeavor to explain these general principles briefly from the standpoint of the practical engineer rather than from that of the academician, and to show how simple most of them are; also to point out some results and deductions which may be of practical value.

The analogy of a flow of heat through the walls of a furnace to that of a flow of electricity (a current) through a similarly shaped conductor are in some respects very close, and anyone familiar with the latter should not have any difficulty to understand the former. For instance, for the purposes of analogies only, suppose the inside of a cold furnace to be filled with mercury and the outside placed in a bath of mercury, all openings, of course, being closed; then by passing an electrical current from the metal in the inside to that on the outside, and measuring the voltage between the inside and outside, the electrical resistance of the walls as a whole could thereby be measured, supposing them to be sufficiently conducting. Or, knowing this resistance and the voltage, the current could be determined from them.

It will at once be seen that for any given dimensions and material of the walls, the greater the voltage the greater the current and vice versa. Hence, the voltage in that case is a direct measure of the current which leaks through. If, however, there are several furnaces, all of like dimensions but made of different materials, then the voltages for the different furnaces are no indication of the relative leakages of current, and to assume it to be so might lead one to very erroneous conclusions.

If, for instance, to take exaggerated cases in which the results require no refined calculations, the walls of three furnaces of like dimensions were made of copper, carbon and moist brick respectively, then with the same voltage for all three, the current would be enormously greater for the copper than for the carbon, and much greater for the carbon than for the moist brick. The voltage, being the same in all three, is, therefore, no indication whatsoever of the relative currents.

Now let the voltage be higher for the copper than for the carbon, the greater current will then correspond with the *greater* voltage. If, on the other hand, the voltage is greater, though not unreasonably so, for the brick than for the carbon, the greater current (namely, that in the carbon) will now correspond with the *lower* voltage; that is, just the reverse of the previous case. Hence, again, the current does not correspond with the voltage.

The same is true if a low voltage is applied to the copper and a somewhat higher one to the carbon, that is, just the reverse of the above case, yet the current for the copper would then still be the greater notwithstanding the fact that the voltage is now less. Finally, let the voltage for the brick be less than that for the carbon, the current through the brick will then still be less than for the carbon, although the relative amounts of the voltages are now the reverse of what they were above.

The conclusions from these exaggerated cases are so evident that no numerical calculations are necessary to further demonstrate them. In less exaggerated cases the difference is only one of degree, although specific cases might be selected in which the voltage might be made to appear to be a measure of the current, but if this were true as a general law, it would, of course, have to apply to all cases and it has been shown above that it does not.

The explanation, of course, is that the current is equal to the voltage divided by the resistance, hence the current depends quite as much on the resistance (hence on the material) as it does on the voltage; it is only when the resistance remains the same (namely, when the material and dimensions remain the same) that the voltage is a measure of the current.

This analogy to thermal flows is virtually a perfect one, as far as the subject now under consideration is concerned, namely, the insulation of the walls considered by themselves. The difference in temperature between the inside and outside surfaces is analogous to the difference in electrical potential, namely, the voltage; the thermal resistance (insulation) is analogous to the electrical resistance, and the flow of heat, or heat loss through the walls, is analogous to the flow of electrical current.

Hence by analogy, for any particular furnace (with fixed inside temperature), the higher the outside temperature (hence the less the drop), the less the loss, and vice versa. But in comparing furnaces made of different materials, whether of the same size or not, the outside temperature (assuming the inside temperature to be the same in all) is, as far as the walls by themselves are concerned, absolutely no indication whatsoever of the loss of heat—that is, of the insulation efficiency. The losses may be very different when the temperatures are the same, or they may be greater when the temperature is less, or less when it is greater. The loss depends not only on the temperature, but also on the total thermal resistance of the walls, and this in turn depends not only on the thermal conductivities, but also on the dimensions. But even for the same size, the outside temperature is no indication of the loss.

According to the thermal Ohm's law the flow in watts W is equal to the drop in temperature T in centigrade degrees divided by the thermal resistance R in thermal ohms.* Or $W = T/R$. Hence $T = WR$; that is, the drop of temperature in the walls is equal to the *product* of the flow and the resistance, but is not proportional to either alone unless the other is constant. If neither the flow nor the resistance is known, then the temperature is no indication of either, as for the *same* temperature in two cases W may be larger and R small, or the reverse; that is, the same temperature may be due in one case to a great flow through a small resistance and in another to a small flow through a great resistance.

Several numerical examples were given by the writer on page 125 of the March issue of this journal, hence they need not be repeated here.

The above applies to the insulation afforded by the furnace walls considered by themselves, as the heat insulator of the furnace, and as this is in almost all cases one of the two chief purposes of the walls it applies to the general case, and the principles enunciated above may be said to be the chief fundamental ones concerning the relations between the flow, the thermal resistance and the drop of temperature. It is, of course, assumed herein that all the heat reaching the outside surface is freely dissipated as fast as it gets there and without any obstruction to such dissipation; the effect of such possible obstruction will be considered later.

Besides these fundamental considerations there are some minor or less important ones, just as is the case in most other problems, and they modify the general results somewhat. They are of more interest to the academician than to the practicing engineer. To include them in calculations may be said to be a refinement, or a second approximation, as it is often called; but in most furnace calculations which the engineer has to make, and which are necessarily somewhat crude in the present state of our knowledge or lack of knowledge of the physical properties of the materials involved, these minor features may generally be neglected or allowed for as correction factors instead of attaching the same importance to them as to the fundamental ones, whereby the problems would often be hopelessly complicated. Among these may be mentioned the

*For the definition of a thermal ohm see the article on this subject in the January issue of this Journal, p. 13. (Article on thermal resistance and conductance; the therman ohm and thermal mho.)

temperature coefficient of the thermal resistance, and the further variation of this coefficient with the temperature. This will undoubtedly effect the results somewhat, from the academician's standpoint, and will afford him the means of making the formulas and analyses very complicated, while in their unburdened or fundamental form they are very simple and easily understood. Moreover, as our knowledge of the numerical values of the resistivities or conductivities themselves is to-day still in a somewhat crude state it seems unreasonable to lay much stress at present on any refinements of these values. If the engineer has given to him reasonably reliable values for the usual materials when hot and when cold, it will suffice for most of his purposes and it will generally not be necessary for him to complicate the analytical work with the introduction of temperature coefficients.

Another one of these less important features, which, in the opinion of the writer, are best considered as secondary ones, to be introduced as corrections rather than being given the importance of a fundamental feature—is the rate of dissipation of heat from the outside of the furnace to the air or water jacket when this dissipation is restricted as distinguished from being virtually free and unlimited, as it would be with a good water jacket, and as it was assumed to be in discussing the fundamental relations above. If the outside surface is large enough or sufficiently exposed to draughts or water circulation to dissipate all the escaping heat freely so as not to bank it up, then the resistance which the flow of heat meets at this surface may be neglected; or it may be roughly allowed for on the basis that for a given kind of surface and a fixed room or cooling water temperature the greater the heat dissipated per square foot the higher will be the temperature which that surface will assume. With restricted dissipation that temperature will, of course, be higher, but in either case the rise will not be nearly as fast as the increase of the heat dissipated per square foot; according to Stephan's law the radiation varies with the fourth power of the temperature (absolute scale). hence to dissipate a considerably greater amount of heat by radiation would involve only a relatively small rise in temperature of the surface.

While, therefore, it is undoubtedly a fact that theoretically the temperature of a given surface will increase when the amount of energy dissipated from it increases, yet the obstruction to the heat flow which causes this appears to be too small relatively to the resistance of the walls to be given much importance; the wall insulation is the important one, the other is of only secondary importance. If, for instance, the fall of temperature from the inside to the outside of the wall is a drop from 1800° to 118° (see example below), namely, 1682°, and that from the outside (118°) to the air at 20° is 98°, the walls will, as an insulator, be more than 18 times as important as the superficial resistance to dissipation; the latter is in this case only 5.5 per cent of the total effective resistance, and it is, therefore, certainly of only secondary importance

In this connection, however, attention might be called to the advantages of treating or covering the outside surface of the walls to obstruct and hinder as much as possible the dissipation of heat from it, as could be done for low temperatures by a thin layer of cork, for instance. This would increase the insulation efficiency, or for the same efficiency, would reduce the thickness of the wall required for insulation; the hotter the outside of the wall of a given furnace, the less the heat loss through that wall. But in that case this outside coating must, of course, be able to withstand that higher temperature when exposed as it is to the free oxidizing influence of the hot air

If it were possible to find some coating for the outside of a furnace which would resist the dissipation of heat to so great an extent as to form a sufficient insulation of the furnace, then the insulation afforded by the walls might be neglected and the walls made as thin as other considerations would permit. In that case the lower the temperature just outside of this wall, the less the heat loss. But this is, apparently, more of theoretical than of practical interest, as at present the chief insulation is afforded by the body of the furnace wall.

Theoretically the temperature which the outside surface of a furnace will naturally assume when the dissipation from that surface is appreciably restricted, that is, when it is not free and unlimited, is that at which the heat flow to which the dissipation from that surface is restricted, is exactly equal to the flow reaching it through the walls; this is the stable state and the temperature will rise or fall until that state is reached. For a given furnace with a given inside temperature and with restricted dissipation from the outside, the higher the temperature of the outside surface the less will be the flow through the walls, but the greater will be the dissipation of heat from the surface; hence, as the one diminishes while the other increases, the stable state is reached more quickly and the changes of temperature corresponding to changes in flow are necessarily less than they would be for either alone or if both changes were in the same direction.

As was shown above, the resistance due to this restricted superficial dissipation is of relatively small importance compared with that offered by the walls, and the temperature rise which it produces is, therefore, probably of more interest to the academician than to the engineer. Moreover, that so-called stable temperature is so greatly dependent upon the surrounding temperature, hence on the weather and the seasons, as well as on draughts, the temperature of nearby objects, the presence or absence of nearby walls or other objects, etc., that no great importance can be attached to it.

As a recent statement, however, attaches sufficient importance to this outside temperature to recommend it as a "very good" means for measuring the losses of heat through different furnaces and for comparing the losses and the efficiency of the material of the walls, the writer has made the following approximate analysis of the relations, not because he considers it of much importance, but because the question was raised.

To treat it analytically involves very troublesome equations of the fourth degree. Moreover, the laws concerning the convection of heat from surfaces are complicated and involved if at all definite. But to solve this problem in this way one would express the heat flow through the walls algebraically in terms of the unknown outside temperature, then express that dissipated from the outside by radiation and convection in terms of this outside temperature also. The flow is, of course, the same in both and so is the outside temperature. Hence this gives two equations with two unknowns, and they can, therefore, be solved for both. The result seems to be of too little interest to warant the labor.

The engineer is usually satisfied when he knows that such a result is not of much importance, and this can be shown by means of a simple graphical method applied to the following actual numerical cases.

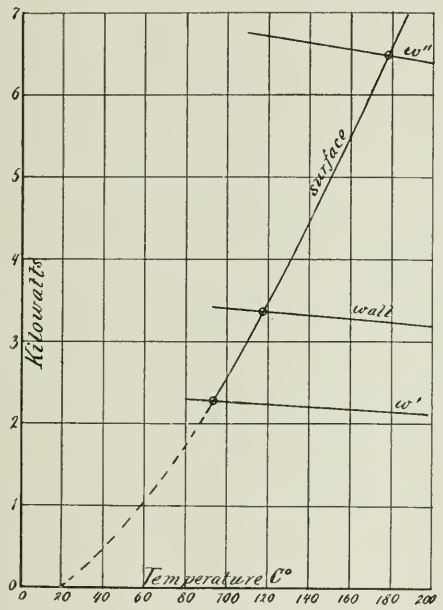
The following dimensions and data have been chosen to simplify the calculations rather than to serve as an example of a good design; also chiefly in order to adapt them to the small range of the available data, given in the very convenient curves of Mr. Snyder, for the sum of the radiation and convection losses from the surface of bricks to the air at various temperatures, as given in the *Transactions* of the American Electrochemical Society, Vol. 18, 1910, page 239. The writer here assumes their correctness, as he has not confirmed them.

Assume a furnace of silica brick; inside dimensions 6 in. x 6 in. x 6 in.; thickness of walls, 9 in.; hence outside dimension 24 in. x 24 in. x 24 in.; inside temperature, 1800° C.; room temperature, 20° C.; thermal resistivity of silica brick, about 48 thermal ohms. Hence inside surface is 216 sq. in.; outside, 3460 sq. in. = 24 sq. ft.; the mean (geometric) cross-section perpendicular to the flow is the square root of $216 \times 3460 = 866$ sq. in.; hence the total thermal resistance is $48 \times 9 \div 866 = 0.50$ thermal ohm. The graphical solution consists in calculating by means of the well-known formula

watts = drop in temperature ÷ resistance in thermal ohms.

the flow through the walls for each of several assumed temperatures of the outside of the walls; then plotting them as shown in the adjoining figure and drawing a curve through them, this curve is marked "walls," representing the part which the walls take in determining the total loss. It will be seen that this line is nearly horizontal, showing that the loss is nearly the same for all of these outside temperatures, hence nearly independent of them. Under such circumstances it would be very poor practice to measure the flow by means of these small differences of temperature.

Then calculate for each of those same assumed temperatures of the outside of the walls, the flow of heat in kilowatts which will be dissipated from the 24 sq. ft. of outside surface by multiplying that surface by the constants for each of those temperatures, obtained from Mr. Snyder's curves, on the assumption that they are virtually the same for silica brick as for firebrick. The results when plotted are shown in the curve



HEAT CONDUCTION THROUGH FURNACE WALLS.

marked "surface." It will be noticed that this curve is very steep, which means that this temperature varies only slightly for a very large variation of the flow. That is, the flows are not greatly dependent upon those temperatures. Hence for this reason also it would not be good practice to measure the former by means of the latter.

It is evident that the stable state will be reached when the temperatures for each of the two curves are equal, hence the desired value is at the point of intersection of the two curves at which, of course, the flows are also equal. Hence in this case the outside stable temperature will be about 118° C. and the loss of heat about 3.35 kw.

As remarked above, this temperature is a very unreliable figure as it depends on the weather, draughts, etc., and, in the opinion of the writer, would for that reason alone be unfit to be taken as even an approximate measure of the relative losses or of the relative insulating values of different wall materials. But to show that it would be a poor figure to use as a measure for comparing different wall materials, even under very carefully maintained laboratory conditions, hence even for the academician, two more curves have been calculated, one marked w' for a material having a resistance 50 per cent greater, namely, 0.75 thermal ohm, hence probably greater than any suitable material would have, and the other, w'', for one having a resistance 50 per cent less, namely, 0.25 ohm, hence half as great as the original one and extremely low. These three examples, therefore, cover very wide ranges.

It will be seen that the stable temperature for w' will be about 94°, hence only 24° less, although the difference between the two materials as insulators is quite great, namely, 50 per cent, and although the relative losses are reduced from 3.35 kw to 2.3 kw, hence also quite a great difference.

The former loss is very nearly 50 per cent greater than the latter, which shows that the total loss is determined almost entirely by the resistivity of the wall material and that the slight change in outside temperature is of practically no importance in determining the flow or loss and may, therefore, safely be neglected.

The results for the curve w'' are similar. The stable temperature then will be 178°, or only 60° higher, although the material is only half as good an insulator as the original. The flow, 6.5 kw, is very nearly double the original, showing again that the resistance of the wall material is decidedly the governing factor for determining the flow and that the outside temperature may safely be neglected, except that one must, of course, expect a somewhat higher outside temperature when the flow is considerably greater.

This analysis shows that the curve marked "surface" is very steep as compared with the others, which means that as far as the surface dissipation is concerned only a few degrees rise (horizontal distances) will cause the surface to dissipate very much more heat (vertical distances). Hence to determine the flow by means of this temperature is like determining the movement of the long end of a lever by a crude measurement made at the short end; a very small error in the measurement will give rise to a very large one in the result that one is striving to get.

To measure such temperatures accurately on the outside wall of a furnace and to expect to get concordant results would, in the writer's opinion, be a hopeless task; they would probably differ by 50 per cent on different parts of the same furnace at the same moment. And even if a mean of such very unequal readings had any value as such, it would be of little use because we know too little about the constants for the dissipation of heat from surfaces, and, therefore, could not intelligently interpret the temperature in terms of resistances or losses even if the values of the temperature were accurate.

There is still another feature involved in this question of heat insulation which may also not be clearly understood and which is of greater importance. It bears on what might be called "over insulation." In this feature the electrical analogy misleads one instead of helping one.

Regret has often been expressed that nature has given us no practically perfect heat insulator (other than a vacuum), such as rubber is as an electrical insulator. This is not so serious, however, because in the very cases in which it would be most desirable, namely, for high-temperature furnaces, such an insulator could not be used even if it existed.

Suppose, for instance, a furnace was surrounded by a vacuum, which is the most perfect heat insulator known. A vacuum insulates only against convection of heat, but not against radiation, as we know, because we get our heat from the sun through a vacuum: but in the present argument the radiated heat is considered zero by the assumption of a perfect insulator.

In such a furnace the heat will at first flow toward the outside; as it meets with the vacuum it can go no further, hence will accumulate and it will continue to do so until the outside is at the same temperature as the inside. This, therefore, means that the walls of the furnace will throughout their whole thickness reach exactly the same temperature as the inside of the

furnace, and for high-temperature furnaces this means that the whole furnace would soften and melt. Hence a perfect insulator could not be used even if it existed. Some flow of heat must, therefore, be allowed for in order to keep the walls from melting; the problem is to find the least possible amount which will still do this. This much-desired perfect heat insulator around the cylinder of a gas engine, for instance, would be fatal to the cylinder.

Again it might be argued, why not surround the outside of a furnace with cotton, mineral wool, very fine magnesia or some similar material which has a very high thermal resistance due to its holding air in its pores so that the air cannot move, still air being one of the best insulators.

The same reasoning as above will show that in such a case the outside of the bricks, where the outflow of heat nearly ceases, would soon become very highly heated and unless they and the outside porous insulator can stand this high temperature the furnace would fail.

It has no doubt often happened that parts of furnaces have thus been melted away due to over insulation, when such melting might have been prevented by allowing a greater heat flow to pass through those parts by insulating them less. Too great an economy of heat loss may result in other more serious losses; a good thing may be overdone; the loss of a furnace wall may be much more expensive than the loss of some heat.

One of the cheapest and best insulators seems to be a joint. Hence, for good insulation the bricks should, if practicable, be laid dry and so that their flat sides are perpendicular to the flow of heat, as this gives the maximum number of joints in the path of the flow. But, as described above, this might result in the melting of the inside layer of bricks, hence it should be done cautiously.

Of course, whenever the maximum temperature of the furnace is never high enough to soften the brick, then over-insulation need not be feared.

Philadelphia, Pa.

Notes on Cripple Creek Mill Practice.

Editorial Correspondence.

The interest which, for the past two years, has centered in the cyanidation of Cripple Creek raw ore will not abate much this year. In fact it is more likely to increase for the reason that we do not yet know the chemical details of one of the processes used, and further, that a wholly new principle is to be applied this year in a new mill now building. The successes of 1909 and 1910 will be followed, we hope, by still another in 1911, and the field for local treatment of Cripple Creek ores be thereby greatly enlarged. It seems certain even now that the list of metallurgical failures at Cripple Creek will not be added to if intelligence is displayed by those who direct future ventures.

At the present writing the secret of success is told in a few words—ordinary cyanidation assisted by preliminary concentration and the application of cheap chemicals with the cyanide. The object of the concentration is to remove a small quantity of refractory mineral, the retention of which would only increase the time of extraction of the whole mass, or possibly defeat the whole process. The comparatively small quantity of the ore thus removed can be sold to smelters or given such special treatment as it requires. But the prospect now is that even this step will be rendered unnecessary by the Clancy process which will be in operation in a few months.

Stratton's Independence Mill.

The method of treatment used in this mill was worked out by Mr. Philip Argall after many years' experience with the cyanide process and Cripple Creek ores. Based on experiments made at a much earlier date, the plant finally was put in operation in April, 1909, after some unavoidable delays and in spite of the unfavorable criticism of doubting Thomases. The problem was to treat at a profit a large dump of low-grade ore, amounting to almost 1,000,000 tons, which had accumulated for many years. The average value of the dump was determined by extensive sampling to be $3.80 per ton. This figure is much lower than the cost of treatment at some of the earlier mills and is even less than the lowest treatment rate now offered by custom mills on low grade Cripple Creek ores, viz., $4 per ton. That the cost of treating this dump in the Independence mill is about $1.50 per ton is evidence of the general improvement in milling methods and of the ability of the technical staff in charge of the work. The mill was the first successful cyanide plant treating sulpho-telluride ores in the Cripple Creek district, although several successes had been scored previously in cyaniding the oxidized ores. It has been steadily enlarged since beginning operations from an original capacity of 4500 tons per month to practically more than 9000 tons per month at the present time.

Crushing and Grinding.—The dump ore is mined by electric shovel, loaded into 4-ton cars and hauled up an incline to the coarse crushing plant. Two cars are in use, one being loaded while the other is dumping. In this way a car is loaded and dumped about every three minutes, which permits crushing a day's run of 300 tons for the mill in about four hours. The cars are dumped directly over a 7½ D Gates gyratory crusher which reduces the ore to 4-in. size. The crushed ore is discharged over a grizzly with 2-in. openings, the oversize falling onto a picking belt where pieces of steel and wood are removed. The belt discharges into a 5 K Gates gyratory which crushes the ore to 1½-in. size. The product of the second gyratory is combined with the undersize of the grizzly and elevated by a belt conveyor to the storage bin in the concentrator building.

The design of the storage bin and feeder insures a positive steady feed of ore. The bin is circular with conical bottom, the cone terminating in an opening 6 ft. in diameter, through which the ore feeds into an open cone about 12 ft. in diameter at the top and placed with its upper openings in the same horizontal plane with the 6-ft. opening of the main bin. The lower cone terminates in an opening about 12 in. in diameter, through which the ore is discharged by a revolving table feeder. By this construction the ore flows freely and the bin may be completely emptied without extra attention.

The method of adding lime for protective alkalinity differs from the usual practice of adding it in a solution form. As the ore leaves the storage bin it is moistened with a small quantity of mill solution, which settles dust and slakes the solid lime, which is added continuously at the rate of about 1000 lb. for the day's run of 300 tons of ore. This gives a protective alkalinity in the solution of .002 per cent. The mixed ore and lime are then elevated to bins above the rolls, and further reduced in two successive sets of rolls to ⅜-in. size. This material, which constitutes the Chilean mill feed, is then elevated and distributed to three bins, one for each mill. The Chileans are of the fast running type, 6 ft. in diameter, making 33 r.p.m. They grind in mill solution which is kept at a strength of about ½ lb. KCN per ton and discharge through 12-mesh screens. The capacity is about 100 tons of ore each per 24 hours, 60 per cent of the pulp passing 150-mesh screen.

Classification.—Sand and slime are classified prior to concentration in two spiral classifiers, each of 300 tons daily capacity. The sand is delivered 5 ft. above the feeding point, carrying from 4 per cent to 7 per cent slime, depending on the varying nature of the ore. The slime emerges 6 in. below the feeding point, carrying about 3 per cent of sand coarser than 150-mesh.

Sand Concentration.—Mill solution is added to the sand as it leaves the classifier, and the pulp is distributed to 20 Card tables. Two grades of concentrate are obtained, the first carrying about 5 oz. gold per ton, which is shipped to the smelter; the second with about 0.75 oz. gold, which is sold to

chlorination works. The Card table is preferred for this work as it gives a cleaner division between the two classes of concentrate and a consequent financial advantage in their disposition. The sand tailing is laundered to a sump and delivered by centrifugal pump to the cyanide plant for further treatment.

The coarser and heavier sand is separated by spitzlutten and reground in a tube mill. The reground pulp is classified by cones into sand and slime, the former being treated on two Card tables and the latter on four Deister slimers. All of these tables produce first and second grade concentrates.

Slime Concentration.—The slime from the classifiers is thickened in 16 8-ft cones. The underflow is distributed to 10 Deister slime tables and four vanners. Like the Card tables, the Deisters make two grades of concentrate, the first carrying about 7 oz. gold per ton and the second from 0.50 oz. to 0.70 oz. The total concentrate removed by sand and slime treatment amounts to about 3 per cent by weight of the ore treated and contains 40 per cent of its value. The tailing from the Deisters and vanners is laundered to a sump which also receives the overflow from the thickening cones and the tailing from the tables treating reground sand. A centrifugal pump returns the slime to the cyanide plant for further treatment.

Cyanide Treatment.—The regular cyanide treatment is supplemented by the use of bromo-cyanogen. The sand and slime pumped from the concentrator are delivered to the head of the cyanide mill where they are reclassified. The sand is discharged to a reciprocating conveyer which distributes to six sand filter bins, each of 250 tons capacity. The sand enters the bins with an assay value of 0.06 oz. to 0.08 oz. gold per ton and is discharged, after treatment, with 0.03 oz. to 0.04 oz. The treatment is as follows: After a bin is filled and drained it is leached with solution strengthened by the addition of 25 lb. KCN to each bin. When this solution has filtered the sand is given successive washes with barren solution and water and sluiced out.

The slime from the classifiers flows to four large settling tanks, the overflow from which is usually returned to mill solution storage. It is the intention to keep the mill solution at a strength of ½ lb. KCN and at a value not exceeding $0.50 per ton. When this is exceeded the tanks are not allowed to overflow, but are decanted to the precipitating department. The thick settled slime is drawn from the bottom of any one or all of the tanks and pumped to the agitator for 10 hours agitation, with air at 30 lb. pressure. A charge consists of from 75 to 100 tons of dry slime and from one and one-half to two times that amount of solution. Thirty lb. KCN are added to each charge. The agitation tanks are modified from the standard construction in air agitators by shortening the central tube one-half, thereby securing the full benefit of the air in agitating the pulp in the upper half of the tank. After 10 hours' agitation the charge is pumped to a second agitator for special chemical treatment.

Filtration.—The vacuum filter is of peculiar construction, generally according to the Cassell type, and consists of 110 leaves 7.5 ft x 6 ft. The capacity is sufficient to handle the contents of an agitation tank in seven or eight charges in 24 hours, three hours being the time required for a complete cycle of loading, washing and discharging. Vacuum is maintained at 10-12 in., which is practically the best obtainable in slime filtration at the altitude of 10,000 ft., where the barometer normally stands at about 20 in.

Precipitation.—The filtered gold solution is clarified in two Johnson presses and flows to the zinc boxes for precipitation. The daily tonnage of solution precipitated is about 350 tons, or little more than the tonnage of ore treated. Occasionally, when the mill solution rises too high in value, several hundred tons may be precipitated in addition to the regular quantity.

The gold precipitate receives the usual acid treatment and is discharged to a pressure tank connected with a small filter press. The press cake is dried, roasted and ground in a small tube mill using pieces of shafting for the grinding medium. The object of the last step is to put the precipitate into as uniform condition as possible so that an accurate sample can be taken. Prior to the introduction of this practice a great deal of time was consumed in making settlement with the smelter on account of the variation of the sample taken for assay.

The Independence mill was erected at a cost of over a quarter of a million dollars. The preliminary experiments indicated that a fair profit could be made by the treatment of 10,000 tons per month, and it is interesting to know that the actual working costs have checked the estimates remarkably close. No official figures have yet been given by the company, so that details of costs and profits are not actually known to the public, but it is stated that operations are being conducted at a satisfactory profit.

The mill is equipped with duplicate pieces of machinery at those points where a break would interfere seriously with continuous operation. Cement floors, launders and sumps effectually prevent the loss of valuable cyanide solution and contribute to the cleanliness of the plant. Crushing and concentrating in cyanide solution, which was one of the features concerning which skeptics expressed their doubts, has not given rise to any trouble, either by serious loss of solution or by ill effects on the workmen. Altogether the plant has been a success on which the company and staff are to be congratulated.

Following is a condensed statement of the flow of ore through the mill:

Ore from dump in 4-ton cars, to 1.

1—7½ D Gates gyratory, discharging to 2.

2—Grizzly; oversize to picking belt, thence to 3; undersize to 4.

3—5K GGates gyratory, discharging to 4.

4—Incline belt conveyer to 5.

5—Mill storage tank; lime and a small quantity of mill solution are added to the discharge and the mixture elevated to bins, thence to 6.

6—Two sets of rolls; product elevated to bins, thence to 7.

7—Three 6-ft. Chilean mills, 12-mesh screen; discharge to 8.

8—Screw classifiers; sand to 9, slime to 14.

9—Distributor to 20 Card tables; high grade concentrate to smelter and low grade to chlorination mill; tailing to 10.

10—Sand tailing sump; centrifugal pump to cyanide department, 17. Coarsest sand reground in tube mill; product to 11.

11—Two classifying cones; underflow to 12, overflow to 13.

12—Two Card tables; concentrate shipped, tailing to 16.

13—Four Desiter slime tables; concentrate shipped, tailing to 16.

14—Sixteen thickening cones; underflow to 15, overflow to 16.

15—Ten Deister tables and four vanners; concentrate shipped and tailing to 16.

16—Slime tailing sump; centrifugal pump to cyanide department, 17.

17—Cyanide mill; screw classifiers; sand to 18, slime to 19.

18—Sand leaching tanks; filtrate to 23, sand discharged.

19—Four settling tanks; overflow to mill solution or to 23; thickened pulp to 20.

20—Agitating tank for 10 hours; thence to second agitator for special chemical treatment for six hours, thence to 21.

21—Storage tank for filters, to 22.

22—Vacuum filter; gold solution to 23, slime to discharge

23—Two Johnson clarifying presses, to 24.

24—Seven zinc boxes; barren solution to mill storage; precipitate to acid tank and filter press. Press cake dried, roasted, pulverized, sampled and shipped to smelter.

The Colburn Mill.

The latest development in the local treatment of ore at Cripple Creek is the decision of Mr. E. A. Colburn, principal owner of the Ajax mine, to erect a mill embodying the much discussed Clancy cyanide process. This will be the first commercial application of the Clancy principles and the outcome

will be awaited with great interest. Metallurgists generally are favorably impressed with the principles of the process and earnestly hope that the first installation will successfully meet the prime requisite of making money.

The Colburn mill, as it is to be known, is being erected near the Ajax shaft, above the town of Victor. Both mine and dump ore will be treated. The plant will comprise two principal buildings, one for crushing and sampling and the other for cyanide treatment. The excavations have been made, concrete foundations put in and it is expected that the plant will be in operation this summer, probably in June.

An aerial cableway will be installed for transporting ore from the dump to the sampler, while mine ore will be delivered from the shaft by electric tramway. The sampler will have a capacity of 100 tons in eight hours and will have to be run two shifts to supply the cyanide plant, the capacity of which will be 200 tons in 24 hours.

The sampler will be arranged for the separate treatment of leasers' and company ore. Coarse crushing will be done in a Symons gyratory, the product from which will be elevated and diverted to steel bins. One of the latter will be a scale hopper bin for leasers' ore and the other an ordinary bin for company use. From these bins the ore will be elevated to the roll and screen system. The sample will be taken by two Snyder samplers when the ore is reduced to ¼-mesh. The final roll product will be reduced to 12-mesh and elevated by belt conveyer to the cyanide plant. An exhaust dust collecting system will gather the dust from various parts of the sampler and conduct it to a settling chamber with hopper bottom, from which it can be drawn onto the conveyer belt and sent with the ore to the mill.

At the head of the cyanide mill will be three grinding and classifying units, each consisting of a storage bin for crushed ore, a classifier and a tube mill. The bins will be circular in form, built of steel, with bottoms of rock and cement sloping at an angle of 45°. The ore will flow from the bins into the classifiers, which will be of the spiral vane form. The slime will overflow to the agitators and the sand will be ground in tube mills. The tubes will be of a shorter type than customary, 5 ft. x 16 ft., and will have El Oro lining and screw feed. They will grind in closed circuit, their discharge being returned to the classifiers.

The air agitators will be of special construction, suited to the application of electrolysis as well as of agitation. The form will be circular, with cone bottom, and the ratio of height to diameter of the circular portion will be 1:1. The central air tube will be about half the height of the tank, so that the pulp in the upper half of the tank will be thoroughly agitated by the air rising from the central tube.

The agitators will be fitted with electrodes for the application of the electric current during agitation, both for extraction of the gold from the ore and for regeneration of the cyanide solution. The cathode will be in the form of iron sheets suspended in the pulp around the inner periphery of the tank, and the anode of fused ferric oxide suspended near the center. There will be five of these tanks, which will be used for storage as well as for agitation.

The Moore vacuum filtration system will be used. On account of the location of the mill with respect to the town of Victor and adjacent mining property it will not be possible to discharge the slime tailing by sluicing. An automatic system of stacking the tailing on the dump is now being worked out whereby the tailing will be removed from the Moore tank by a screw conveyer and delivered into a hopper. From the hopper it will be carried by bucket tramway to the dump. It is planned to make the loading and dumping automatic and to have the bucket dump at a series of different points on successive trips.

The Merrill system of zinc dust precipitation will be installed and the gold refined at the mill.

The plans contemplate a mill as nearly automatic in operation as possible, with a minimum labor requirement. It is expected that two men for each of two shifts in the sampler and two for each of three shifts in the mill, in addition to the superintendent and chemist, will be sufficient. The mill design and construction is under the direction of S. A. Worcester and the metallurgical details are being worked out by A. W. Warwick.

Essential Points in the Management of a Chemical Industry.

By John Woods Beckman.

In every chemical industry the "chemical control" and the "chemical balance sheet" are two fundamental points which demand the closest attention. They are the backbone. Without them no chemical industry can be carried on successfully nor can the fullest return from the investment in the plant be expected.[1]

"Chemical control" is understood in a general way to mean not only the absolute knowledge, on the part of the factory managers, of the chemical constituents in the raw materials that enter the factory at one end and the finished product leaving at the other end, but the "chemical control" involves the chemical knowledge of each individual step in the whole process.

Naturally the chemical control is distinctly a chemical question, and is in the special charge of the chemical engineer at the plant, and to do the chemical work it necessitates the equipping of a first-class laboratory, with an outfit well adapted to accurate, quick routine work, as well as offering the possibility of carrying out the most complicated analysis and doing some research work. The laboratory and its staff of men and boys are the tools for the efficient carrying out of the chemical control, and the control can only be carried out in conjunction with the laboratory. On this account the chemical staff can not be too well selected, as it is the heart and brain combined of the factory.

A great number of analyses of more or less importance have to be turned out daily by the laboratory. The most important ones are the determinations made of the incoming raw materials, which cannot be given too much care. The importance of these determinations is twofold, being based on financial reasons as well as on factory-operation reasons.

The raw materials purchased, or credited to the factory, have to come up to contracted for or specified contents, and the only guide as to the accuracy of the bills presented is a most careful determination of percental composition of the raw materials.

With this knowledge at hand of the incoming raw materials it is possible for the chemical engineer to arrange the process accordingly, but of almost equal importance is the knowledge of the actual amounts of every small unavoidable admixture occurring in the raw materials. These are often the makers or the spoilers of a process, and on this account it is of vital importance to the industry that the raw material analysis be carried out to the utmost completion, even having the "traces" determined.

Catalytic action is more and more being appreciated in the carrying out of a chemical process, and Ostwald says that "there is probably no kind of chemical reaction which can not be influenced catalytically, and there is no substance, element or compound, which can not act as a catalyzer," and his words are being verified each day. Many are the chemical industries that have stumbled and fallen for unexplainable reasons which could have been remedied if a complete knowledge had been available of all the ingredients of the raw materials.

As is well known there are catalytic agents which assist the chemical reaction, while there are such which produce opposite phenomena, called negative catalysis, retarding and upsetting

[1] In this article no mention is made of the management of financial matters nor of labor.

the normal course of the chemical reaction. With a fair knowledge of the catalytic phenomena, and with a complete knowledge of the constituents of the raw materials, it is possible for the chemist very often to remove stumbling blocks in the factory, and again improve conditions by some small addition of a foreign substance to act as a catalytic agent. The only safe way that makes it possible to improve the process or prevent destruction is the complete knowledge of the ingredients in the raw materials.

With these complete records at hand it is possible in most cases, if the mechanical manipulation of the process is the same day out and day in, to speedily overcome any unevenness in the working of the plant due to any increased or decreased amount of admixtures to the raw materials.

The utmost care in regard to the raw materials will undoubtedly repay in more than one way, but perhaps equally important from another point of view is the careful chemical control of the various steps in the chemical process. Day out and day in without fail the routine analyses—the safety valve of the process—have to be made and tabulated.

The tabulation of the analyses from day to day is exceedingly important. It shows at a glance how the whole process is working and whether the fullest yields at the different steps are obtained. The more graphical the tables are the more valuable they are. The day to day variations are often scarcely noticeable, but the week to week and month to month, and often year to year, variations count, and if they are carefully studied often losses can be avoided and improvements noticed that otherwise would totally have slipped attention.

This daily factory record is of fully equal importance to the operation of the factory as the log is to the captain of a vessel to bring it safely into port. The records have to be planned and kept in such a manner that they give sufficient information, making it possible for a person familiar with the work to take them up thousands of miles away and be able to draw conclusions and direct the operation of the plant at that distance. Any clerk sufficiently instructed can keep the records, but the planning and the drawing of valuable conclusions from them are only possible for the chemist with an observing mind and a wide knowledge.

Where chemical control to the full extent of the meaning of the word is instituted in the factory there is no hesitancy in mode of operation; complications that may arise can be traced to one or more factors, which will be very apparent after a comparative study of the records, and after doing this it will be in most cases an easy matter to bring back the process into the successful grooves.

The factories where chemical control is not instituted and no records exist excepting such as may by chance have been scribbled on the wall by a foreman or a laborer, or scattered notes in the office with no meaning at all, are in a state of upheaval when a snag is struck. General consternation reigns. The chemist, the foreman, the laborer, the manager all have their own opinion why this and that and another thing happened, and what is the most expeditious way of overcoming the trouble. Time is lost and the whole process is liable to be upset with subsequent money loss to the stockholders.

The "chemical balance sheet" is, again, a record which is of equal importance with the "chemical control." The former deals exclusively with weights, while the latter deals exclusively with percentages. The "chemical balance sheet" is generally understood to represent the bookkeeping of the process. The balance sheet has to be kept up for every day with the same care as the bank keeps up its balance sheet. Every cent has to be accounted for which has been handled during the day in the bank. Similarly should the factory management be able to account at the end of the day for every pound of the raw material stock, how far it has gone on its way through the factory, how great the losses are in the different steps of the process, and finally should the losses, together with the amount present in the finished product, total up to the raw material stock consumed during the day.

In some processes these items are hard to obtain daily, especially so in continuous processes. In such cases the balancing should be done at as frequent intervals as possible to avoid any cumulation of errors. This balancing of the process cannot be too rigid, missing nothing, and should in accuracy compare with the financial balancing of the assets and liabilities.

In close connection with above records come the determinations of the efficiency[2] of the various steps and apparatuses used in the process. The factory with the highest efficiency record is the most economical factory, and is the money-making factory, often much to the astonishment of its competitors. It is able to pay dividends, while its competitors are barely able to keep the wheels turning.

The daily records of the efficiency will quickly show their value and repay the trouble in keeping them. Suppose that an apparatus designed to dry with steam only utilizes 50 per cent of the drying capacity of the steam at that temperature, it is apparent that an enormous waste of fuel and calories takes place at that point. This would not be noticed to its full extent excepting by the careful keeping of a daily efficiency record.

In any chemical or electrochemical industry it is possible to determine efficiencies. They may be relative or absolute. The absolute efficiency is in some cases impossible to obtain, and while the absolute is very much desirable, the next best is a relative efficiency. The zero point from which to start the efficiency determination has to be arbitrarily chosen, and it may occasionally happen that the efficiency shows a figure higher than 100 per cent, which only indicates that the zero point has been chosen too high.

The points briefly dealt with above are very far reaching. They are not new, though altogether too little attention is given them for the good of the chemical industries and the pockets of the stockholders. The same principles have been applied to questions of money in the financial world and in other industries. In the chemical industries, perhaps due to the comparative vagueness of these, since they do not deal with measures as tangible as in other industries, these principles have been slow to be applied. But they have been applied in some chemical industries, and those are the money-making ones, prospering even though there may be very heavy competition and a very small margin of profit.

Niagara Falls, Canada.

American Vanadium Facts is the title of a new and neat house organ issued by the American Vanadium Company, of Pittsburgh, Pa., miners of vanadium ores and manufacturers of "Amervn" ferrovanadium. The first issue, which has just been received, contains articles on general properties of vanadium and ferrovanadium, on vanadium steel forgings, on vanadium steel governor pins and their endurance and on vanadium steel castings. The Union Steel Casting Company, of Pittsburgh, is reported as having obtained recently the following results with vanadium steel castings: Tensile strength, 123,090 pounds; elastic limit, 86,220 pounds; elongation in 2 inches, 10 per cent; reduction of area, 15.6 per cent. These figures were obtained on 34-inch blooming mill pinions. After a special heat treatment a coupon from these pinions gave 20 per cent elongation and 40.6 per cent reduction without material change in the tensile and elastic figures.

[2]When I speak here of efficiency, I generally mean the "scientific" efficiency, although in some cases, under practical conditions, it may be more economical to aim at what might be called the "commercial" efficiency. Undoubtedly in most cases the economical way of operating is trying to reach the upper goal which is the "scientific" efficiency. But under special conditions and in special processes, the aim at reaching a high efficiency may influence the quality of the product or the output to such an extent that it may be better to aim at an efficiency that has been established through practical working as being the most economical, and this may be called the "commercial" efficiency.

A French Plant for the Fixation of Atmospheric Nitrogen

In our February issue, pages 102 and 103, a brief description was already given of the new works of La Nitrogene Company in La Roche-de-Rame, Hautes-Alpes, France, using the Pauling process for making nitric acid and nitrates from air.

A very full description of the same plant is given by Jules Vanderpol in the January issue of *La Houille Blanche*, from which the following translation has been prepared.

Reactions.

The chemical reactions are summed up as follows: The reaction in the arc in the electric furnace is

$$N + O = NO \qquad (1)$$

The gases which leave the electric furnace have a temperature of 900° C. to 1000° C. and contain 1 per cent to 1.5 per cent NO and 98.5 per cent to 99 per cent air. When this gas mixture is cooled to 600° C. the NO reacts with the oxygen in the air according to the equation

$$NO + O = NO_2 \qquad (2)$$

This reaction is somewhat slow on account of the low concentration of NO in the gas mixture (the speed of formation being inversely proportional to the concentration of NO) and is complete only at a temperature of 130° C. It is, therefore, necessary to let the gas mixture remain for some time in the large-sized tower in which this reaction goes on. This is the object of the oxidation tower which follows the cooling apparatus.

There remains the absorption of NO_2. As a matter of fact the gas is really a mixture of NO_2, NO, N_2O_3 and N_2O_4, the N_2O_3 being formed by the reaction of NO_2 on NO and the N_2O_4 being due to the partial polymerization of NO_2 in the oxidation tower.

The gas is subjected to the action of water at a temperature near 30° C., the reaction being

$$3\,NO_2 + H_2O = 2\,HNO_3 + NO \qquad (3)$$

Secondary reactions also take place, the oxides N_2O_4 and N_2O_3 reacting with the water as follows:

$$N_2O_4 + H_2O = HNO_3 + HNO_2 \qquad (4)$$

$$N_2O_3 + H_2O = 2\,HNO_2 \qquad (5)$$

These reactions are immediately followed by the decomposition of nitrous acid.

$$3\,HNO_2 = HNO_3 + 2\,NO + H_2O \qquad (6)$$

These reactions take place in the acid absorption towers. According to (3) only two-thirds of the NO_2 is changed to nitric acid, the remaining one-third being reduced to NO.

According to (6) only one-third of the nitrous acid is oxidized to nitric acid, the remaining two-thirds being changed to NO. Hence, a new oxidation of NO becomes necessary. This is the reaction between the peroxide NO_2 and oxide NO.

$$NO + NO_2 = N_2O_3 \qquad (7)$$

The absorption by water is completed by a reaction with soda

$$N_2O_3 + H_2O = 2\,HNO_2 \qquad (8)$$

$$2\,HNO_2 + Na_2CO_3 = 2\,NaNO_2 + CO_2 + H_2O \qquad (9)$$

The nitrite produced always contains a certain proportion of nitrate, since the gases when they leave the absorption towers do not contain exactly one molecule of NO for one of NO_2. The NO predominates and the reactions are

$$3\,NO + NO_2 + 5\,O = 2\,N_2O_5 \qquad (10)$$

$$N_2O_5 + H_2O = 2\,HNO_3 \qquad (11)$$

$$2\,HNO_3 + Na_2CO_3 = 2\,NaNO_3 + CO_2 + H_2O \qquad (12)$$

These last reactions take place in the nitrite tower. These are the principal reactions. In reality they are still more complex and change with temperature, pressure, speed and quantity of gases in movement, etc.

La Nitrogene Company Works.

Electrical energy for the works at La Roche-de-Rame of La Nitrogene Company is generated from water-power, there being four turbine-driven main generators, each of 2000 hp. The energy is transmitted over an overhead transmission line of aluminium of 600 m (2000 ft.) length to the furnace house.

The furnaces are installed in a hall (Fig. 1), 15 m (50 ft.) wide and 45 m (150 ft.) long. They are built of fire brick reinforced with steel and are arranged in groups of three. Their horizontal section is 1 m x 1.20 m (3 ft. 3 in. x 3 ft. 8 in.) and their height is 3 m (10 ft.). Their design is characterized by simplicity and robustness.

The design of the furnace is shown in Fig. 2 in several sections. Each furnace contains two vertical chambers *f f*, each chamber containing a pair of electrodes *e e* of the form of a horn lightning arrester, as shown in the diagram at the right-hand bottom of Fig. 2. The electrodes are of steel, interchangeable and inside cooled by circulating water.

The arc between the two electrodes is blown upward into the form of a large sheath 60 cm to 70 cm (24 in. to 28 in.) high by means of hot compressed air, which is blown into the furnace through the tuyère *t* at a speed of 400 m per second. To permit the air to rise between the electrodes, the latter

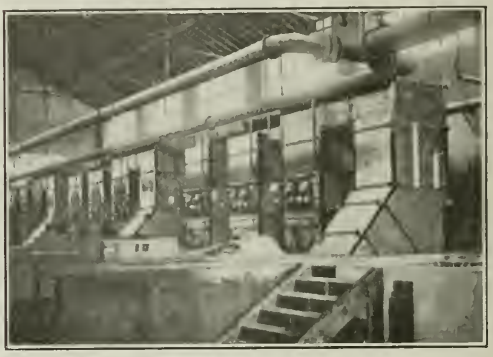


FIG. 1.—FURNACE HALL.

should not be too near together. But the distance between the two lower ends must also be sufficient to light the arc. To overcome this difficulty Pauling uses auxiliary electrodes *c* ("couteaux d'allumage") in form of thin copper blades, the ends of which can be brought near together (2 mm to 3 mm or 80 mils to 120 mils) to light the arc again if by accident it should be extinguished.

i are insulators, *f* is the passageway through which the cool air enters, *s* is the exit of the gas mixture produced. The wall which separates the two arc chambers from each other contains in its center another channel *u* by means of which a cold gas mixture is fed to the upper portion of the arc flame. This gas mixture ("umlauffluft" or circulating gas) is taken from the gases which have already been subjected to the reaction and have been cooled in their passage through the cooling apparatus. The "Umlauffluft," therefore, serves to cool the gases coming from the arc, without diluting their content of nitrogen oxides. This "Umlauffluft" is not indispensable for the satisfactory operation of the furnace, as the plant has worked satisfactorily without the "Umlauf" being in use.

The copper blades forming the auxiliary electrodes last in the average 20 hours and can be replaced in a moment. The main electrodes last 200 hours, as they are internally cooled.

An opening in the exit passage *s* of the gases permits the introduction of a pyrometer for measuring the temperature and of a pressure gage to control the fan which sucks the gases to the cooling apparatus.

Each furnace has two arcs in series. The 50-period, 6000-volt, three-phase currents supplied from the power plant are used directly. The furnaces are single-phase and the three phases of the three-phase supply are connected to the three sets of furnaces in each group, respectively.

The electrical accessories are two self-inductance coils, a transformer used in starting the arc and a rheostat, all being placed in front of each furnace.

Nine furnaces, each of 600 kw, are in normal operation, but the installation of 1000-kw furnaces is contemplated, as they are more stable in operation and have a higher efficiency.

The air supplied to the furnaces comes from a 250-hp turbo-compressor running at 3000 r.p.m. The air is preheated in a preheater built of metallic tubes, the heat in the gases coming from the furnace being used to preheat the gases which go to the tuyères.

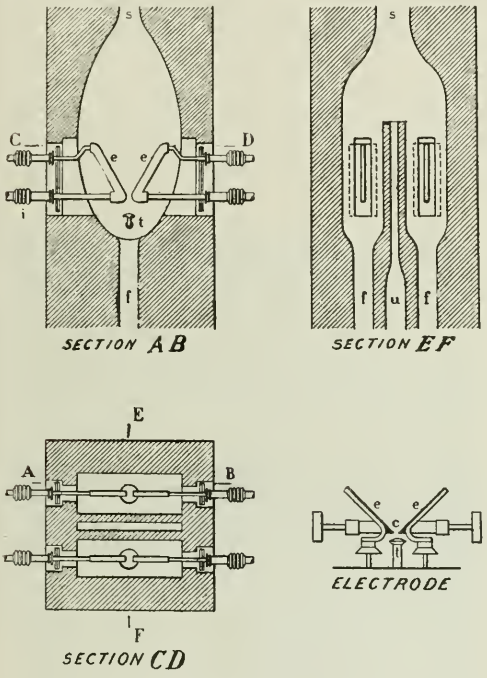


FIG. 2.—SECTION OF FURNACE.

Fig. 3 is a diagram of the electrochemical plant. *B* is the furnace house, *E* the preheater, *F* the groups of furnaces, *D* the repair shop and laboratory, *R* the cooling towers, *G* a chimney, *O* the oxidation tower, *N* nitrite towers.

H is the absorption shed containing the absorption towers A_1, A_2, A_3, A_4, A_5 and the concentration apparatus C_1 and C_2, while the acid is stored in *M*.

The gases coming from the electric furnace contain 1.15 per cent to 1.50 per cent of NO and have a temperature of 900° C. to 1000° C. They are first passed through the preheater *E*, where they give off some of their heat. This is important from the viewpoint of energy economy.

The gases are then passed through the cooling towers *R*. These two towers are alternately in operation. They are 5 m (16 ft.) in diameter and 12 m (40 ft.) high and are filled with brick which offers a large surface to cool the gases. When the bricks of one tower *R* have become hot, the gases are switched over so as to pass through the other tower, while fresh air is passed through the first tower (natural draught being obtained by means of the chimney, which is 26 m. or 85 ft., high) and the brick in it is thereby cooled. Hence while the gases are cooled and the brick heated in one tower, the brick in the other tower is cooled by fresh air, and so on.

The oxidation of NO to NO_2 begins when the temperature of the gases has sunk to 600° C. But this temperature is not yet reached in the cooling towers.

The gases are now sucked out of the cooling towers by means of a 15-hp fan which produces suction as far back as in the furnaces. By the fan the gases are driven into a large reinforced concrete tower, called the oxidation tower and shown in Fig. 4. It is 23 m (75 ft.) high, 10 m (33 ft.) in diameter and the thickness of the walls is 10 cm (4 in.).

Two lines branch off at this point. The first branch line represents the "umlauf" gas line through which part of the cooled gases is returned to the furnaces.

The second branch line consists of an aluminium pipe line, cooled with water, and this is put in service when the cooling by the towers has not been sufficient.

The oxidation tower (Fig. 4) is filled with acid-proof refractory brick. It is in this tower, in which the gases remain for a relatively long time, that the cooling of the gases is completed and the NO is changed to NO_2.

The absorption apparatus and concentration apparatus for the acid are placed in the shed *H*, in Fig. 3. This is 22 m (72 ft.) high and covers an area of 15 m x 30 m (49 ft. x 98 ft.).

The absorption of the gases by the water takes place in the five absorption towers A_1, A_2, A_3, A_4, A_5 (Fig. 3), through which the gases pass in the direction from A_1 to A_2 to A_3 to A_4 to A_5, while the water (gradually containing more and more acid) flows in the opposite direction.

Montejus, operated by compressed air, raise the solution to

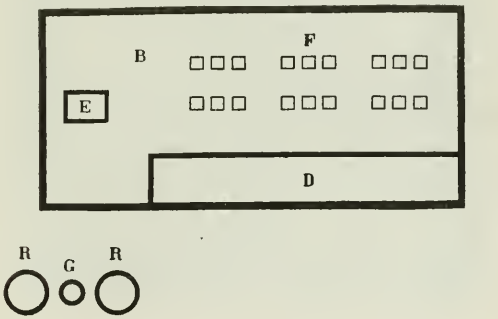


FIG. 3.—DIAGRAM OF ELECTROCHEMICAL PLANT.

the top of the different towers. Each tower contains 250 tons of acid-proof stoneware packings.

In general the concentration of the acid in the tower A_5 is 5° Beaumé, further 8° to 10° Be in A_4, 12° to 15° Be in A_3, 18° to 20° Be in A_2, and 25° to 30° Be in A_1, which corresponds to 35 per cent to 40 per cent of HNO_3.

From the absorption tower A_1 the acid is passed to the concentration apparatus C_1 and C_2 (Fig. 3). It is passed through a series of porcelain or fused-quartz vessels arranged in stairway fashion. While passing through the vessels the acid is

heated by direct contact with hot gases which come from the furnace. These gases are thus charged with water and nitric acid vapor. To condense the nitric acid the gases are passed through a cooling coil of stoneware which offers a very large cooling surface. The gases are then passed into the oxidation tower, where they mix with the gases coming from the furnaces.

The acid obtained by this concentration process contains about 50 per cent HNO_3 (about 36° Be) and it is possible to push the concentration up to 60 per cent (40° Be) by this process. But further than this it is not possible to go in this way, since the vapor produced has a concentration which increases with the concentration of the solution, and for a con-

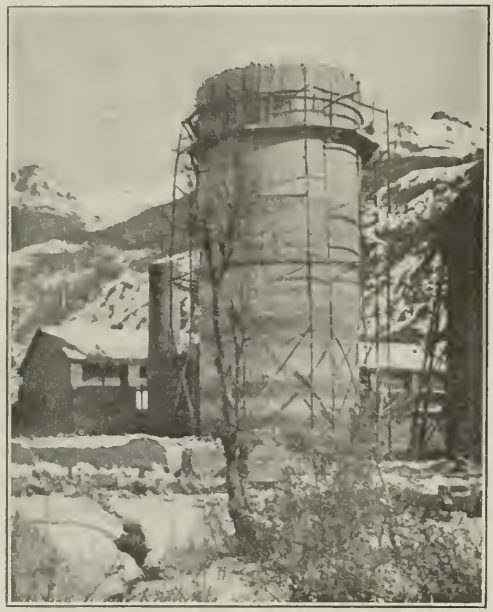


FIG. 4.—OXIDATION TOWER

centration of 60 per cent the vapor produced has exactly the composition of the liquid.

To obtain higher concentrations other means must be resorted to. At the Patsch plant an electrolytic process of Pauling is used (as described in his United States patent, noticed in our Vol. VI, 1908, page 300). But on account of the high cost of the apparatus, which requires considerable amounts of platinum, this method is not used in the French plant

In the latter the following process is employed: The nitric acid, of 60 per cent, is mixed with two parts of its weight of sulphuric acid of 92 per cent, then introduced into a tower filled with Volvic packing and heated from the outside In this way a nitric acid with a concentration of 98 per cent may be easily obtained. The sulphuric acid withdrawn from the bottom of the tower, which has been diluted by the absorbed water to a concentration of 77 per cent or 80 per cent, is concentrated and then used again.

The gases which pass out of the absorption towers contain NO and NO_2. They are passed through an acid filter in which the last traces of the acid are condensed. The gases then pass into the nitrite towers (V in Fig 3), when they are caused to react with a sodium carbonate solution. The sodium nitrate solution obtained has a concentration of 20 per cent, and is then submitted to evaporation. The hot furnace gases are used for this purpose. The nitrite crystals obtained are white and contain 95 per cent nitrite and 3 per cent nitrate.

The sodium carbonate solution used in this part of the process is the only raw material employed in the plant besides atmospheric air and water.

As to the efficiency it is stated that Pauling guarantees a yield of 60 grams of 100 per cent acid per kw-hour of electrical energy, measured at the entrance of the electric transmission line into the factory. Pauling also guarantees that the cost of the electrochemical plant proper is not more than 120 francs ($24) per kilowatt.

The Process of Zinc Distillation

By George A. Wettengel.

The experiments that have been in progress for the last year at one of the zinc smelting plants in Kansas with a rotatable zinc smelting furnace (United States patent 901,405; this journal, Vol. VI, November, 1908, page 457) will be of interest, as this type of furnace has many advantages over the present type of stationary furnace in labor saving and increased life of retorts.

The present type of furnace necessarily requires much hard manual labor and the chief object of my improvements is to provide a furnace adapted to be mechanically operated. This is accomplished in the following manner:

Fig. 1 is a front view of the furnace in operating position, Fig. 2 shows the furnace in the charging position. The furnace consists of a steel shell, with two tires supported on rollers, one of the tires has a gear attached, engaging the driving mechanism by which the furnace can be rotated to any desired position. The inside of the shell is lined with tile, with suitable shelves for suporting the retorts. The pipes supplying

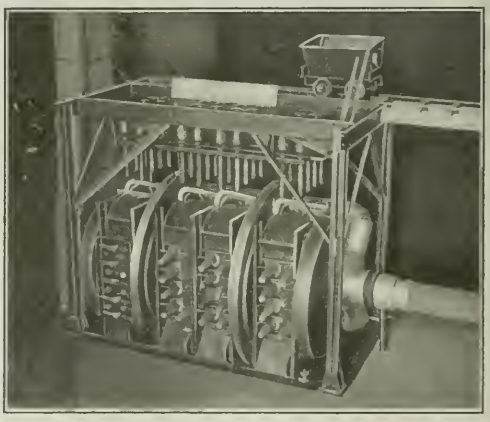


FIG. 1.—ZINC FURNACE IN OPERATING POSITION

gas and air to the burners are attached to and rotate with the furnace.

Above the furnace is the charging floor, containing hoppers, each just large enough to hold the ore mixture for one retort, with shut-off slides and telescopic spouts that are lowered to the top of the retorts so that all of the ore mixture goes into the retorts. Below the furnace is a large hopper for the residues

The ore mixture is dumped onto the charging floor, each hopper filled, and then the furnace is rotated upward so that the retorts are vertical with their open ends in line with the hopper spouts. These are now lowered and slides opened, whereupon the ore mixture falls by gravity into all of the retorts

simultaneously. Then the furnace is rotated downward so that the retorts are now in a horizontal or any position which may be desired for the destillation.

The condensers are put on, luted, stuffed and vented. After completing the distillation process the condensers are removed and the furnace again rotated so that the retorts are vertical, but have now their open ends downward. The residues then fall out by gravity into the hoper under the furnace.

The furnace is then rotated again so as to bring the retorts into a horizontal position and is then cleaned of adhering loam and any remaining residues and then recharged as before. But instead of rotating it to its first operating position it is now rotated with the open ends of the retorts pointing in the opposite direction. The retorts are now 180° from their first operating position. This inverts the retorts so that the bottom of each is used as the top, and vice versa. This brings the slags to the top of the retort chambers, which it melts and drips down onto the distilling ore mixture and is discharged with the residuum.

This inverting of the retorts makes them practically self-cleaning, almost entirely avoid scraping, and increases their life by using two sides. It keeps them practically straight.

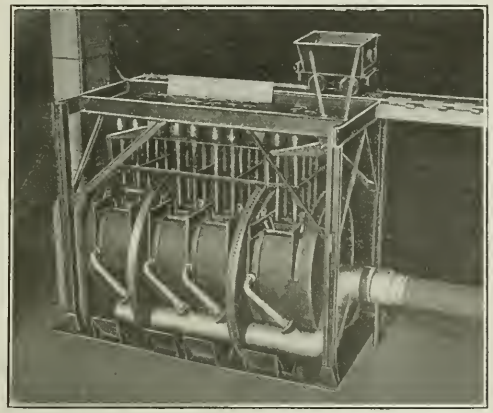


FIG. 2.—ZINC FURNACE IN CHARGING POSITION.

This method avoids the present danger of cracking the retorts and of the slags perforating them.

Distribution of Charge and Dryness of Ore Mixture.

To charge the present type of furnace the charger stands in front of the white-hot retorts facing the charging car which contains the ore mixture and shovels the contents of the latter into the mouths of the retorts, filling them one by one with a semi-cylindrical scoop, requiring eight or nine scoopfuls to fill a retort, the amount depending on the force and skill with which it is thrown in by the charger. This slow process is not only hard, laborious and unhealthful work, because of the intense heat, but also unsatisfactory as all of the retorts do not receive the same amount of ore mixture, it being packed unevenly within the retorts.

The ore is charged into the retorts wet enough to stick together, requiring about 15 lb. (2 gal.) of water per retort. This not only chills the retorts and absorbs heat to distill off the water, but also retards the process of zinc distillation. The steam generated by the intense heat frequently expels the entire charge before it can be vented (commonly called spiessed).

With my rotatable furnace each retort is charged simultaneously from individual hoppers from the charging floor above with a predetermined amount of ore mixture which is uniformly distributed throughout their length. This greatly reduces the time of charging, dispenses with the chargers, necessarily decreasing the cost of operation, increases the time for the distillation process and also increases the capacity of a furnace.

The ore mixture is charged dry, necessarily a more rapid and thorough distillation of zinc is obtained with a considerable saving in fuel consumption without the present danger of chilling the retorts or expelling the entire charge. Thus a more rapid and thorough distilling process is attained with an increased life of the retorts by dry charging.

Sagging of the Retorts.

Since the retorts in the present type of furnaces are only supported at their ends, they gradually sag and distort, not being strong enough to preserve their shape and successfully support the weight of the contained ore mixture and resist the intense heat to which they are subjected during the zinc distilling periods.

The compressive stresses produced in the upper half of a retort by the weight of the contained ore mixture is safely resisted while the bottom half is in tension, the stresses of which the material cannot safely resist as it has a very low tensile strength.

The retort continues to sag until it fails by cracking across the bottom. This not only requires replacing the retort, but occasions large losses of zinc.

With a rotatable furnace any tendency of the retorts to sag, distort, crack or lose their shape from the weight of ore mixture and the intense heat to which they are subjected during the process of zinc distillation is resisted by inverting them for each distillation process. This not only keeps them practically straight, but also avoids the present danger of losing their shape and cracking, necessarily increasing their life and the zinc recoveries.

Slags Perforating the Retorts.

In the present type of furnaces the tops and sides of the retorts are not seriously affected by the repeated distillations, the bottoms, however, are usually attacked by the corrosive slags and perforated, particularly when the ore treated contains a large percentage of iron, manganese or lime. After a retort has been perforated the slags will drip on top of the retort lying below, which will also be perforated, but from the outside. This not only occasions large losses of zinc, but necessitates replacing the retorts.

In a rotatable furnace the slags formed and adhering to the bottoms of the retorts during the process of zinc distillation are brought to the top by inverting them for the next distillation process. The bottom being on the top and vice versa. This adhering slag gradually drips off of the retort and drops on top of the smelting ore mixture and is discharged with the residuum. This not only prevents all the slags from accumulating on the bottoms of the retorts, which reduces their capacity and efficiency, but prevents the corrosive slags from perforating the retorts as in the present practice.

After rotating the furnace downward for discharging the residue it is left in this position for several minutes, during which time small pieces of slag fall out, also burning iron which can be distinguished by bright sparks similar to the sparks from a piece of iron brought to a welding heat. These sparks of burning iron continue to drop from the top of the inverted retorts after they are brought to a horizontal position. This increases the life of the retorts by removing most of the iron and slag, which corrodes the retorts.

Uniformity of Temperature.

In bringing the present type of furnace to the proper temperature for distilling the ore mixtures in the lower tiers of retorts, the upper tiers are subjected to a higher temperature than is required. This intense unequal heat has an injurious effect on the upper tiers of retorts.

In my method the rotatable furnace with its retorts is reversed each day to overcome the unequal heating of the top and bottom tiers of retorts. This reversing brings the retorts

from the top, or hottest, to the bottom, or coolest, part of the furnace each day, thereby prolonging their life.

Condenser Loam.

In the present type of furnace the conical condensers are inserted with their large ends projecting into the open ends of the retorts about 1 in. and lying on the bottom of the latter, thus forming a crescent-shaped joint with an opening at the top of about 1 in., which must be luted up air-tight with loam. After the condensers are removed the loam adhering mostly to the upper half of the retorts must be removed each day to again admit the condensers. This is accomplished with a heavy

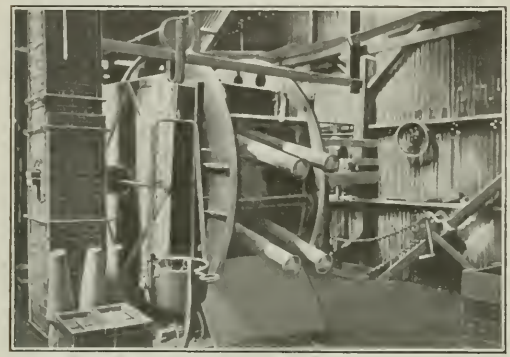


FIG. 3.—EXPERIMENTAL FURNACE IN OPERATING POSITION.

bar, commonly called a bumper, with which the workman bumps off the adhearing loam frequently, chipping off small pieces of the retort with it, until they become leaky or breaking them so that they must be replaced. In a rotatable furnace the retorts are inverted each day, bringing the loam on the top one day and on the bottom the next and vice versa. This greatly reduces the present danger of chipping or breaking the retorts in removing the loam.

Metal Drawing.

In the present type of furnace the metal is drawn off by inserting an iron scraper into the ends of the condensers and scraping out the loam stuffing and the liquid metal into a ladle attached to a crane conveniently supported on a truck which travels on a track parallel with the working face of the furnace. When the ladle is full of metal it is swung out from the furnace, the top skimmed off and its contents poured into molds. Occasionally a condenser is knocked loose accidentally whereby the liquid metal rushes out covering the metal drawer, often seriously burning him. This is very hard, dangerous and unhealthful labor on account of the intense heat and fumes escaping from the ends of the condensers.

In my method the metal is drawn off by rotating the furnace to an angle of about 15 deg. below the distillation position, whereupon the metal rushes out of all of the condensers simultaneously into a trough extending the length of the working face of the furnace and then collects in a ladle at the end. The contents of the latter is then poured into molds. This not only requires a fraction of time to draw the metal as compared with the present method, but greatly reduces the hard, dangerous and unhealthful labor, as the operator is away from the intense heat, fumes and liquid metal issuing from the condensers.

Agitating the Ore Mixture

I rotate my furnace upward after each drawing of metal so that the ore mixture falls by gravity to the rear ends of the retorts. I then continue rotating the furnace until the retorts are again brought to the distillation position, but inverted. By this occasional agitation of the ore mixture the particles of ore and carbon will be more thoroughly mixed and brought into more intimate contact with new surfaces whereby a more rapid and thorough distillation process is attained. This not only increases the recoveries of zinc, but also reduces the time required for the process of zinc distillation.

Composition of Residue.

In the present type of furnace the ore mixture is not uniformly distilled, necessarily the residues at the open ends of the retorts are rich in zinc because the ore mixture does not receive heat enough to thoroughly distill it. Two in. or 3 in. of this residue (called sampling) is scraped out of the open ends of the retorts and is mixed with the ore mixture to be charged for the next distillation process. This not only increased the labor, but reduces the amount of fresh ore mixture that can be charged into the retorts.

By rotating my furnace after each drawing of metal so that the retorts are brought with their open ends upward the ore mixture falls by gravity to the rear ends of the retorts, thereby bringing the ore mixture from the open ends of the retorts where it receives insufficient heat to the rear ends where it will be thoroughly distilled. This not only greatly reduces the labor and increases the amount of fresh ore mixture that can be charged into the retorts, but increases the recoveries of zinc.

New Retorts.

In the present practice when a new retort is placed in a furnace it is not charged the first day, this time being allowed for burning and glazing the retort. A retort is replaced 12 times a year, necessarily losing 12 days per year by not being charged. In a rotatable furnace the retorts are replaced 2 4/10 times a year, necessarily losing only 2 4/10 days per year for burning and glazing. An increase of 9 6/10 days per year per retort capacity of a furnace, or an increase of 9 7/10 per cent in capacity. This greatly increases the capacity of a furnace without any additional labor.

Heat Transmitting Efficiency.

In the present type of furnace with its stationary retorts the heat transmitting efficiency decreases as the slags accumu-

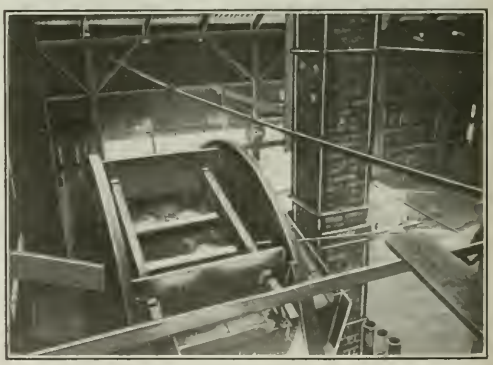


FIG. 4.—CHARGING POSITION OF EXPERIMENTAL FURNACE

late on the bottom of the retort chamber until it is from 3 in. to 4 in. thick during its 30 days' use, necessarily considerable more fuel is required to bring it to the proper temperature for the distillation process than for a new retort.

In a rotatable furnace with its inverting and self-cleaning retorts the slags formed on the bottom of the retort chamber during the distillation process one day are brought to the top of the retort chamber by inverting them for the next distillation process, whereupon the slags drip off onto the smelting ore mixture and are discharged with the residues. These slags corrode the walls of the retort, decreasing their thickness and

necessarily greatly increasing their heat transmitting efficiency.

Capacity of a Retort.

In the present type of furnace with its stationary retorts the slags gradually accumulate on the bottom of the retort chamber until it is from 3 in. to 4 in. in thickness, necessarily greatly reducing its capacity.

In my rotatable furnace with its inverting and self-cleaning retorts the capacity does not decrease, but remains the same until replaced.

In my rotatable furnace the ore mixture is charged dry. It is possible to charge about 10 per cent more ore mixture dry than wet as it fills the retorts more compactly.

Temperature of Furnace While Charging.

In the present type of furnace the heat is practically shut

FIG. 5.—RETORTS.

off for several hours during the charging operation, necessarily greatly reducing the temperature of the furnace at a period when it is necessary to bring the cold and wet ore mixture in the retort to the distillation temperature as quickly as possible.

In my rotatable furnace the temperature is not reduced except as required during the distillation process. All the retorts are charged simultaneously from individual charging hoppers from above with dry ore mixture, therefore, greatly reducing the cooling off period while charging.

Use of Metal Tools.

In my rotatable furnace there are no metal tools used in drawing the metal except the last drawing. This not only prevents breakage and abrasion of the condenser, but greatly reduces the absorption of iron from the metal tools used and a purer metal is obtained.

Fig. 3 is a front view of the experimental rotatable furnace in the operating position. The furnace is rotated as required by means of a wrench and link belting which engages sprocket wheels on the shafts of two of the supporting rollers. It has six ordinary retorts 8 in. in diameter by 48 in. long inside dimensions. To maintain the temperature as nearly uniform as possible pyrometric cones were used inside of one of the retorts (No. 2) as an aid in regulating and recording the temperature. Therefore, only five of the retorts were charged. The furnace was charged in the usual way for several months with such excellent results that I decided to try charging it from the top to demonstrate its practicability.

An inclined runway and platform was constructed to charge from. The platform was on a level with the working face of the furnace when in the charging position. After the residues were discharged, the furnace was rotated so that the retorts were in a vertical position with their open ends upward (see Fig. 4).

It was now charged from above by wheeling the ore mixture up the incline runway onto the platform, then shoveling it into the vertical retorts, filling them one by one with dry ore mixture instead of wetting it as in the present practice. After all of the ore mixture was charged, the last retort was only about one-third full, thereby demonstrating that considerable more dry mixture could be charged into the retorts than of the wet.

The furnace is now rotated to the operating position and the distillation process continued similar to the best present-day practice. The top charging was discontinued after charging about one week as it was hard work to wheel the ore up the steep incline runway.

Fig. 5 shows the six retorts after they were removed from the furnace, having been in 167 days. These retorts are practically as straight as on the day they were put into the furnace. Compare these with the crooked retort to the right of No. 6, which was taken from one of the ordinary type of furnaces after being in use less than one month.

Retort No. 1 was cracked accidentally in putting it in the furnace. It was charged 81 times. Charging was then discontinued as the crack was opening up and losing zinc.

No. 2 was never charged. Pyrometric cones were put in this retort to regulate the temperature.

No. 3 was charged 151 times when a small hole was discovered in one side.

No. 4 was charged 167 times and was still in excellent condition.

No. 5 was charged 143 times when a small hole was found in one side.

No. 6 was still in excellent condition after being charged 167 times.

Fig. 6 is a view of retort No. 4 cut in two, vertically, showing it to be in excellent condition after being in continuous operation for 167 days. There was very little slag in it, and very little corrosion had taken place as the walls of the retort were from ¾ in. to ⅞ in. in thickness. None of the retorts were perforated by the slag.

This is a remarkable showing when compared with the life

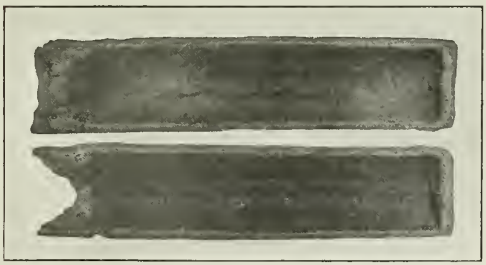


FIG. 6.—RETORT NO. 4 CUT OPEN.

of retorts in the usual type of furnace, which average about 30 days.

Joplin ore was used in these experiments with 4 per cent to 5 per cent iron.

The average loss was 8 per cent. This is an excellent showing considering the small size of the furnace and the difficulty in maintaining the correct temperature.

After the retorts were removed from the furnace a new set was put in, which were in continuous operation for 154 days. The retorts were still in good condition at this time.

St. Louis, Mo.

Pure radium has been prepared by Madame Curie, who announced her success in September, 1910. The production of radium salts has not increased greatly, and the cost of obtaining them, either by purchase or loan, is still excessively high. Prices are quoted at about $100 per milligram. Radium is loaned in Paris for scientific purposes at a cost of $100 to $200 per day for 100 milligrams. The production in France in 1910 was 1.92 grams, and was sold for about $150,000.

Recent Progress in Calorimetry.

The Real vs. the Supposed Errors.

By Walter P. White.

In pure science and in its technical applications an important place belongs to calorimetry, the measurement of quantities of heat, of heat energy, of heat as distinguished from temperature. It is well known that nearly all large users of coal or other fuel no longer buy by weight alone, but by fuel value, thus actually paying for heat, the thing they are really after. The fuel testing made necessary by this practice is probably the most extensive and familiar use of the calorimeter, but nearly every physical or chemical process is accompanied by a heat effect whose determination is necessary for complete knowledge, and heats of fusion and vaporization, variations of specific heat with temperature and pressure, and heat effects accompanying solution, dilution and other chemical processes occupy prominent places in the literature of their respective subjects.

In spite of its importance, calorimetry has been singularly backward in development and has long been considered to be a subject wherein high accuracy was probably unattainable. Within the last five years, however, a great improvement has taken place. A careful study of the various sources of error—which in itself furnishes a good example of the value of systematic scientific methods—has shown that previous practice was largely based on notions and estimates widely at variance with fact and has rendered possible a progress based on knowledge. In considerable part, however, it is only fair to say the improvement has been due merely to the application of a little common sense to practices which rested on nothing but the thoughtless acceptance of traditions handed down from earlier and cruder days. The net result is that the measurement of heat in a calorimeter may now usually be made more accurate than the necessary associated determinations. For instance: Weighing is well recognized as one of the most accurate of physical determinations. In determining the heat of combustion of many substances the error may now be no greater than the unavoidable error in weighing the sample.

The calorimetric method in which this progress has been most conspicuous is the most common and familiar one, often known as the method of mixtures; it alone will be regarded in the present paper.

In principle the mixtures method is exceedingly simple. Heat is added to a vessel of water (or other fluid) and the resulting temperature rise is measured. The heat is then given as simply the product of temperature rise by the heat capacity of the water and calorimeter. The numerous complications occurring in practice all arise from one cause—the loss of heat which begins as soon as the water is heated. This (1) necessitates a correction, the "cooling correction," for the heat lost; (2) requires experimental arrangements which increase the difficulty of dealing with evaporation; (3) calls for stirring, and the stirring, if effective, produces appreciable heating; (4) makes it necessary to take observations of time, since the temperatures are continually changing; (5) gives opportunity for error if in the various changes any parts of the system fail to follow the change quickly enough; opportunity, that is, for "lag" errors of various sorts.

Of these complications the cooling correction has generally been counted the chief. Omitting details for the present, the principle of the cooling correction is as follows: The heat loss can be found if we know two things—the temperature of the calorimeter, and the way in which the heat loss depends on that temperature, or the "cooling constant," as it is called. Observations on the calorimeter temperature are easily made during the heat transfer. Observations of the cooling rate, made before and after the transfer period, define very accurately the "cooling constant" for the particular set up and the particular half-hour of the experiment. A simple calculation then gives the correction.

This procedure, in the form just outlined, evidently assumes that the environing temperature remains constant, since this also affects the cooling of the calorimeter. To promote this constancy the calorimeter has usually been surrounded by a water jacket, except at the top, which is left exposed to the air of the room. This arrangement is evidently far from perfect, for the jacket temperature also may vary, and the influence of the room is not wholly excluded. And, in fact, the changes may easily be of sufficient amount to be a source of error. The remedy, however, is easy and simple; it is to make the jacket surround the calorimeter completely and then measure the temperature of the jacket and allow for any changes. This gives directly the difference of calorimeter and jacket, which is really the quantity on which the heat loss directly depends. This method is one of the recent improvements and has been adopted independently by several different workers. If certain slight lag effects are also guarded against, the cooling then depends on simple, definite conditions which are thoroughly known. Since the whole correction is only a few per cent of the total heat quantity measured, there would seem to be no reason why it should cause appreciable error.

There has been, nevertheless, a widespread opinion that the cooling correction error is the largest in calorimetry, and that this condition is almost unavoidable. Indeed, this belief has probably gone far to discourage attempts at reducing the other errors. No adequate experimental evidence appears ever to have been offered in favor of this notion as to the hopelessness of the cooling correction, and lately decisive evidence has been obtained of its falsity. For instance: In a recent series of over 100 calorimetric determinations at the Geophysical Laboratory the time, the initial temperature, etc., were varied among the various observations so as to make the relative magnitude of the cooling correction vary within wide limits. Although the average accidental error was only from 20 to 40 parts per 1,000,000, and most of that was evidently due to the temperature measurement, yet there was no sign of any error due to the cooling correction. With proper precautions, therefore, the cooling correction error is not only not large, it is imperceptible. The hopelessly uncertain cooling correction is a myth.

With *evaporation* the case is altogether different, and irregular evaporation is probably responsible for much of the irregularity attributed to the cooling correction. There are cases where, in work intended to be specially accurate, the published data show that the heat loss by evaporation was often quite as great as that due to the regular cooling. Evaporation depends on a condition, the saturation of the air, which is at once very variable and hard to determine. Hence, as a rule, the only satisfactory way to regulate evaporation is to prevent it. Fortunately, this is not difficult.

Two ways are available. One, of course, is simply to cover the calorimeter tightly. The other is to cover it loosely, or not at all, but to keep it below the temperature of the jacket. In that case the air within the calorimeter chamber very soon becomes saturated at the temperature of the coolest body in it, the calorimeter, and evaporation then ceases. Of course, there will be a little more evaporation when the calorimeter is warmed, but the total amount of this can be kept small by restricting the size of the calorimeter chamber. For instance, if a 2-liter calorimeter is heated 2 deg. and the air space around it is only 1 liter, the heat abstracted by the extra evaporation will not be far from three calories, or about 0.8 per mille of the total heat measured. If a correction is applied for this, the *error*—that is, the uncertainty—will be only a few tenths per mille. Of course, this assumes that the chamber is well inclosed all the time, that its walls are thoroughly dry, and that care has been taken to keep well above the dew point. The evaporation of small drops of water from the *outside* of the calorimeter is especially productive of error on account of its great uncertainty. The remedy is obvious.

The *heat produced by stirring* is much less serious than the effect of evaporation, but may still cause appreciable error if improperly treated. This is because the heat production in-

creases at least as fast as the third power of the speed of stirring, so that a stirring rate only a little in excess of what is necessary may cause a rather large surplus of heat. Previous practice has almost universally been widely astray in this matter. For instance, Richards and Burgess have shown that stirring has nearly always been insufficient. The opinion has also been common that reciprocating stirrers, with horizontal vanes moved up and down, give less heat for equal efficiency than screw propellers. The opposite is the case. The greatest objection to the reciprocating stirrer, however, is the increased evaporation it usually causes.

The object of stirring, of course, is to secure inside the calorimeter uniformity of temperature; just how great this uniformity must be will depend on the accuracy sought. The heat of stirring will not be a source of error so long as it is constant, for it can then be allowed for, but its variations, due to variation in the speed of stirring, will cause error, and a variation of 1 per cent in the rate will cause a variation of 3 per cent in the heat. Of course, by regulating the speed closely enough it will nearly always be possible to avoid appreciable heating error, but what is wanted practically is to secure uniformity of temperature without making careful regulation necessary. The ideal calorimeter would be accompanied by a statement of the proper speed of stirring for any given error due to uneven temperature, and of the heat production at that speed. The allowable variation in speed could then be found at once. As it is, the user will generally have to determine these quantities for himself.

(1) The greatest error from lack of uniformity in temperature is usually due to the lag of the surface of the calorimeter behind the center while heat is being added; this is easily determined if a thermo-element can be used. (2) The heat of stirring can easily be found by observing for alternate periods of 10 minutes (say) with the stirrer first in action, then still, etc., and the calorimeter very near the temperature of the jacket.

In one calorimeter (with propeller) of 1½ liter capacity, stirring, certainly effective enough to reduce the error from uneven temperature below 0.025 per mille, raised the temperature about 0.001° per minute. This would demand a speed of stirring constant to 6 per cent for an accuracy of 1 per mille with a 2° rise and 10-minute interval. This result may be useful as a rough guide for those who are not in a position to make thermoelectric observations, but who can, of course, determine the heat of stirring.

In this case the error from heating would ordinarily greatly exceed that from uneven temperature, hence slower stirring would usually be better. With stirring one-third as fast the error from uneven temperature would still probably not exceed 0.08 per mille (for the same time and temperature intervals), and the error from heating would be no larger than this, even with a speed varying 10 per cent. On the other hand, if the speed were to be doubled, the heating would be 0.1° in 10 minutes, and the accuracy would be very limited, even with a stirring speed regulated to 1 per cent. This shows very clearly the disadvantage of a speed even a little too great. It should be added that the *error* from uneven temperature here is (arbitrarily) taken as one-sixth the actual temperature difference. The calorimeter here referred to was especially designed for facility in stirring; there were no dead spaces where the water could linger, and the flow of the circulating water was not throttled at any point.

The timing of the observations should never cause appreciable error. For, let K be the cooling factor of the calorimeter—that is, let the calorimeter cool in one minute the fraction K parts of its excess temperature above the jacket. K, for ordinary calorimeters, is not very far from 0.003. Hence a calorimeter, starting at jacket temperature and heated by any amount, will cool about 1/330 of that in each minute, or 50 1,000,000 of it in one second. As the timing can easily be done to half a second, the probable error will be less than 25 parts per 1,000,000. (If the calorimeter does not start from the jacket temperature the error will not be very different.)

The relation between time error and proportional temperature error thus depends on the cooling rate of the calorimeter. This relation is important, for it also holds for all lags.

The *lag effects* are rather complicated, but fortunately the only serious ones are easily preventable.

The one most prominently advertised thus far is the thermometer lag. A mercury thermometer will often lag ten seconds or more behind the fluid surrounding it, thus making its readings higher than the real temperature of a cooling calorimeter. Ten seconds, we have just seen, corresponds to a temperature error of 0.5 per mille. The thermometer lag, however, has lately been shown to involve no error, because it enters both in the cooling correction determination and in the final temperature, and enters oppositely in the two, so that *if the same thermometer is used throughout* the two opposite effects are equal and cancel each other. The simplest way to demonstrate this is to consider that a uniform lag on *all* the readings amounts to the same thing as reading them correctly, but a few seconds earlier each time—which, of course, would introduce no error. (For a more complete demonstration those interested may be referred to several articles on calorimetry in the *Physical Review* for November and December, 1910. These articles also supplement the present paper in other respects.)

The unavoidable lag effects are (1) the lag of the cooling itself: as the calorimeter is heated it takes a little time (about two seconds) for the air currents to get started around it. (2) The very trifling lag of the projecting metal parts of the calorimeter. (3) The lag of the surface behind the center as the calorimeter is heated. This is due to the time required for the heat to pass outward through the water, and depends on the rate of stirring. It may be, under proper conditions, from three to nine seconds. The total effect of these unavoidable lags, therefore, may be from five to eleven seconds, or from 0.25 to 0.5 part per mille. But these lags also have this property: their effect is independent of the manner of heating, whether sudden or slow, but is always proportional to the temperature rise. Their effect, therefore, is precisely equivalent to that of a slight change in the heat capacity of the calorimeter. Hence, if a calorimeter has been calibrated, and is not afterward changed, the error due to these lags will be altogether inappreciable, otherwise results will (as far as their influence is concerned) be consistent, but subject to a systematic error about equal to the present uncertainty in the specific heat of water.

Two remaining effects are perhaps better regarded not as lags, but rather as cases of variable environing temperature. These are the effects of non-metallic supports or projections, and of covers not touching the calorimetric fluid. Such bodies warm up gradually after the calorimeter is heated, and during all that time the heat flow into them is changing in an unknown way. For instance, calorimeters have been known to be set on a solid plate of hard rubber, a gratuitous introduction of error so needless as to be quite inexcusable. A safe rule is to make all non-metallic supports so small that their total heat capacity is negligible, which is easily done. The stem of a glass thermometer, however, may introduce a slight error through its varying temperature. Uncertainty in the temperature of the cover can be avoided by depressing the center of the cover so that a considerable part of it touches the water.

The determinations at the Geophysical Laboratory, mentioned above, were, of course, subject to error from evaporation, stirring and lag. Hence those results show that accidental errors from *all* these causes taken together may be made quite inappreciable.

The chief *unavoidable calorimetric error*, then, is just where it would be if a perfect heat insulator existed—that is, in the temperature measurement.

The rise of temperature employed in calorimeters is seldom over 4 deg., and often less than 2 deg. The most sensitive mer-

cury thermometers are rarely read closer than to 0.001 deg. Now, even with perfect judgment on the observer's part, and perfect behavior of the instrument, there is evidently always a possible accidental error of half the smallest unit recorded. But two to four accurate thermometric readings enter into a single determination. Hence, the unavoidable accidental thermometric error may be as high as 1 per mille, altogether aside from imperfections in the instrument or the observations, which will ordinarily make the error still larger.

Two remedies are possible. The first is to increase the *precision* of the thermometric measurements. This has hitherto called for electric thermometers. These, of course, are complicated in comparison with mercury thermometers, though they also have some decided advantages besides the increased precision. There is, however, a second and very easy method of reducing the thermometric error, and that is to increase the temperature interval by diminishing the amount of water heated or by increasing the amount of heat. This method runs contrary to a generally accepted opinion which insists that small temperature intervals are necessary for accuracy. But the reasoning on which this opinion is based appears upon examination to be rather unsatisfactory. That reasoning, briefly, is as follows: The heat loss from a calorimeter is not proportional to the excess of temperature above the environment, but increases more rapidly than it. If heat loss is plotted against temperature difference the result is a line which curves upward. Now this curve, like all curves, can be made to differ as little as we please from a straight line by dealing with a sufficiently short portion of it. And so, by limiting sufficiently the temperature rise in the calorimeter, it is possible to use a linear cooling law—that is, to take the cooling as proportional to temperature difference, which, of course, simplifies calculations somewhat. Hence a small enough interval should be used to make the cooling law linear—say, from 2° to 4°.

It is evident that this reasoning does not indicate that actual error will arise from the use of a large interval unless the observer insists on coupling with it a computation based on the linear law appropriate to a small interval. But he is not in the least obliged to do this, and there is no question that the proper expression will represent a large interval quite as well as the approximate linear law does a small one, thought not quite so simply. Hence the only considerable advantage of the short interval is a saving in labor, and while this may sometimes be worth while it is, of course, a trifling consideration whenever accuracy is of importance. Moreover, the extra labor involved in cooling computations for large intervals may, by a method to be described later, be made to average less than a minute per determination, and the error from improperly assuming a linear relation for an interval of 8° is no greater than that of the best calorimetric determinations yet made with mercury thermometers. Hence the generally accepted view that small intervals are conducive to accuracy and generally advantageous is a fallacy. (The small interval may, however, be advantageous in cases where the thermometric accuracy is more than sufficient for the object in hand.)

With a mercury thermometer the gain in accuracy from using a large interval would probably not be proportional to the increase of the interval. Just how great the gain would be in any given case is perhaps a little uncertain at present. With electrical thermometers the gain at first is practically proportional to the increase in interval and even for a 10° interval and with the best instruments the error in the thermometer is probably considerably larger than that in the calorimeter itself.

The increase of the interval meets with an obstacle in the important department of combustion calorimetry, for the water must circulate freely about the bomb and hence its amount often cannot be readily reduced. But it does not seem impossible to get around this difficulty in a satisfactory way. To take one possible method out of several: If the bomb itself were used as the stirrer, being raised and lowered by a wire, the stirring of even a small amount of water would be more effective than that of a larger amount by the customary reciprocating stirrer, while the great drawback of the reciprocating stirrer, evaporation, could in this case be easily controlled.

The foregoing article has pointed out that the cooling correction, instead of being the largest, is easily made the least of the sources of calorimetric error, that evaporation, while much more serious, can also be controlled without difficulty, as can the heat produced by the stirrer. The way to control evaporation is to practically prevent it. With the stirrer the essential thing is to find the proper speed. Lag effects, though numerous and complicated, may easily be made to produce negligible errors. The total inaccuracy of the calorimeter, therefore, is much less than the usual error of the temperature measurement. This error can be greatly reduced, though at the cost of considerable complications, by using electrical thermometers. It can be very easily, though less effectively, reduced by increasing the temperature interval through which the calorimeter is heated. The general opinion that small intervals are generally advantageous or conducive to accuracy is a fallacy.

In a subsequent article there will be considered the practical application of these conclusions to the various details of a calorimetric determination.

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A Commercial Nitrogen Plant.

By Albert Alexander Somerville.

The air we breathe is a mixture of nitrogen and oxygen. The latter is a supporter of respiration. It is also a supporter of combustion. Nitrogen is just the opposite. It will not support combustion nor is it itself combustible under ordinary conditions. When studying the properties of a substance at elevated temperatures it is desirable to prevent that substance from combining with oxygen to form an oxide. This can most readily be done by working the substance in a vacuum or an atmosphere of nitrogen or hydrogen in order to exclude the oxygen.

Obtaining a vacuum at room temperature, and keeping it while the temperature is being raised in a furnace, are two things of different degrees of difficulty. A containing vessel through which gases will not pass at high temperatures when the pressures inside and outside are different is a rare thing. Someone has said that a vacuum is the cheapest thing you can produce, but it requires a lot of trouble to retain it.

If a metal is to be annealed it must be heated to a considerable temperature. A partial vacuum may be obtained and so only a small amount of oxygen allowed to unite with the metal to form an oxide. If it is desired to completely prevent oxidation then it is better to fill the containing vessel with some other gas so that the pressure inside and out will be equal and, therefore, reduce the tendency for the oxygen or the air to enter. Or, still better, if the inside pressure can be maintained at a value slightly higher than that of an atmosphere, thus causing a slow leakage outward, the chance for oxygen to enter is very small.

Two gases that may be used for filling the vessel are hydrogen and nitrogen.

Hydrogen is readily obtained by the decomposition of water, H_2O, by an electrical battery into hydrogen and oxygen, collecting the two gases at the two poles of the battery. Due to the low specific gravity of hydrogen a tube may be kept filled with it to the exclusion of other gases simply by inverting the tube and admitting the hydrogen at the lower open end. The two objections to its use are that hydrogen passes through the walls of a heated vessel about the easiest of any gas, and also that in some proportions with air it forms a dangerous explosive mixture.

On the other hand, nitrogen is readily obtainable from the air, of which it forms about 78 per cent by volume. It is

neither combustible under ordinary conditions of practice, such as are here considered, nor a supporter of combustion, nor does it enter readily into combustion with any other element. Also it does not so readily pass through the walls of a containing vessel.

Nitrogen is obtained by separating it from the oxygen of the air. This is done by causing the oxygen to unite with something to form an oxide and so leave nothing that will support combustion or oxidation. It may be noted that the air consists of about 99 per cent nitrogen and oxygen and the remaining 1 per cent consists of a group of gases known as the

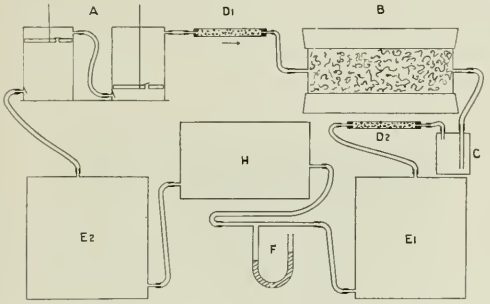


FIG. 1.—DIAGRAM OF APPARATUS FOR SEPARATING NITROGEN FROM AIR.

argon groups, none of which support combustion, and also one other, namely, carbon dioxide.

The oxygen of the air may readily be made to unite with burning phosphorus and so leave nitrogen, or air may be passed over hot copper with which the oxygen unites to form copper-oxide and thus leaves only nitrogen in which any other metal may be heated without oxidizing. The equation representing the chemical change is simply

$$Cu + O = CuO.$$

A nitrogen producing plant may be made on this principle that is quite efficient. The highest efficiency is reached by having a cyclic process and using over and over again the same product. The same is true of any kind of a system, for instance, in a refrigeration plant the same ammonia is used over and over again.

In a nitrogen plant a cyclic system may also be used consisting of a motor-driven pump, a furnace containing metallic copper, storage tanks, drying agents, pressure indicators, meters to measure rate of flow, and lastly the station at which it is desired to use the nitrogen, all connected in series so that the same gas is continually flowing through the same circuit. Such a system probably has the highest efficiency possible.

The diagram, Fig. 1, shows the general scheme. A is a pump (two cylinders in series), B a furnace containing copper, electrically heated, C a wash bottle, indicating flow of gas, D_1 and D_2 dryers, E_1 and E_2 storage tanks, F a pressure gage filled with mercury, and H the station where the nitrogen is used. The capacity of the storage tanks and conduits is nearly a fixed quantity and may be made anything desired.

The pump may be any of several motor-driven types for the movement of gas, and the rate of flow of the gas may be regulated either by varying the speed of the motor and pump or, what is better, by running these at normal speed and adjusting the partially closed stopcocks.

If there is an oil immersion used in the pump A it is well to follow this by a drying tube D, in order to remove any moisture from the air before it enters the furnace. This may consist simply of a large glass tube filled with calcium chloride or phosphorus pentoxide.

From this drying tube the air passes into the furnace and over the heated copper. Here the copper is oxidized and the oxygen of the air unites with it to form copper oxide. The gas that remains is principally nitrogen and will not support combustion or oxidation. The furnace is made by filling an iron pipe 3 in. in diameter and 2 ft. in length with copper shavings or droppings, capping the ends of this pipe, tapping small pipes into these caps for an inlet and outlet and providing an electrical heating coil around this receptacle.

This electrical heater may be made of wire or ribbon wrapped around the iron pipe and insulated from it either by a roll of asbestos paper or by special kinds of fireclay. The wire may be covered on the outside with asbestos or mineral wool for thermal insulation. This furnace is run at a temperature somewhat above visible red heat, 600° C. to 800° C., and if the air is not passed through it too rapidly practically all the oxygen is removed on the first trip around the circuit.

In order to determine the rate at which the gas is flowing a meter may be put in circuit, or if only a general indication is desired a wash bottle C may be used to indicate, by its bubbling, the general rate of flow and in this wash bottle something may be used, such as pyrogallic acid and potassium hydroxide, to remove any remaining traces of oxygen that may have passed through the furnace.

This wash bottle may be followed by another dryer D_2 in order to catch any liquid or vapor that may be carried over from the bottle and then connect with a storage tank E_1 and pressure gauge F. The storage tank may vary from the size of a bottle to that of a gasometer as desired. Since the pressure is generally only slightly above an atmosphere an ordinary mercury manometer may be used to indicate pressure and will indicate any large leaks.

The gas is now available for use and may be run into any station or place where it is desired. Presumably this would be a second furnace where it is desired to heat some substance in which oxidation is to be prevented. In the case of the writer he has used it in connection with the measurement of the electrical resistance of about 40 materials up to temperatures approximately 1000° C.

The nitrogen is not allowed to escape the air from this furnace, but instead is carried to another storage tank and eventually back to the pump at the starting point. Since the plant was filled with pure air at the start and about 20 per cent by volume has been removed to form copper oxide, the pressure would accordingly fall making a tendency for air to leak in from the outside in order to equalize the pressures. This may be remedied by providing an intake right at the pump so that any new air that comes into the circuit must at once pass over the heated copper and so have its oxygen removed. Finally

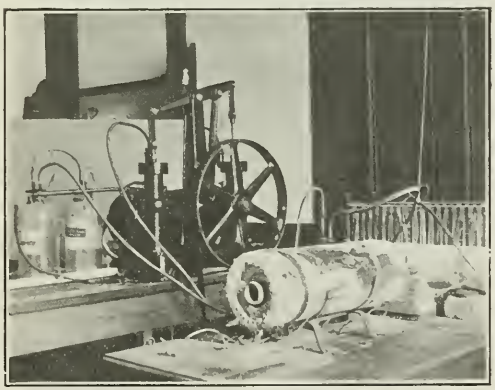


FIG. 2.—VIEW OF APPARATUS.

the circuit becomes thus practically filled with pure nitrogen.

All connections are made with pressure tubing with joints well covered with shellac.

Due to keeping the inside pressure of the circuit above one atmosphere there are usually some small leaks so that a fresh supply of air must be pumped in occasionally. In order to remove the oxygen from this copper must be changed into copper oxide and so the supply of metallic copper is finally depleted.

In order to convert the copper oxide back to metallic copper it is only necessary to disconnect the furnace from the remainder of the circuit and run illuminating gas through the furnace while it is hot. The hydrogen in the gas unites with the oxygen of the copper oxide to form water and leaves metallic copper and carbon dioxide. The equation representing it is as follows:

$$4\,CuO + CH_4 = 4\,Cu + 2\,H_2O + CO_2.$$

This reduction is easily accomplished and the furnace is then connected into the nitrogen circuit again.

Cornell University.

Evaporators and Vacuum Pans.

By B. Viola, M.E.,

Member American Society of Mechanical Engineers.

Evaporation for the purpose of concentrating liquid plays a most important part in the sugar and the chemical industries. This process is accomplished in any of three different ways: viz., by heating the liquid by the combustion of fuel, by heating the liquid with indirect steam heat, and lastly, by indirect steam, with simultaneous reduction of atmospheric pressure upon the surface of the heated liquid, lowering the boiling point.

Any liquid exposed to the atmosphere will evaporate more or less rapidly at ordinary temperature. The normal pressure of the atmosphere at sea level is equal to that exercised by a column of mercury 29.92 in. equal to a pressure of 14.707 lb.

FIG. 1.—SUGAR MANUFACTURE IN SICILY IN 1570. (FROM THE "NOVA REPERTA" COLLECTION OF JOANN STRADANUS.)

per square inch. At this pressure water boils at a temperature of 212° Fahr. If the pressure be decreased the boiling point of water or any liquid falls and it is due to this phenomenon that multiple-effect evaporating apparatus can be constructed in which 1 lb. of steam may evaporate in practice up to 5 lb. or 6 lb. of water. The evaporation increases with rising temperature; it increases until evaporation takes place not only on the surface, but all through the bulk of the liquid when the boiling point is reached. The temperature of such exposed liquid cannot increase while the liquid is boiling, as any addition of heat will but accelerate the process of evaporation.

The heat of evaporation can be determined for any liquid. The total heat of steam is $t + t_1$, where t is the amount of heat in heat units, or thermal units, which must be communicated to the liquid in order to accomplish the boiling of it. t_1 is the amount of heat which evaporates the boiling liquid.

The heat required to raise the temperature of 1 lb. of water 1° Fahr. is called the British thermal unit and to convert 1 lb. of water at 212° Fahr. into steam at 212° requires 966 b.t.u. This item is called heat of evaporation. The quicker this heat is added to the boiling liquid the quicker the evaporation will take place.

Different liquids have different boiling points and are also influenced by the presence of bodies in solution.

The amount of water evaporated in a unit of time is in direct proportion to the heating surface with equal temperature.

Increase of temperature of the heating medium has the same effect as increase of heating surface—that is, the amount of water evaporated in the time unit is directly proportional to the temperature of the heating surface, its dimensions remaining

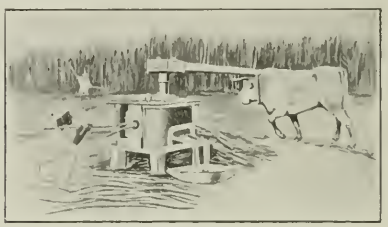


FIG. 2.—EARLY METHOD OF EVAPORATION.

equal. It is by far easier to enlarge the heating surface, hence this is always attempted in the construction of plants.

The earliest systems of evaporation were those on open fires in flat or cylindrical shaped open pans (Figs. 1 and 2). These systems can still be found for boiling sugar juices in India and in the West Indies, especially where families boil their own sugar from the juice of sugarcane. We find the first improvements in evaporators at the beginning of the nineteenth century when Dubrunfaut constructed his rocking-pan. This has a flat bottom which is heated by coal or wood fire and has a bill-shaped mouth for emptying the same. In Fig. 3 *A* is the pan, *a* the mouth, *b* an axle placed in the bearings, *c* is a moving and *d* a fixed roller for raising the pan. *B* is the oven, *G* the fire grates and *b* the fire door.

Halette's construction (Figs. 4 and 5) was a considerable improvement in evaporators. In this, *A* is a copper tank with a half-round front piece *B* with valve *a* in the bottom. This evaporator rests by two buttons on two iron plates *b* which are fastened on the wooden support *c*. At *e* is an iron leg which is turning around the axle *f* like a leg of a table. By loosening the bolt *d* the bolt plates could be turned aside and the tank could be lowered by turning the leg *e* around the axle. On the rod *g* hangs the rost-shaped heating apparatus, consisting of two prisms *hh* with 24 cone-shaped holes in them where the 24 copper tubes are tightened by bolts *i*. The steam enters through the pipes *ll* and from there passes in the direction of the arrows to the copper heating tubes, holes being drilled in the prisms *hh* between the pipes for that purpose. The use of this apparatus was not very advantageous, as the steam entering the heating pipes condensed too quickly and the pipes filled with water, which was difficult to drive off because of the high pressure required, which often burst the pipes.

Neither the evaporating system on open fires nor those with steam pipes, placed in the bottom of open kettles, could give satisfactory results, especially in the manufacture of sugar. First, the evaporation was too slow; a great amount of sugar was destroyed on account of the high boiling point and then the heating capacity of the fuel and steam was very poor. Therefore, some better method was necessary to increase the evaporative capacities of the kettles.

In order to consider everything connected with the building of such evaporators we can say that:

a. Besides the heating surface, the heat-conducting power of the material, of which the heating surface is composed, is very essential. According to Wiedemann and Franz, in Poggendorf's *Annalen*, the relative conductivity of silver is 1000; gold, 532; copper, 736; iron, 119, and lead, 85. Silver would, therefore, furnish the most efficient heating surface, but its high price excludes it from practical use for this purpose. We generally find copper employed for sugar, malt, tannin, etc., while lead or lead-lined copper and iron are used for the evaporation of acids. Further,

b. The thickness of the heating surface decreases the conductivity of heat transmission. However, the thickness employed for copper in practice is sensibly constant.

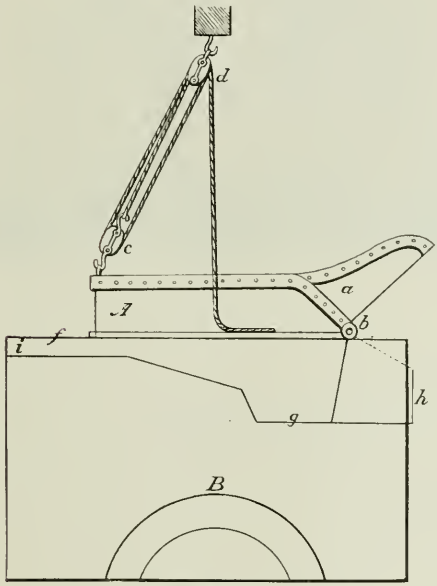


FIG. 3.—DUBRUNFAT ROCKING PAN

c. The transmission of heat is directly proportioned to the mean difference of temperature.

d. Thick or viscid liquids acquire heat more slowly than water or thin liquors and in such cases the coefficient of transmission is often only 0.5 or 0.2 of the coefficient of water.

e. The transference of heat is reduced to an undeterminable degree by incrustations with crystals or scales.

f. When a continuous change exists in the flow the quicker liquids or steam are dirven through the heating pipes the quicker will be the heat transmission. The shorter and smaller the diameter of the steam-heating pipe the more efficient is the heat transmission.

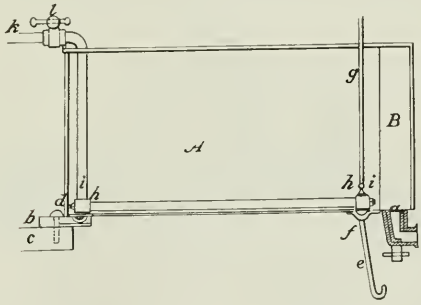
g. The transmission of heat is affected also by the position of the heating tubes.

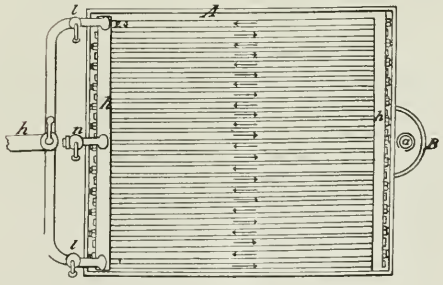
With vertical tubes the condensed water is always running down on the whole surface and prevents the conductivity of heat because the steam is not in intimate touch with the heating surface; while with horizontal tubes the condensed water drops off quickly and the steam can effect its full heating capacity. Boiling depends not only on the material and area of the heating surface, but also on the pressure resting upon the surface of the liquid which is to be evaporated. The lower the pressure the less resistance the steam bubbles find in rising to the surface and so, of course, boiling can be reached by a lower temperature. This is always the case with evaporators and vacuum pans, where the boiling of the liquids is done under a partial vacuum.

The vacuum in an evaporator is always given in inches of mercury. Thus a vacuum of 25 in. means that the excess of atmospheric pressure over the pressure in the effect would support a column of mercury 25 in. high; hence, expressed as a pressure, a vacuum of 25 in. means a pressure of 29.92 — 25 in. = 4.92 in. = 2.42 lb. per square inch.

The first apparatus for a reduced pressure was built in 1813 by Howard (Fig. 6) and is shown in section according to J. Foster. *A* is a boiler for boiling the liquid. It is formed of two copper pans strongly bolted together and has a double bottom where the vapor is supplied by the pipe *C; D*, the condensed water pipe to which is attached the air cock; *E*, dome in which the vapors are collected and the syrup which may be mechanically drawn in with the vapor falls again into the vessel; *G*, the condensing vessel to which a pipe is attached and communicates with the air pump; this pump maintains the vacuum, but is not shown in the plan.

The cold water enters the space *C* through the opening *H* and by moving the handle *I*, the valve *K* and the piston *L* are raised. *F*, the vapor pipe from the pan to the condenser. This is used for returning the liquid drawn in by the vapors into the space *O*, from where it is easily removed by the cock *R; M* is the cold-water feed pipe; *N*, a proof-stick for taking a proof of the syrup in the vessel without opening communication between the interior and the air outside; *Q*, thermometer which indicates the temperature of the syrup.





FIGS. 4 AND 5.—HALETTE EVAPORATOR

This apparatus spread among the English colonies and on account of its advantages it was possible to get 1/6 larger results than with the open pans. The temperature difference between condensed water and syrup was 9° to 12° and the interior pressure about 50/76 and 55/76 atmosphere. For the condensing of 500 kg of vapor 10 cu. m to 11 cu. m of injection water was used and reached a temperature of 30° C.

This had several disadvantages, however; it was not possible to watch the boiling, because the vessel was closed; the dis-

charge of the syrup was very slow and needed too much power and steam. In later years the evaporating effect was improved by placing a serpentine inside and also glasses for seeing inside the vessel.

After Howard's apparatus was built Roth constructed one in France which had a jacket also and a serpentine inside (Fig. 7). The steam became condensed by a heavy water injection. The condensed water receiver had to be large enough to hold the whole quantity of one charge and was discharged after each operation. A vacuum was made before the operation started through a stream of steam (Fig. 7).

The boiling vessel *A* is similar to Howard's, but it contains a serpentine in which the steam circulates. *D* is a vessel made of iron for condensing the vapor. It is fitted with a vacuum gage *m* and a water gage *n*. *E*, the steam pipe communicating with the boiler; *F*, *G*, *H*, cocks for admitting the steam to the double bottom, the serpentine and the vessel. F_1 and G_1, cocks for the condensed water flowing from the double bottom and the serpentine; *i*, butter cock to prevent the mass from becoming frothy; *K*, syrup supply cock and pipe; *L*, outlet cock for the boiled syrup which is discharged into the vessel *O*; *M*, the vapor pipe to the condenser; *N*, metallic perforated plates, through which the cold water runs in very small streamlets and through which the vapor passes; *P*, injection cock connected to the pipe *S* which supplies the cold water; *Q*, cold-water supply cistern; *R*, cock for discharging the hot water and the air at the end of each stroke.

It is operated as follows: The vesel being empty the cocks

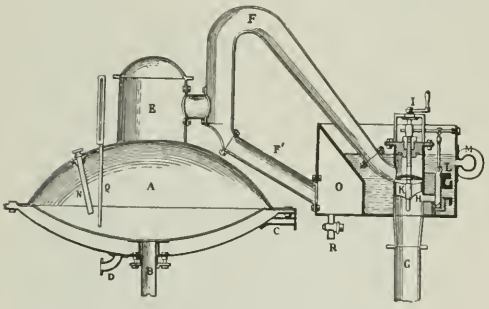


FIG. 6.—HOWARD EVAPORATOR

H and *R* are opened and the steam rushed into the vessel and into the condenser. A mixture of air and steam escapes by the cock *R* and after a few minutes, when it is suposed that all the air has been driven out, these cocks are shut and the cock *K* is opened to admit the syrup.

At first the liquid rises slowly with the progressive condensation of the vapor in the vessel caused by the cooling of the outer case. Afterward, when a portion of the syrup has been supplied into the vessel, it rises very quickly on account of the rapid condensation which its entrance causes.

When the syrup in the cistern is drawn out to a certain level the cock *K* is shut and *F* and *G* opened to admit the steam into the double bottom and the serpentine. The condensed water cocks are also opened and in a few moments the injection cock *P*. In a short time a proof is taken of the syrup and when it has arrived at the required point the injection cock is closed; the syrup is hot and when it reaches 80° or 90° the cock *L* is opened, also the air cock and the boiled syrup runs out. In order to begin a new operation, the cock *R* has to be opened to allow the hot water to run out and steam is again admitted to the vessel to drive out the air.

The water consumption of Howard's and Roth's apparatus should be alike and otherwise Roth's has the same disadvantages as Howard's. The vacuum could never be kept up equally, as the cooling water brought in too much air and some gases were also always discharged from the syrup mass.

For the evaporation of larger quantities of syrup Pelletan has built an improved apparatus. In Figs. 8, 9 and 10 *AA* is a half-cylindrical kettle 20 ft. long and 2 ft. in diameter and is placed in the jacket *BB*; it is covered with the top *CC* and rests on legs *VV*; *D*, a condenser, 15 in. in diameter and 6 ft. high with a valve and the record plate *I*, for regulating the cooling water, which condenses half of the steam coming from the kettle; *K*, another valve for the discharge of condensed water or to connect it with a pump; *H*, the steam inlet valve for a

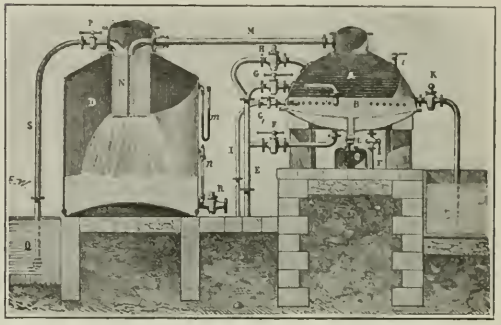
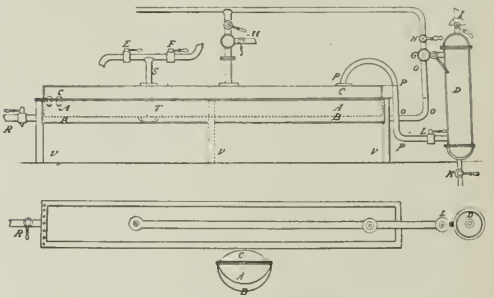


FIG. 7.—ROTH EVAPORATOR

three atmosphere steam pressure in the globe *G* and from there through pipes *OO* in the jacket for the heating of the kettle.

The steam by passing the condenser *D* sucks the air out and drives it into the jacket and passes through valve *R* when the liquor reaches the cooking point. All the steam passes through pipe *PP* in the condenser and the part not condensed is carried along by the steam stream into the jacket. In this way the vapors of the kettle and the live steam together make the evaporating heat. By this method Pelleton was able to keep up a vacuum of 20 in.

Valve *L* cuts off the evaporator from the condenser and is closed when the kettle has to be discharged. With the steam valve *M* the evaporated liquid is pressed through pipe *S*, which



FIGS. 8, 9, 10.—PELLETON EVAPORATOR

reaches down to the bottom. The same pipe can be used to fill the kettle. Valve *E* is for the charging and valve *F* for the discharging of the kettle. The liquor was boiled here by 65° C. and the heating steam was 100° C. The quantity of cooling water used was about five times that of the evaporated liquid. This was the first apparatus where juice steam was used for helping in the evaporation and was first practically utilized in the United States by Rillieux in his construction, as we will see later on.

Before going further I wish to mention the capacities of the heating surfaces of the different evaporators or effects. I do

not want to figure the wearysome formula in this direction, but mention briefly according to practical experience the heating effect of a vacuum apparatus per square foot and per hour of a vacuum at 25½ in.

With exhaust steam at 230° Fahr. (20 lb.) from water, 2.5—2.66 gal.

With exhaust steam at 230° Fahr. (20 lb.), thin liquids, 1.5—1.65 gal.

With exhaust steam at 230° Fahr. (20 lb.), thick liquids, 0.73—1 gal.

With high-pressure steam, 266° Fahr (40 lb.), from water, 2.8—3 gal.

With high-pressure steam, 266° Fahr. (40 lb.), thin liquids, 2—2.5 gal.

With high-pressure steam, 266° Fahr. (40 lb.), thick liquids, 1—1.3 gal.

As stated above, if the normal pressure of 14.707 lb. per square inch can be decreased, the boiling point of water or any liquid can be lowered and this is the principle on which multiple evaporating apparatus is constructed. If steam at 212° Fahr. is condensed to water at 180° Fahr. it gives a certain quantity of heat and this is the difference between the total heat of steam at 212° Fahr.

This heat can raise the temperature to evaporate water which is boiling at 180° Fahr. under a certain pressure; the steam given up at this temperature can be used again to boil more water under a lower pressure and this way can be repeated many times.

The following system of construction can be mentioned:

a. Evaporators with horizontal heating pipes.

b. Evaporators with vertical heating pipes.

c. Film evaporators.

In constructing evaporating apparatus (and especially those with horizontal and vertical heating tubes or vacuum pans with heating coils) attention should be paid to the following points:

a. The heating surface should be as large as possible, since the larger the heating surface the quicker evaporation takes place.

b. The space above the liquid should be as large as possible in order to avoid drops of juice being carried along by the steam from the boiling liquid.

c. The pipes connecting the steam room of one apparatus with the heating room of the next should be made as wide as possible in order to diminish the velocity of the flowing steam and thus afford such particles of juice as may be carried along an oportunity to settle.

d. A juice catcher should be inserted between the evaporator and the condenser for the purpose of catching the drops of juice that may be carried along by the steam and conducting them back into the evaporator. The more the velocity of the steam is diminished in this intermediate chamber the smaller will be the loss from evaporation of the liquor.

After many trials in calculating the capacity of the actual evaporation in each apparatus of a double, triple and quadruple effect, for evaporation of 90 per cent and 75 per cent of the original weight of liquid, it was shown that the total evaporation in each vessel is constant and of the same order (Hausbrand).

The quantity of water to be evaporated in the first vessel is on an average of the total evaporation:

Double effect	0.466
Triple effect	0.300
Quadruple effect	0.216

The extreme cases:

Double effect	0.434 — 0.484
Triple effect	0.2777 — 0.3152
Multiple effect	0.1926 — 0.2335

The evaporation effected by heating is in all cases the least in the first body, but the increase in the following vessels is not very great, at most 4 per cent. In the mean it may be assumed that this evaporation in the separate vessels is in the

Double effect	1 ÷ 1.045
Triple effect	1 ÷ 1.01 ÷ 1.04
Quadruple effect	1 ÷ 1.005 ÷ 1.012 ÷ 1.02

The total quantity evaporated in the last vessel is

Double effect	0.534
Triple effect	0.3703
Quadruple effect	0.284

of the total evaporation of the apparatus.

In the mean the evaporative capacity in each vessel is in

Double effect	1 ÷ 1.045
Triple effect	1 ÷ 1.0075 ÷ 1.138
Quadruple effect	1 ÷ 1.0055 ÷ 1.109 ÷ 1.196

According to the observations made by Classen in sugar factories and similar establishments the following ratios of transmission coefficients were calculated:

Double effect	1 ÷ 0.66
Triple effect	1 ÷ 0.70 ÷ 0.33
Quadruple effect	1 ÷ 0.91 ÷ 0.75 ÷ 0.55

Should these figures be reliable and the same temperature difference taken in all the vessels the heating surfaces will be in the ratio:

$$\text{In the double effect...} 1 : \frac{1.045}{0.66} = 1 : 1.58$$

$$\text{In the triple effect....} 1 : \frac{1.0075}{0.70} : \frac{1.138}{0.33} = 1 : 1.44 : 3.414$$

$$\text{In the quadruple effect.} 1 : \frac{1.0055}{0.91} : \frac{1.109}{0.75} : \frac{1.196}{0.55} = 1 : 1.105 : 1.48 : 2.175$$

(*To be continued.*)

Notes on the Chemistry and Metallurgy of Vanadium.—II.

By Warren F. Bleecker.

Methods of Analysis.

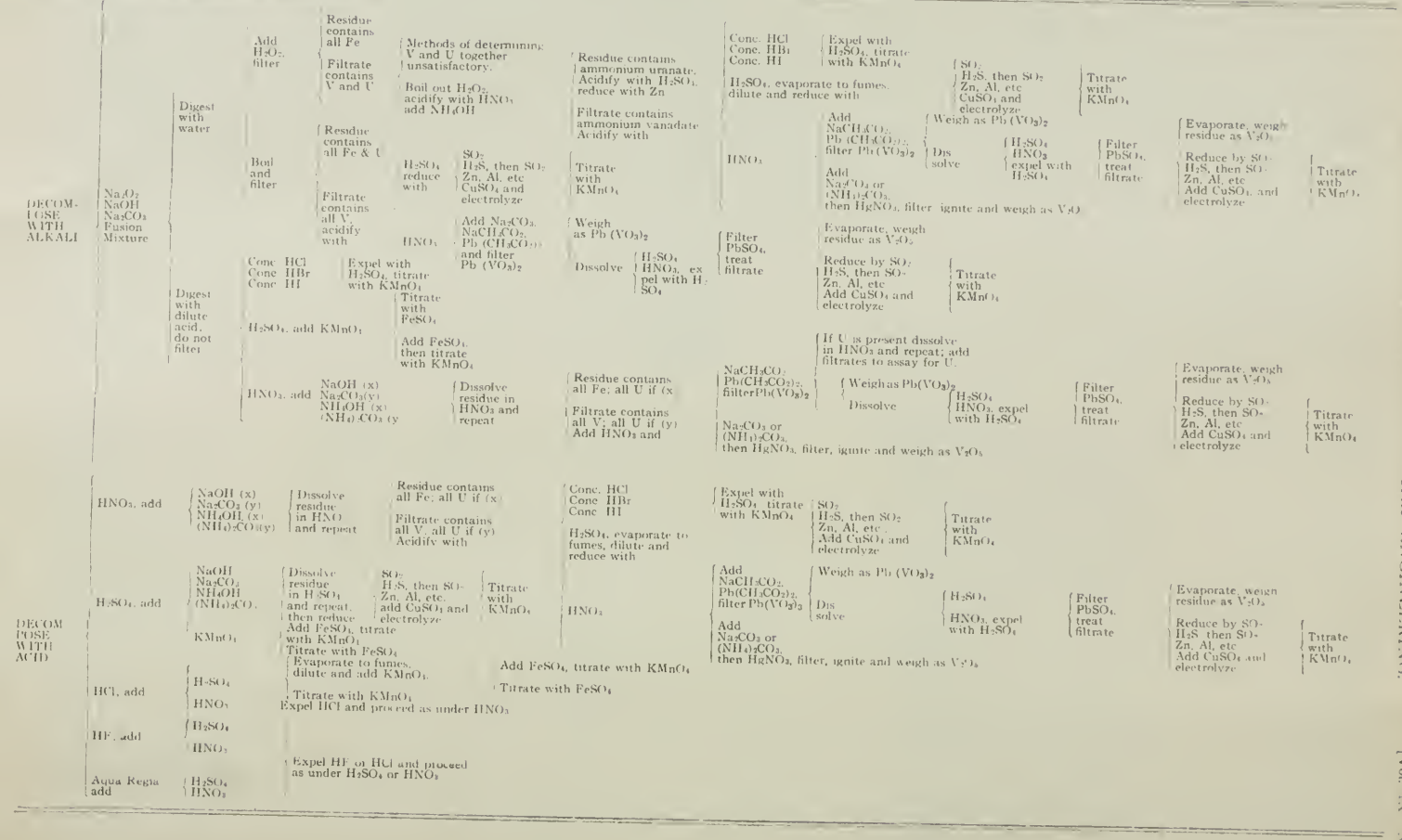
Regardless of the fact that the analysis of vanadium ore and products may be, and in most instances is, a simple procedure, chemists do not agree as to what methods give the best results. For this reason there have been proposed a comparatively large number of analytical schemes, each of which perhaps may give concordant results if used with the care and skill presupposed by the author, but which, when used by analysts who have devoted but little time to the study of vanadium compounds, are absolutely useless so far as correct results are concerned.

Vanadium may be determined either gravimetrically or volumetrically, either directly or indirectly, and it may be reported as the pentoxide or as the metal itself. At present there is a confusion due to chemists reporting vanadium in certain compounds, especially in ores, as the pentoxide, and in other cases, especially ferrovanadium, as the metal. Because ferrovanadium is at present the most important vanadium compound and because the vanadium is in this case metallic, I prefer to follow those who invariably report vanadium as metallic and not as the pentoxide.

In general, the analyst will meet with the following vanadium ores and products for analysis: Vanadiferous sandstone, roscoelite, vanadinite, patronite and carnotite; vanadate of iron, ferrovanadium, vanadic acid. In the mill there will also be the various solutions and intermediate products which are peculiar to the methods of treatment.

It is important to remember that the various ores and vanadium products met with may require diverse procedure in order to obtain satisfactory results in a reasonable time. For example, it is very necessary to know the exact grade of a carload of vanadium ore within close limits; a tenth of 1 per cent may mean 10 per cent of the entire metallic content, a variation which (in the assay) multiplied by five in the assay of zinc ore, for example, is allowable, but which in low-grade vanadium ores should in all cases be the maximum. As a mat-

TABULATED STATEMENT OF METHODS OF VANADIUM ANALYSIS.


DECOMPOSE WITH ALKALI
Na2O2 NaOH Na2CO3 Fusion Mixture
Digest with water
Add H2O2, filter
Residue contains all Fe
Methods of determining V and U together unsatisfactory.
Filtrate contains V and U
Boil out H2O2, acidify with HNO3, add NH4OH
Residue contains ammonium uranate. Acidify with H2SO4, reduce with Zn
Filtrate contains ammonium vanadate Acidify with
Residue contains all Fe & U
Boil and filter
Filtrate contains all V, acidify with
H2SO4 reduce with
SO2 H2S, then SO2 Zn, Al, etc CuSO4 and electrolyze
Titrate with KMnO4
HNO3
Add Na2CO3, NaCH3CO2, Pb (CH3CO2)2 and filter Pb (VO3)2
Weigh as Pb (VO3)2
Dissolve
H2SO4 HNO3, expel with H2SO4
Filter PbSO4, treat filtrate
Evaporate, weigh residue as V2O5
Reduce by SO2 H2S, then SO2 Zn, Al, etc Add CuSO4 and electrolyze
Titrate with KMnO4
Conc. HCl Conc. HBr Conc. HI
Expel with H2SO4, titrate with KMnO4
H2SO4, evaporate to fumes, dilute and reduce with
SO2 H2S, then SO2 Zn, Al, etc CuSO4 and electrolyze
Titrate with KMnO4
Add NaCH3CO2, Pb (CH3CO2)2, filter Pb (VO3)2
Add Na2CO3 or (NH4)2CO3, then HgNO3, filter ignite and weigh as V2O5
Digest with dilute acid, do not filter
Conc. HCl Conc. HBr Conc. HI
Expel with H2SO4, titrate with KMnO4
Titrate with FeSO4
H2SO4, add KMnO4
Add FeSO4, then titrate with KMnO4
HNO3, add
NaOH (x) Na2CO3 (y) NH4OH (x) (NH4)2CO3 (y)
Dissolve residue in HNO3 and repeat
Residue contains all Fe; all U if (x
Filtrate contains all V; all U if (y) Add HNO3 and
If U is present dissolve in HNO3 and repeat; add filtrates to assay for U.
NaCH3CO2 Pb(CH3CO2)2, filterPb(VO3)2
Weigh as Pb(VO3)2
Dissolve
Na2CO3 or (NH4)2CO3, then HgNO3, filter, ignite and weigh as V2O5
HNO3, add
NaOH Na2CO3 NH4OH (NH4)2CO3
Dissolve residue in HNO3 and repeat
Residue contains all Fe; all U if (x
Filtrate contains all V, all U if (y) Acidify with
Conc. HCl Conc. HBr Conc. HI
H2SO4, evaporate to fumes, dilute and reduce with
Expel with H2SO4, titrate with KMnO4
SO2 H2S, then SO2 Zn, Al, etc. Add CuSO4 and electrolyze
Titrate with KMnO4
DECOMPOSE WITH ACID
H2SO4, add
NaOH Na2CO3 NH4OH (NH4)2CO3
Dissolve residue in H2SO4 and repeat, then reduce
SO2 H2S, then SO2 Zn, Al, etc. add CuSO4 and electrolyze
Titrate with KMnO4
Add FeSO4, titrate with KMnO4
Titrate with FeSO4
KMnO4
HNO3
Add NaCH3CO2, Pb(CH3CO2)2, filter Pb(VO3)2
Add Na2CO3 or (NH4)2CO3, then HgNO3, filter, ignite and weigh as V2O5
Weigh as Pb (VO3)2
Dis solve
H2SO4 HNO3, expel with H2SO4
Filter PbSO4, treat filtrate
Evaporate, weigh residue as V2O5
Reduce by SO2 H2S, then SO2 Zn, Al, etc Add CuSO4 and electrolyze
Titrate with KMnO4
HCl, add
H2SO4
Evaporate to fumes, dilute and add KMnO4.
Add FeSO4, titrate with KMnO4
Titrate with KMnO4
Titrate with FeSO4
HNO3
Expel HCl and proceed as under HNO3
HF, add
H2SO4
HNO3
Aqua Regia add
H2SO4 HNO3
Expel HF or HCl and proceed as under H2SO4 or HNO3

ter of fact I have always repeated assays which do not check within four-one-hundredths. On the other hand, in the assay of mill solutions and products a variation of 1 per cent or more may be allowable, excepting, of course, the final assay of marketable products.

A method of analysis should, therefore, be chosen which will give sufficiently accurate results in the quickest possible time.

For convenience of illustration and reference I have designed the accompanying analytical scheme by means of which the various methods may be compared. While there is no pretension of completeness, the outline is sufficiently comprehensive to embrace most of the important analytical methods which have been proposed.

Decomposition and Solution.

The first operation in the analysis of vanadium compounds is obviously solution, but the difficulties attendant upon the quantitative solution of vanadium are not always understood or appreciated, possibly for the reason that a portion of the vanadium content goes into solution with extreme ease and rapidity.

As will be seen by referring to the table, the following reagents are used in dissolving vanadium, no one reagent being suitable for all of the vanadium compounds met with: Sodium peroxide, caustic alkali, sodium and potassium carbonates, mineral acids and water. In some cases heat is not necessary, but in all cases heat aids solution of vanadium salts, and when the fixed alkalis are used in the dry way, fusion must be made.

It is desirable at all times to have the substance to be analyzed in a fine state of subdivision, at least through an ordinary 100-mesh screen, and in the case of ores, particularly roscoelite, to pass all of the material through a 200-mesh screen. In this way the protective surface of insoluble compounds, while relatively much greater, is thin enough to allow the dissolving reagent to act upon practically all of the vanadium in the largest of the particles present in the mass. If fusion methods are used fine grinding is not so important, yet a much lower heat is required to effect decomposition if the material is in a finely divided condition than if merely ground to 20-mesh, as is directed by the authors of some methods. And, in any event, there is nothing to be lost in fine grinding, excepting, perhaps, a little time, which is amply regained in subsequent operations and which is again paid for by confidence in the completeness of the decomposition.

There is no ore or compound of vanadium that is not attacked by sodium peroxide in a state of fusion. Decomposition by means of this powerful reagent is, therefore, the most important, and if subsequent operations of the analysis will permit a general rule might be laid down that it should always be chosen when there is any doubt regarding the reagent to be used.

Fusion with sodium peroxide may be made in either iron or nickel, the iron being attacked more vigorously, but being at the same time much cheaper. Fused silica has been tried, but with little success. In the first place it is attacked to a considerable extent by the sodium peroxide, and, in the second place, the crucibles crack after a few melts. I tried four fused-silica crucibles and obtained the same results with each. Fused silica would be desirable for the reason that the large mass of iron or nickel hydroxide would be avoided, and filtering would be easier, and the analyst could be more certain that all of the vanadium was in the filtrate upon the removal of the residue insoluble in the sodium peroxide.

Having fused the vanadium compound with sodium peroxide (melting is not necessary, and should be avoided) it is a troublesome operation to filter the alkaline solutions after the requisite amount of water has been added to dissolve the melt and dilute the alkali so that it will not attack the filter. Analysts have sought to avoid this operation by devising methods which would permit of the determination of the vanadium in the presence of iron and other substances which may be present in the original compound.

Decomposition by Sodium Peroxide.

To effect decomposition by means of sodium peroxide I prefer to mix 200 mg to 2 gm of the finely divided substance with 1 gm to 5 gm of sodium peroxide in a small nickel crucible and heat gently until the mass darkens, as may be seen after heating for a few moments. As soon as the heating is finished the mass is removed by carefully dropping the hot crucible in a small beaker of cold water, the operation being conducted as follows: About 40 cc of cold water are poured into a beaker with a capacity of at least 200 cc. The hot crucible is grasped by the edge with a pair of curved crucible tongs and dipped into the water with a swirling motion in order to avoid breaking the beaker. None of the water should be allowed to splash into the crucible at this point because of the possible loss of vanadium by spattering. The crucible is finally set in the water which will have cooled it and distributed the energy somewhat. A cover glass is placed over the beaker, and by a gentle rotary motion small portions of the water are thrown into the crucible, causing a more or less violent action and tending to throw a portion of the melt out of the crucible and the beaker as well. If care is used nothing will be lost, and finally the beaker and contents may be set on the hot plate to complete the disintegration and solution. The crucible may then be removed with a glass stirring rod, thoroughly scrubbed with a policeman and rinsed with a stream from a wash bottle. The entire operation from the time weighing was begun ought not require more than 10 or 12 minutes. The residue will settle readily in the hot alkaline solution, which may be decanted through a filter. The first wash water also may be decanted. One or two washings on the filter will remove all of the vanadium from the residue and unless washing has been carried too far, there will be no iron in the filtrate.

In case it is desired to dissolve the iron with acid it is, of course, not necessary to filter at this point, and the acid may be added directly to the solution after the crucible has been removed. In case uranium is present a considerable quantity of hydrogen peroxide is added to the alkaline solution, when the uranium will go into solution along with the vanadium. If the hydrogen peroxide is expelled by boiling before the filtration is made the uranium will be found for the most part in the residue, because the uranate of sodium is only soluble in an alkaline solution containing an excess of hydrogen peroxide.

Assuming the absence of uranium and the removal of the iron at the first opportunity, the vanadium in the filtrate may be determined in a number of ways, as will be seen by referring to the table. The very shortest method from this point on, and one which will give perfectly reliable results, is one which I devised by the use of electrolysis.

Reduction by Electrolysis.

The filtrate, free from iron and containing no uranium, is made acid with sulphuric acid and about 10 cu. cm added in excess. It is then heated to boiling and about 50 mg of pure copper sulphate added. The hot solution is electrolyzed for 10 minutes, the same procedure and current conditions being observed as in the electrolytic determination of copper with a rotating anode. The copper will not be completely deposited in 10 minutes, but it will not interfere in the subsequent titration with potassium permanganate. The solution is immediately titrated without further heating. After titration the solution is again heated to boiling and again reduced as before. The copper deposited on the cathode may be dissolved almost instantly in the solution by momentarily reversing the current, thus avoiding the use of a greater quantity than originally added. The analysis may be thus repeated as many as five times, and the results should agree as closely as the analyst can read the burette. A complete analysis may be made by this method in less than an hour, titrating three times.

Other Methods of Reduction.

The reduction of vanadium assays by means of hydrogen sulphide involves the removal of the precipitated sulphur which always separates out of the acidulated solution, and it is only necessary when some interfering second group element is present. The result must be checked by a second reduction with sulphur dioxide and it is seldom that the results agree within the allowable limit of error.

The reduction of vanadium assays by means of sulphur dioxide is the one generally employed. The results are invariably high and due correction must be made by a blank test using pure water acidulated with sulphuric acid. Moreover, the presence of sodium and ammonium salts, as well as silicates, interferes seriously with accurate work when this reducing agent is used. It is customary, therefore, to remove these substances before reduction by the lead acetate method of Roscoe, the steps of which are shown in the table. The objections to this method are the length of time involved in making the separation (especially if much silica is present) and the constant opportunity for error due to the several different manipulations involved.

Numerous methods of reduction with metallic zinc, magnesium, aluminium and the like have been repeatedly tried and without success. It is stated that one metal will carry the reduction completely to one stage of oxidation and another metal will carry it completely to another, thus providing a method of assay not involving the removal of iron, but I have never been able to do these things, and have no hesitancy in saying that so far as useful results are concerned they are impossible under the stated conditions.

A method of analysis which does not involve the separation of iron has been proposed, and is being used, I understand, by some large producers and buyers of vanadium ores and products. I have never had the opportunity of making a personal investigation of the method, but it has been highly recommended from reliable sources.

Decomposition may be made by either acids or alkalis, preferably with sodium peroxide. The crucible is removed as described above, and dilute sulphuric acid is added in slight excess. The hydrogen peroxide is destroyed by the addition of potassium permanganate, and the vanadium is reduced by the addition of ferrous sulphate, the final titration being made with potassium permanganate. The ferrous sulphate solution need not be standard and the amount added is not measured. The theory of the operation is that, beginning with a solution containing both iron and vanadium in their highest state of oxidation, the addition of ferrous sulphate will reduce only the vanadium, and the amount necessary to complete this reduction will, of course, be oxidized, and any excess of this quantity will be observed by a spot plate test, using potassium ferricyanide as an indicator.

The method in detail as follows was outlined to me by Mr Arnold J. Guerber, who, however, does not claim the invention.

Fuse 0.2 gm to 5 gm of the product or ore, depending upon the grade, with 2 gm to 10 gm of sodium peroxide in an iron crucible. Leach melt with water, heat, and add 20 cc sulphuric acid (1-1). Boil the solution until the hydrogen peroxide is decomposed, as shown by the disappearance of the red or pink color. Add sufficient permanganate to give a permanent pink. Now add ferrous sulphate until a spot test with potassium ferricyanide gives a distinct blue. Run back carefully with permanganate until this color just disappears in spot after 15 seconds. Then read $KMnO_4$ burette and titrate to a distinct pink color, which should endure at least 30 seconds. Solutions:

Potassium ferricyanide—dissolve 50 mgm in 10 cc water.

Ferrous sulphate—3 gm $FeSO_4$ and 10 cc H_2SO_4 in 150 cc water.

Potassium permanganate—1 cc = about 0.005 gm Fe.

Fe factor $\times$ 1.6316 = V_2O_5; or Fe factor $\times$ 0.916 = V.

Theoretically the method is open to objection because the action of iron and vanadium in the presence of a reducing agent is reversible, as may be easily proved experimentally.[3] If a solution of hypovanadic acid or its salts be added to a solution containing ferric iron, the presence of ferrous iron is immediately observable. According to this we should expect low results, owing to the mass action of the reduced vanadium upon the ferric iron, giving a test for ferrous iron before the vanadium is all reduced. That correct results are obtained may be due to the fact that a stated length of time is given for the spot-plate reaction to take place, or to some other modifying condition which compensates for the reduction of the iron which must take place.

A method has been proposed by means of which the titration of vanadium in the presence of iron is made directly with a standard solution of ferrous sulphate. Since the method seemed attractive a great many experiments were performed to try it out. The results were very unsatisfactory, because in almost every instance a spot-plate test for ferrous sulphate would show its presence long before reduction was complete, and there appeared to be no constant error that could be taken into account.

Separation of Vanadium and Iron.

It is usually a very difficult matter to make a complete separation of iron and vanadium. This is true on account of other elements being present, for the separation of iron and vanadium in a high-grade vanadate of iron is very easy. It is in the low iron products that the separation becomes a serious matter. The elements which interfere with the separation and which are almost invariably present are alumina and lime. In the high-grade vanadium compounds these elements are not usually present in such quantities as to affect the separation of the iron unless the product is a vanadate of aluminium or a vanadate of lime. But in the case of ores, such as vanadiferous sandstone or roscoelite, the alumina and lime are high, while the iron is low, and it is here that the difficulty arises.

If to a concentrated solution of aluminium vanadate in a mineral acid either ammonia or ammonium carbonate be added there will result a flocculent yellowish precipitate which consists of aluminium hydroxide contaminated with aluminium vanadate. If the solution be largely diluted before the ammonium salt is added the precipitate will usually be white and consist only of the aluminium hydroxide. If the above experiment be repeated, using the vanadate of calcium instead of the vanadate of aluminium, the results will be very similar. It must be remembered, therefore, that when either lime or alumina are present in the sample taken for assay the solutions must be dilute in order to effect the separation of the vanadium from these elements; indeed, all of the vanadium may be carried down under certain conditions of concentration. It is equally true that iron may also carry vanadium down as vanadate of iron, but if the solution is sufficiently dilute and there is an excess of the vanadium solvent present it is not likely that iron alone will cause the trouble that either of the above-mentioned elements will.

Care must be taken, however, to oxidize both the iron and the vanadium before attempting a precipitation. The iron must be oxidized because ferrous hydroxide, as is well known, is colloidal. The vanadium must be oxidized because salts of hypovanadic acid are partially insoluble in dilute alkalies, especially ammonia.

It will be clear, then, why sodium peroxide is the ideal reagent for separating vanadium from iron, the solutions being made dilute before attempting to filter. All of the elements are oxidized, and the aluminium goes into solution with the vanadium, as well as most of the lime. If desired, the aluminium, lime and silica may be separated later by any one of a number of methods.

If an acid decomposition has been made and it is desired to make the iron separation, we are usually advised to add am-

monia and ammonium carbonate. We are also further warned, and very properly, that the separation is not complete with one precipitation, but that several subsequent precipitations may be necessary in order to remove all of the iron from the vanadium and all of the vanadium from the iron. I have found that the separation appeared in many assays to be impossible, and after a half dozen times of redissolving and reprecipitating there was apparently as much vanadium in the iron residue as after the first precipitation. In such cases if the third precipitation be made with sodium carbonate, the separation will be complete. I have assumed that this phenomenon is due to the fact that the aluminium which was retaining the vanadium each time is dissolved, or, rather, not precipitated, and cannot further interfere with the complete removal of the vanadium from the iron.

The vanadium in mill solutions which do not contain iron may be readily determined as follows: 25 cc of the solution is diluted with about 150 cc water and an excess of sulphuric acid is added; 50 mg of copper sulphate is added to the solution and the assay heated to boiling and electrolyzed as described above. The relatively large quantities of sodium, lime and other salts present do not interfere with the determination.

Vanadic acid may be decomposed readily by digesting the assay in a hot solution of either sodium or potassium hydroxide to which a little hydrogen peroxide has been added, and the subsequent determination made by any method desired. If one attempts to decompose vanadic acid which has been fused, with an acid, the operation will be found to be tedious and unsatisfactory, whereas the alkali will effect a complete solution almost immediately if there is an oxidizing agent present.

Inaccuracies of Some Methods.

There are a number of methods which are not included in the table, because of their inaccuracy or general unfitness for commercial work. Of these I shall mention but one or two. One of the first methods for the determination of vanadium consisted in getting the vanadium into alkaline solution and precipitating it as ammonium metavanadate by means of ammonium chloride. This method has caused much discussion, and Gooch and Gilbert[2] did some very painstaking work to determine its accuracy. They determined that the precipitation is complete if the solution is so saturated that the ammonium chloride separates out, and the solution so saturated is allowed to stand for 24 hours. I should like to point out that the vanadium salts used in these experiments were pure, and that sodium salts, silica and the like were entirely absent, conditions which seem to make the experiment of academic interest merely and of small value to the analytical chemist.

Several investigators have tried to determine vanadium as the sulphide, either by precipitation from alkaline solution with hydrogen sulphide or by the addition of sodium sulphide and acidifying the solution with a mineral acid. The precipitate formed in these instances is an impossible colloid and represents only a fraction of the vanadium in the original solution.

Brearly and Ibbotson[3] suggest, beside the ammonium metavanadate method, the determination of vanadium by precipitating a dibarium vanadate from an alkaline solution, which vanadate may be ignited and weighed. The insurmountable obstacle here seems to be as in the case of lead vanadate, Prescott and Johnson[4], that the compound is not of constant composition.

The same authors also state that it has been proposed "to weigh the carbon dioxide evolved on reducing vanadium pentoxide with oxalic acid."

Koppel and Goldmann[5] have shown that the action of oxalic acid upon vanadic salts gives hypovanadic salts and vanadyl oxalate. It does not seem clear that these products are formed in any definite ratio, and it is a fact that the reduction of numerous samples of a standard vanadyl sulphate solution with oxalic acid gave results which were so widely different that they were of no value, and no result was obtained which agreed with the true value as determined by standard methods.

The mercurous nitrate method of Rose should be mentioned as a standard method involving serious difficulties. As given by Treadwell[6] the silica would be weighed as vanadium pentoxide. The removal of the silica is a tedious procedure, but unless it is removed there is a very strong probability that a portion of it at least will be precipitated with the vanadium. Moreover, the vaporization of mercury is not an easy matter, as is well known, and unless the quantity of vanadium pentoxide in the assay is very small, the result is apt to be high on account of the traces of mercury remaining in spite of the heat applied.

The modification of Roscoe's lead-vanadate method,[7] in which the final acid solution containing the vanadium is evaporated and the residue weighed as vanadium pentoxide, is not altogether satisfactory. The lead cannot be entirely removed with hydrochloric acid, and the evaporation of sulphuric acid is unpleasant as well as difficult. The safest procedure is to precipitate the lead with hydrogen sulphide from a hydrochloric acid solution, and the opportunities for error here are too well known to require discussion.

Colorimetric methods have been advocated, but do not seem to have met with an enthusiastic reception.

The reduction of vanadium by means of the halogens has been very carefully worked out, and the investigators[8] seem to be unanimously of the opinion that the results are accurate. The care with which the work must be performed and the length of time required seem to be obstacles which will forever prevent these methods coming into general use.

(To be continued.)

The New Ozone Purification Plant of the Water Works of St. Petersburg.

By Dr. G. Erlwein.

The City of St. Petersburg, in Russia, has built a large ozone purification plant for their water-works to sterilize the drinking water. The special object is to combat the epidemics of cholera morbus which have occurred every year now for some time, and the typhoid epidemics which have already become chronic. This ozone purification plant has started operation at the beginning of this year.

For the water supply of the City of St. Petersburg three large water-works were formerly used, of which only the largest one, however, used sand filtration, while the other two pumped the river water from the Neva River without purification into the water supply pipes of the city.

The new ozone purification plant is used in connection with one of the two water-works formerly operating without filtration and supplies water to a district consuming daily 50,000 cu. m (13,200,000 gal.) of water. The purification of the water consists in rapid filtration with subsequent ozonization. In view of the large amount of suspended material in the water, especially when ice is in the river, the rapid filtration is combined with clarifying by means of aluminium sulphate. The Penkowaja ozone purification plant consists, therefore, of two main parts, first the filtration plant and second the ozone plant proper.

As will be seen from the diagram, Fig. 1, the river water is pumped directly from the Neva River and the suspended matter is first removed. For this purpose, before the water enters the sedimentation basins, aluminium sulphate solution is added to

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[1] *Western Chemist and Metallurgist*, Vol. 5, No. 2, p. 54.
[2] *Zeit. f. Anorg. Chem.*, 1902, p. 174.
[3] "Analysis of Steel Works Materials," Ed. 1902, p. 87.
[4] Prescott & Johnson, "Qualitative Chemical Analysis," Ed. 1908, p. 136.
[5] *Zeit. f. Anorg. Chem.*, 1903, p. 281.
[6] Treadwell & Hall, "Quantitative Analysis," Vol. 2, p. 231.
[7] Treadwell & Hall, "Quantitative Analysis," Vol. 2, p. 233.
[8] *Zeit. f. Anorg. Chem.*, 1902, pp. 66, 71, 456; 1904, p. 246.

the water in proper dilution according to the well-known American practice.

The sedimentation basins are shown on the left of Fig. 2. Their number is eight, and by means of pipe lines they are connected with the filter tanks shown on the right hand of Fig. 2.

The rapid filter tanks, of which there are 38, are of the Howatson systems, which is very similar to the well-known American Jewell filter. Their construction is well known, but it may be mentioned that Howatson fills his filters not with the

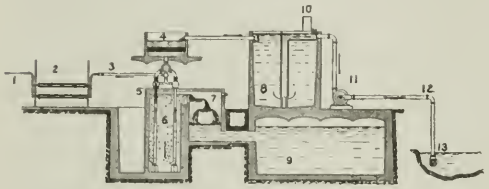


FIG. 1. DIAGRAM OF OZONE WATER STERILIZATION PLANT AT ST. PETERSBURG.

1, air pump; 2, ozonizer; 3, ozone pipe; 4, rapid filter; 5, injectors; 6, sterilization tower; 7, cataracts; 8, decantation vessel; 9, storage of purified water; 10, aluminum sulphate addition; 11, pump; 12, raw river water; 13, Neva River.

usual crushed filter sand, but with "so-called prepared silex (flint)."

The filtration speed is 4.5 m (14.8 ft.) per hour.

The purification of the filters is carried out in the well-known manner by passing filtered water back through the filter and simultaneously stirring the silex filling by means of agitating devices, which are clearly shown in Fig. 3.

The filtered water is passed through a pipe line directly to the ozone-sterilization towers. A small part of the filtered water, however, is conducted into a storage tank, which contains the

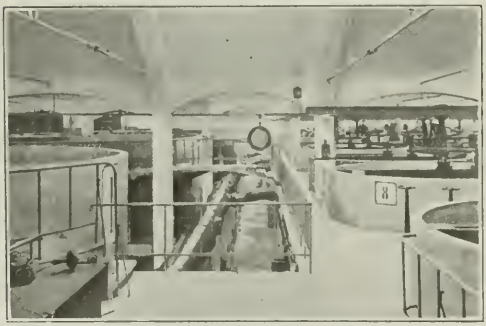


FIG. 2—SEDIMENTATION AND FILTRATION PLANT.

water used in washing the filters. While in slow filtration sand filters are expected to remove the bacteria, the rapid filters in this case are not intended to do this. Their principal object is to remove the suspended and coloring matter from the water.

As the next step after the filtration just described comes the ozone treatment proper. In the ozone plant two parts must again be distinguished, namely, the ozonizers and the sterilization towers. Fig. 4 shows the ozone plant, consisting of 128 ozonizers, and Fig. 5 shows the five sterilization towers, one of which is used as reserve. The ozonizers are of the well-known Siemens & Halske tube type (see, for instance, our Vol. I, page 179). These ozonizers change the oxygen of the air into ozone by means of high-tension discharges.

On account of the method of operation of the injector sterilization towers a relatively high value has been chosen for the concentration of ozone, namely, 2.5 grams of ozone per cubic meter of ozonized air. Before entering the ozonizers the air is properly dried by means of a refrigerating machine; this drying increases the yield of the ozone in the ozonizers.

The movement of the air through the ozonizers and pipes is accomplished by means of Otto injectors, which operate with a water pressure of 4 m (14 ft.) and suck the ozonized air from the ozonizers and drive it mixed with the water into the sterilization towers. The absorption of the ozone and the sterilization of the water occurs partly in the injectors at the top of the sterilization towers and partly in the towers in which the ozonized air rises from the bottom in a very finely subdivided state and, therefore, in very intimate contact with the water.

From the injector-sterilization towers the water is passed in cascades (to remove the air) into a pipe line which brings the sterilized water to the storage tank for pure water. From there the water is pumped by high-pressure pumps into the supply pipe lines.

For the operation of the whole plant a special electric station has been erected, containing boilers, steam engines and three-phase generators. By means of three high-frequency converters the three-phase currents produced in the generators are brought to a frequency of 500 periods. This is necessary because the ozonizers require a high-frequency current. By

FIG. 3.—RAPID FILTRATION PLANT, SHOWING STIRRING DEVICES.

means of transformers the voltage of the currents is raised to 7000 volts, which is the operating voltage of the ozonizers.

The power plant contains two steam engines, each of 150 hp, direct-connected with three-phase generators. A third machine is used as reserve. In the other parts of the plant a reserve of about 30 per cent is also provided.

With reference to the machine capacity of 2×150 hp, it is to be taken into consideration that the electric current is used not only for the ozonizing plant, but for driving the numerous motors of the filtration plant which are employed for driving the stirring devices, the filter washing pumps, etc.

The large basins, sedimentation tanks and filtration tanks are built of reinforced concrete.

As to the cost of operation experience has shown so far that if the cost of aluminium sulphate and rapid filtration is included, the cost is about 0.9 kopeken to 1 kopeken per 100 wedro (about 0.4 cent to 0.45 cent per cubic meter of water). For ozonization alone the cost is about one-half this amount. Under conditions as they exist in Germany it may be estimated that the cost of the ozonizing plant, not including aluminium sulphate treatment and rapid filtration, would be about 0.6 pfennig (0.15 cent) per cubic meter.

With respect to the physical and bacteriological purification effects, this combined rapid filtration and ozonizing system fulfills all expectations. The dirty river water of the Neva is changed into a clear and hygienically perfect drinking water. The number of bacilli spores in the ozonized water is very low, varying between 0 and 3 per cubic centimeter. Coli bacteria or pathogenous spores of cholera morbus and typhoid are not found in the sterilized water.

It may be emphasized that the city officials of St. Petersburg

are especially interested in the bacteriological effect of the ozonizing process and have appointed a committee consisting of well-known Russian bacteriologists and sanitary water engineers which supervise the operation of the works and continually control the results obtained.

Before the new commercial ozonizing plant was opened this committee of experts had made very careful experiments on the ozone treatment of water from the Neva River on a somewhat large scale in an experimental plant treating 100 cu. m

FIG. 4.—OZONIZER PLANT.

of water per hour and had thus thoroughly investigated the problem of the general use of ozone for the sterilization of drinking water, especially under such unfavorable conditions as exist in St. Petersburg.

The plant has been built for the city by the Russian Siemens & Halske Company, which has also undertaken the operation of the plant for the first years. The design and the construction of the works was made by the Siemens & Halke Company and

FIG. 5.—STERILIZATION TOWERS.

by the Compagnie de l'Ozone, which furnished the injector-sterilization towers of the Otto type, and by the Felten-Guilleaume-Lahmeyer Company, which furnished a part of the electrical equipment.

Two very large ozone purification plants for the treatment of water, each of a daily capacity of 45,000 cu. m (12,000,000 gal.) will be erected for the City of Paris, in France, within this year. The process of treating water by ozone is thus used now on a large scale to supply drinking water from a central plant, the different steps of this evolution being indicated by the plants erected in the cities of Paderborn, Wiesbaden, Hermannstadt, Florence, Nizza, Chatres, Villefranche, Rovigo, Chemnitz, and St. Petersburg. In view of this progress the doubts which originally existed, especially even among German engineers, concerning the technical and commercial suitability of the process may be considered to have been proven unfounded. It may be expected that in the future other large water-works with particularly unsanitary conditions will make use of this modern and hygienically important method of water purification.

Berlin, Germany.

The Annealing Process as Applied to the Manufacture of Malleable Iron.*

By W. P. Putnam.

To emphasize tht importance of exercising greater care in manufacturing malleable castings and to point out some of the sources of error that are so frequently found in foundries that leave the annealing process to the judgment of some individual with a "trained eye," the work of this paper was undertaken. There is no branch of the foundry business that needs a closer supervision than the process of annealing iron; no greater source of economy; no one department where greater skill is required, and yet it is safe to say that very few foundries

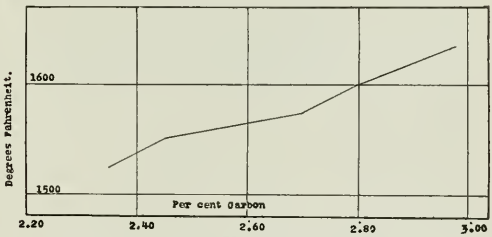


FIG. 1.—PROPER FINISHING TEMPERATURE FOR THE ANNEALING OF WHITE IRON IN RELATION TO CARBON CONTENTS.

recognize the importance of equipping their annealing ovens with proper appliances to insure a uniform product.

To call attention to the variations that can be produced in the annealing process a set of test bars was made and sent out to a number of malleable iron foundries to be annealed in accordance with their usual practice. The bars were all made from the same metal and an attempt was made to obtain a bar free from shrink or draw in the center by casting the bars with large ends. In all of the bars that were broken but very little draw was found.

The analysis of the metal poured into the bars was as follows:

	Per cent.
Silicon	0.74
Sulphur	0.041
Phosphorus	0.148
Manganese	0.24
Combined carbon	2.70

The results obtained on the annealed bars are given in Table I.

It will be noticed that in each case where too high temperatures or a long period of annealing were practiced the carbon was nearly all removed. Either practice is wasteful and produces poor malleables for some purposes.

The annealing process should be conducted to produce definite results. Metal intended for parts where machining is

*A paper presented before the Detroit Foundrymen's Association.

TABLE I.

*No.	Total Carbon.	G. C.	C. C.	E. I.†	T. S.†	Elong. in 2".	Red. in Area.	Kind of Packing.	Time of Anneal.‡	Anneal Temp.§
1.	1.49	1.49	Trace	29823	40996	8.59%	12.13%	Slag	112 hrs.	1450
2.	1.94	1.94	"	34134	44034	10.15%	13.87%	Mill scale	144 hrs.	1700
3.	1.47	1.47	"	29276	39633	6.59%	17.63%	" "	101 hrs.	1620
4.	.86	.77	.09	31428	48489	8.59%	19.18%	Not given	Not given	Not given
5.	1.74	1.74	Trace	30060	45760	8.59%	14.88%	" "	" "	" "
6.	1.71	1.71	"	28326	40408	8.59%	15.00%	None	72 hrs.	1650
7.	1.32	1.18	.14	32224	49107	8.59%	13.23%	Not given	Not given	Not given
8.	.80	.80	Trace	33333	47336	7.60%	17.27%	Slag	208 hrs.	1620
9.	1.58	1.58	"	30833	39862	7.03%	19.51%	Mill scale	168 hrs.	Not given
10.	.40	.40	"	41792	50849	7.45%	20.92%	" "	96 hrs.	2200
11.	1.32	1.32	"	35042	45213	7.80%	12.56%	Slag	144 hrs.	1700
Hard Iron			2.70		50846					

*Foundries represented. †Pounds per sq. in. ‡Total hours. §Degrees F.

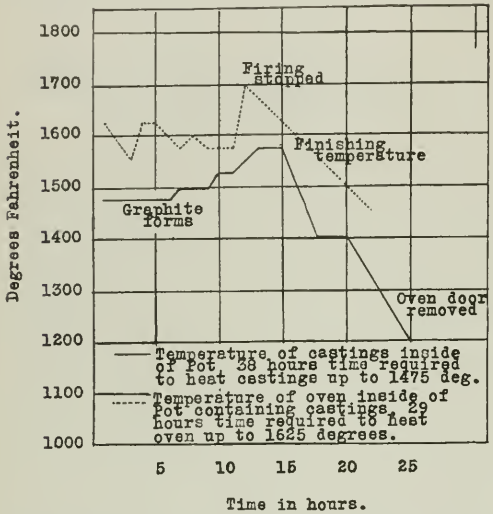


FIG. 2.—TEMPERATURE CURVE OF TYPICAL ANNEALING CYCLE.

necessary should be given a different treatment in the annealing process than metal that does not need to be machined.

To arrive at the proper annealing temperature the critical and proper annealing temperature was ascertained on a number of hard iron specimens ranging from 2.35 per cent to 3 per cent total carbon. These temperatures are diagramatically shown in Fig. 1. These temperatures are accurate within 10 deg. Fahr.

In Fig. 2 is shown the curve of a typical anealing cycle in an oven containing 14 tons of castings. The carbon in this lot of iron was 2.65 per cent. After the metal reaches the annealing temperature there is no further rise in the temperature of the metal until all of the carbon has changed into the graphitic condition. As soon as the change has been completed the temperature will rise in accordance with the temperature of the oven.

It has been found that to heat the castings much above the proper annealing temperature for a long period is not only unnecessary but a waste of fuel and produces poor castings for some purposes. After the carbon has all been changed the oven need not be fired for more than six hours. When the metal has gradually cooled to 1200 deg. Fahr. the cooling can be hastened as rapidly as desired without harming the quality of the castings.

Some of the reasons why hard and brittle castings are occasionally found may be summed up as follows:

1. Under annealing (which may be caused by too low temperature or by holding the castings at the proper temperature for too short a period).

2. Too rapid cooling of the castings after being annealed.

3. Heating anealed castings above the critical temperature to straighten without subsequent annealing.

That there is great need of the systematic handling of the annealing process is clearly evident from the results tabulated in Table No. 1. The variations shown there are typical of castings in general.

Detroit Testing Laboratory,
Detroit, Mich.

Notes on Chemistry and Metallurgy In Great Britain.

(From Our London Correspondent)

The Corrosion of Brass and Copper.

At a recent meeting of the Liverpool Engineering Society Mr. G. D. Bengough read a paper on this subject, in the course of which he stated that modern investigations indicated that for the continuous solution or corrosion of metals and alloys it was necessary that there should be in and around them substances which could form voltaic couples. There must be areas capable of acting respectively as anodes and as cathodes and also a fluid capable of acting as an electrolyte. When sea water was passing through a metal tube probably the bulk of the metal might act as an anode, the salt water was the electrolyte, but the question of greatest interest was what constituted the cathode. Lead, copper arsenide, basic carbonate of copper, ferric oxide, nickel, carbon, cuprous oxide, and even shrimps and seaweed, had at various times been stated to promote corrosion of brass and copper by forming cathodes. The presence of air was, by many chemists, considered essential to the corrosion of brass and copper by acid or saline solutions; and it was at least certain that access of air greatly increased the speed of corrosion.

Some considered that the strongly adherent green deposit frequently found on copper increased the speed of corrosion by absorption of oxygen; some thought it acted as a cathode; while others credited it with a protective action. As to the action of stray currents, it was a fact that corrosion of condenser tubes became considerably more troublesome soon after the introduction of electric lighting on board ship; but, in spite of this and of other more direct evidence, at least one well-known authority maintained that the progress of corrosion on board ship was not affected by external currents because of the very small magnitude of the current which could be conveyed by the water in condenser tubes. Now, a comparison of the magnitude of such a current with the magnitude of the currents set up by the voltaic cells previously referred to, and which did undoubtedly cause corrosion, showed that the two were approximately of the same order, and with length of time both might effect serious corrosion. It was also possible that very small induced currents might act similarly. The presence of depolarizers in the metal or liquid might exercise an important effect on the speed of corrosion, but their influence had hitherto not received much attention. Thermo-chemical action was another possible cause of corrosion, but no experimental establishment of its action had been published. The effect of increased speed of circulation of water had not yet been satisfactorily ascertained, and but little information as to the effect of sewage and other contamination of the circulating water was available.

The methods at present employed for the prevention of corrosion in brass and copper tubes were either mechanical or

electrochemical. In the mechanical class varnishes, asphalt and special cements had been tried without much success. Metallic coatings, especially tin, lead and tin-lead alloys, were better, and a particular form of ferric oxide had stood exhaustive tests on a small scale. For brass tubes the only coating now used extensively was tin, but the layer of tin was necessarily very thin—usually not more than 1/50 mm—and, therefore, very liable to pinholes and slight defects which would result either in the tin being rapidly dissolved away or in the tube being corroded more quickly, according as the tin or the tube were the more electro-positive. The efficiency of a lead coating depended on its purity, and that of a lead-tin alloy on the percentage of tin and of the impurities, while the thickness, continuity and mode of application of the layer were also factors.

Electrochemical methods consisted in the sacrifice of a cheap electro-positive metal for the preservation of the expensive one which was negative to it, for which purpose iron and zinc were generally used. It was often stated that the iron condenser casing helped to prevent corrosion of the tubes, and in some cases it certainly appeared to do so; in others it was not effective and bars of iron or zinc were placed in the water spaces at the ends of the condenser and screwed to the tube-plate; but it appeared that the protection so afforded extended only to the tube-plate and for a short distance along the length of those tubes satisfactorily connected to it.

When bars of iron or zinc were so used corrosion of the ends of the tubes should be less than it was further along, and if this were not found to be so, the meaning would be in the case of any particular tube that it was not in good metallic contact with the tube-plate, or in general that the iron or zinc was not in good contact with the tube-plate, or that the composition of the tube-plate varied too much from that of the tubes.

The protective influence of zinc had only a limited area, and in dealing with the question of the extension of that area it was necessary to consider in detail the protective action of the zinc. If no free zinc were present the brass would tend to dissolve. Brass, however, did not dissolve as such, but its zinc passed into solution to a much greater extent than its copper. When free zinc was present and a current was passing from it to the brass there was a tendency to deposition of zinc on the brass instead of solution of zinc out of the brass, yet this tendency was only slight and no appreciable quantity of zinc was actually deposited. Nevertheless, the tendency existed, and, as it constituted the protection of the brass from corrosion, if it were enhanced the protected area might be extended. The obvious means of effecting this was to pass a small and carefully regulated current from zinc to brass, keeping the e.m.f. low—2 volts to 4 volts—to avoid waste of zinc.

Progress of Electric Smelting in Norway.

The Stavanger Electro-Staalverk—a new company having a capital of £25,000—has been promoted for the electrical production of high-grade steel from the iron and steel scrap from broken-up ships. Besides the furnaces, the works at Jorpeland will include a steel foundry, rolling mills and hammers. The available energy is 2500 hp, and the annual output is estimated at 1400 tons of first-grade steel billets, 600 tons of cast steel and 300 tons of hammered steel. The Arendals Fosse Kompagni, a similar company, which has increased its capital to £233,300, is making arrangements for taking 12,000 hp from a waterfall near Arendal, and has elected to use the Grönwall process. At the Ilen Smelting Works, at Trondhjem, experimental runs, chiefly on copper, have been made in a 1000-hp electric furnace, and it is reported that the company has definitely decided to adopt electric smelting and will shortly start an iron smelting furnace to give an annual yield of 9000 tons.

British Engineering Imports and Exports.

The Board of Trade returns for January, 1911, put the value of imports of iron and steel, including manufactures, at £825,305, an increase of £110,689 as compared with January, 1910; exports amounted to £3,830,687, showing an increase of £351,815. Imports of other metals, including manufactures, were £2,247,501, an increase of £16,402, and exports, £864,988, an increase of £39,575. Imports of electrical goods were £125,805 and exports £202,996, showing increases of £16,334 and £47,922 respectively. Imports of machinery reached £433,719, an increase of £99,903; with exports £2,486,083, an increase of £385,330. Imports of new ships, £9,791, an increase of £9,688; exports of ships, £230,208, a decrease of £1,910,078.

Analytical Notes.

Separation of Iron, Aluminium and Chromium.

Tcharviani and Wunder proceed by fusing the mixed oxides with sodium carbonate in a platinum crucible for seven hours; the cooled contents of the crucible are leached out with water and the iron oxide is filtered off. In the filtrate the alumina is precipitated by excess of ammonium nitrate; the filtrate from this is acidified with nitric acid, and, after the reduction of the chromate by alcohol, chromium is precipitated as hydroxide by excess of ammonia, the precipitate dissolved in hydrochloric acid and reprecipitated with ammonia. By this method the alumina is obtained in a much purer state than either by precipitation from any acid solution with ammonia or from an alkaline solution after decomposition of the aluminate by ammonium chloride.

Estimation of Small Amounts of Lime in the Presence of Much Magnesia.

In the method employed by C. Liesse the solution is diluted to the extent of 1 in 1500 when magnesium oxalate is soluble. One gramme of the substance is dissolved in 25 parts of hydrochloric acid, and after dilution with 100 parts of water the solution is made slightly alkaline with ammonia, heated, filtered, cooled and diluted to 1500 cc. Four grammes of ammonium oxalate are added, the solution is acidified with acetic acid, well stirred and allowed to stand two hours before filtering. Good results are obtained with 99.5 per cent of magnesia and 0.5 per cent of lime.

Estimation of Chromium in Chrome-Tungsten Steel.

Henrik Wdowiszewski recommends a modification of Von Knorre's method. To one to two grammes of borings in a half-liter Erlenmeyer flask 10 to 15 cc. of a 15 per cent solution of disodic hydric phosphate and then 8 to 16 cc. of sulphuric acid, sp. gr. 1.65, and 5 cc. of water are added. On gently heating the metal dissolves in about 15 minutes; the heat is then increased and 2 cc. of nitric acid, sp. gr. 1.4, are added to convert tungsten into soluble phospho-tungstic acid. After dilution of the clear liquid with 300 to 500 cc. of hot water the addition of 3 to 5 grammes of ammonium persulphate oxidizes chromium to chromic acid. Excess of persulphate is decomposed by boiling, a little more sulphuric acid is added and the chromic acid is titrated with ferrous ammonium sulphate, the excess being determined with permanganate.

A Sensitive and Stable Litmus Solution.

A. Püschel states that the sensitive violet-colored constituent of litmus can be separated from the other coloring matter, gum and inert substances contained in commercial litmus by heating with dilute sulphuric acid—4 per cent—which precipitates it, probably as a sulphonic acid. The precipitate is collected, washed, redissolved in hot water and neutralized with sodic hydroxide. The indicator thus obtained has a violet color and is extremely sensitive.

Estimation of Sulphur in Brass and Bronze.

Gustav Thurnauer obtains excellent results by the following method: Thoroughly mix 1 gramme of filings with 6 grammes of a mixture of two parts of potassic chlorate with one part sodic carbonate; transfer to a wrought-iron crucible lined with sodic carbonate, cover with more of the fusion mixture and heat gradually until fusion is complete. When

cooled, digest the mass with hot water, filter off the oxides, acidifying the filtrate with hydrochloric acid, and determine the sulphur as barium sulphate. The author admits that the error may be as much as 0.01 per cent, and adds that if great accuracy be required a silver crucible should be used. If "an error of 0.01 per cent" mean a difference of 0.01 per cent of sulphur in the metal, the inaccuracy due to the use of an iron crucible would amount to a tolerably large proportion of the sulphur present in brass or bronze of average quality.

Market Prices.—February, 1911.

Copper, with the other metals, was depressed toward mid-month. Opening at £55.5.0 it had fallen by the 8th to £54.12.6, thence recovered gradually to £54.16.3.

Tin has again been erratic, opening lower at £195 and dropping with extraordinary rapidity to £182 by the 8th, and £171 on the 9th, recovering sharply to £197 by the 15th, and has now settled again to £193.

Lead has been stronger, being slightly lower at £13.5.0 on the 8th, recovering to £13.10.0, and closing at £13.5.0.

Iron. Hematite has been down from 67/2½, dropping to 66/1½ by the 15th. It has recovered, however, to 67/1 and present transactions rule rather lower.

Scotch Pig has also been down, touching 55/- on the 15th and 54/- on the 18th, and only recovering to 55/1½.

Cleveland Warrants are in sympathy, dropping to 48/11½ by the 15th and only showing 48/11 on the 25th.

Platinum, per oz	£8	6	6
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphide powder, per ton	22	0	0
Borax, British refined crystal, per ton	16	0	0
Sulphate of ammonia, f.o.b. Liverpool, per ton	14	2	6
Sal ammoniac, lump, firsts, delivered U. K., per ton	42	0	0
Copper sulphate, per ton	19	17	6
Caustic soda, ash 48 per cent, ordinary, per ton	5	10	0
Litharge, flake, per ton	15	10	0
Sulphur, recovered, per ton	5	0	0
Camphor, 1-oz. tablets		1	10
Shellac, standard T. N. orange spots, per cwt	3	12	0
Copper ore, 10 to 25 per cent. unit	9 3 to	9	0
Tin ore, 70 per cent, per ton	£123 to	£125	
Carbolic acid, liquid 97/99 per cent, per gal		1	4
Creosote, ordinary good liquid, per gal			2
Naphtha, solvent, 90 per cent at 160° C., per gal			9
Petroleum, Russian, spot, per gal	4⅞d. to		5
Mica, small slab, original cases, per lb	6d. to	1	0

India rubber, Para fine, has risen fairly steadily throughout the month, from 5/6 to 6/1 by the 8th, 6/11½ by the 22d, and closes at 6/9.

During the month the following differences are noted:

	Lower. *Amount.*		
Copper	£	9	0
Tin (unsteady)	3	0	0
Scotch pig			6
Cleveland Warrants			3
Shellac		12	0
Higher.			
Lead		2	6
Sulphate of ammonia	1	0	0
Copper sulphate		12	6
Camphor			3
Tin ore	7	0	0
Carbolic acid			4
Para rubber		1	6
Platinum		8	0

The Camp Bird., Ltd., will in the future issue a monthly statement showing the monthly earnings of the company from its Camp Bird mine and from the holdings in Santa Gertrudis, Mexico. For the month of December, 1910, the estimated earnings applicable to profit are $112,161.

Recent Metallurgical Patents.

Gold and Silver.

Antimonial Gold Ore Treatment.—A new proposal for the treatment of gold-bearing antimony ores is made by John Jones and H. S. Bohm, of Mount Morgan, Queensland, Australia. They claim that aside from being effective for the purpose, the process also is economical by reason of the possibility of regenerating the chemicals used.

A hydrochloric acid generator and tower are part of the plant. The first treatment is with HCl, and as the ore is a sulphide the action of the HCl on the ore is to form H_2S. In another operation the chloride solution of antimony is electrolyzed, generating chlorine which is used to combine with the H_2S formed in the first instance to regenerate HCl, according to the equation $H_2S + Cl_2 = 2\,HCl + S$. The regenerated hydrochloric acid gas is returned to the tower for enriching the solutions.

The cycle of operations proposed is as follows: The pulverized ore is agitated in a dissolving vat with hydrochloric acid solution, heated by exhaust steam or other means. The generated H_2S is conducted to a regenerating chamber where it combines with Cl conducted from the electrolyzing cell, to be referred to later. When the antimony has been dissolved, the solution is filtered and transferred to the electrolyzer, fitted preferably with carbon anodes and antimony cathodes. The Cl from the cell is conducted to the regenerator mentioned above where it combines with the H_2S. The sludge from the first treatment tank contains the gold and other valuable metals, and may be treated in any suitable way, preferably by clorination. (984,090, Feb. 14, 1911.)

Copper.

Hardening Copper.—The much sought process for so hardening copper that it can be used for purposes for which it is not now fit is claimed by R. A. Hamilton and Joshua Henry, of Connellsville, Pa. The treatment enables the copper to be manipulated in the ordinary rolling or hammering processes without losing any of its stability or hardness. The hardness may be varied to any degree required. In carrying out the invention the first step is to heat metallic aluminium in a crucible to a high temperature, and add iron pyrites in sufficient quantity to form a mixture that is brittle under the hammer. The aluminium and pyrites are thoroughly mixed while highly heated, and the resulting uniform mixture poured into convenient molds. To harden metallic copper a suitable quantity of the aluminium-pyrites mixture is added to molten copper, together with a small quantity of borax and charcoal. From 1 oz. to 3 oz. of the aluminium compound is used to each pound of copper. Borax and charcoal are added in quantities of ½ oz. each. After the ingredients have been added the crucible is heated until the constituents of the molten mass have become thoroughly combined, when the metal is poured into suitable molds. (984,137, Feb. 14, 1911.)

Neutralizing Smelter Fume.—In a patent recently granted to Clarence B. Sprague, of Salt Lake City, Utah, and assigned by him to the United States Smelting, Refining & Mining Company, is described the method used at the company's plant near Salt Lake City for the purpose of removing sulphuric acid and noxious solid fume from roaster and furnace gases. It has long been known that owing to the presence of sulphuric acid in the roaster and furnace gases from copper smelters the bag-house system of condensing fume was not applicable, owing to the ready destruction of the bags. Mr. Sprague has sought to make these gases amenable to bag filtration by neutralizing the acid therein contained, after which the bag can remove the solid fume, some of which is noxious, as arsenic, and some of which is valuable, as metallic particles. His preferred means of neutralizing the acid is by introducing zinc oxide fume into the gas flues before the gases reach the bag house, although he suggests that the same result can be

accomplished by roasting or smelting with the regular charge a suitable quantity of zinc ore, which in oxidizing will give a basic element to combine with the acid element in the fume to form a neutral salt that will be without effect on the bags. By this scheme it is sought to eliminate trouble from the damage of crops and animals by the discharge of acid gases and noxious fumes onto the agricultural region surrounding the smelter. (984,498, Feb. 14, 1911.)

Aluminium.

Improved Process of Manufacture of Aluminium Alloys.—It has been found difficult to produce a composite metal containing an appreciable quantity of aluminium and having the properties of malleability and ductility. Messrs. J. G. and W. F. Mellen, of Bridgeport, Conn., have, however, recently patented a process which they claim enables them to produce an aluminium alloy with copper or other metal which does have the properties mentioned. Their process gives an alloy with lowered specific gravity and color modified by aluminium. As a general proposition it may be stated that the greater the proportion of aluminium in the alloy the lighter will be the composite metal both in color and weight and also the harder and less malleable and ductile.

In carrying out the process the copper, or other foundation metal, is weighed and melted in a pot. When the temperature has been raised to the proper point a suitable quantity of aluminium is added and immediately after it has melted a suitable oxidizing agent, such as potassium bichromate, and a suitable flux, such as borax, are added in proportions of about ½ lb. bichromate and ¼ lb. borax to each 100 lb. of alloy. A furious action ensues owing to the oxidation of the aluminium, which brings about an intimate mixture of the metals, the union of oxygen and aluminium causing atomic as well as molecular activity. The action of the borax is to dissolve the oxides as fast as they are formed, and in this way the full benefit of the oxygen is used and the objectionable oxides removed. If the oxidizing agent and flux are added before the aluminium the resulting alloy will be harder; if after the aluminium the alloy will be more malleable and ductile.

Variations in the procedure are described, but the main point to the process is the production of an intimate mixture of the elements of the alloy, a failure to obtain which has been the cause of former unsatisfactory alloys of aluminium. (982,218, Jan. 17, 1911.)

Detinning and Degalvanizing.

A cyclic method for removing tin and zinc from tinned and galvanized iron scraps has been patented by Alexander S. Ramage, of Newark, N. J. His preferred procedure is to form a solvent of sodium plumbate by dissolving lead ashes or litharge in boiling caustic soda. This solvent is contained in several tanks. Iron cages containing tin scrap are then lowered into the bath, whereupon spongy lead is thrown out of solution, being replaced by tin, forming sodium stannate. It is preferable to have several tanks of sodium plumbate solvent so that after the operation is under way the clean tin scrap is treated in the weakest plumbate solution to insure the complete precipitation of the lead, and finished in the strongest plumbate solution to insure the removal of the last of the tin. The spongy lead is sold as such for storage battery or other purposes.

The sodium stanate is then used as a solvent bath for galvanized scrap, whereupon similar reactions ensue, precipitating tin as metal and dissolving zinc, forming sodium zincate. The tin is used to make tin salts, while the sodium zincate is electrolyzed in cells, using iron anodes and thin sheet zinc cathodes. All of the zinc is not precipitated, and the solution is used for the further dissolution of lead to repeat the cycle. The inventor states that the presence of sodium zincate does not militate against the successful use of the process. (983,931, Feb. 14, 1911.)

Synopsis of Metallurgical and Chemical Literature.

Iron and Steel.

The Electric Furnace in Siderurgy.—The January issue of the *Proceedings* of the Engineers' Society of Western Pennsylvania contains an interesting and instructive paper by Mr. P. McNiven Bennie on the present status of the electric furnace in the iron and steel industries. A review is first given of the production of pig iron in the electric furnace in California and Sweden, and the principal types of electric furnaces are next sketched. The author distinguishes the following three general classes: (1) arc furnace heating by radiation (Stassano); (2) arc furnaces in which the current passes from the electrodes in the bath (Girod, Heroult, Keller) and (3) induction furnaces (Colby, Kjellin, Roechling-Rodenhauser). The different uses of the electric furnace in the steel industry are enumerated as follows: First, as to high-grade steel, of crucible quality, the electric furnace is designed to become a serious rival of the crucible method, simply because it is less expensive. Secondly, for steel castings, the electric furnace promises to appropriate to itself a considerable field. It cannot and does not aim to displace open-hearth steel castings; but for a grade of castings somewhat better than open-hearth product, cheaper than the crucible product, and for which there is even now a crying need the electric furnace has found a place in which its popularity is certain to increase. Third, the electric furnace is not claiming rivalry with the bessemer or open-hearth furnace as a steel producer for ordinary grades of structural and similar steel. Rather it is to be considered as a useful accessory or ally of the older methods. It offers an opportunity to do additional refining or special treatment in an expeditious manner secondary to other operations. As to the refractories required the author refers to the possibilities of electrically fused alumina,* making a brick that is fairly inactive chemically at furnace temperatures and this material gives great promise. Generally the roof has been short-lived in the larger steel furnaces, probably due to the chemical activity of lime, silica and iron vapors in the furnace. Any refractory material that will stand the high temperatures and prolong the life of the roof will mark another step forward. The author finally points out that in the electric furnace with a non-oxidizing atmosphere chemical reactions are possible which differ from those in the ordinary oxidizing metallurgical furnaces. He refers especially to the removal of sulphur as calcium sulphide and the removal of phosphorus as phosphide. In the extended discussions which followed Mr. P. P. Reese said that "one of our troubles is in keeping the roof on the furnace. We used a 9-inch silica brick and we tried a 13-inch silica brick and we did not get any more life out of the latter than the 9-inch. I think it is due partly to what Mr. Bennie said, the action of the lime fumes on the silica, but I think it is mostly due to the reflection of the arc. We can string the furnace roof in one heat. We do not do it as a rule, because we try to glaze the roof first. But the roof on the furnace is quite a problem just now. With our side walls we have very little trouble." The speaker appeared to hope that the fused alumina brick would solve the problems. As to the cost of power, Mr. N. W. Storer made the interesting statement that he knew of one power plant in operation at the present time where the cost of power, generated by steam, is less than ¼ cent per kilowatt hour.

Gold and Silver.

Effect of Heat on Precipitation.—At the Knights Deep cyanide plant in South Africa waste heat from the power plant is used to raise the temperature of solutions before precipitation in zinc boxes. In the December, 1910, *Journal* of the Chem. Met. and Min. Soc. of South Africa, Mr. F. D. Phillips presents figures showing the improvement in extraction due to

*See article on Alundum Refractories, page 225.

the warming of solutions, especially during the winter months.

Before Utilizing Waste Heat.

	KCN.	NaOH.	Assay value entering boxes, dwt.	Assay value leaving boxes, dwt.	Tons solution per cubic foot of zinc per 24 hours.	Temperature at head of boxes, deg. Fahr.
June, 1908	0.010	0.0140	0.331	0.033	1.58	56.4
July, 1908	0.010	0.0150	0.336	0.041	1.42	53.8
August, 1908	0.011	0.0140	0.357	0.033	1.59	57.2
Average	0.010	0.0140	0.351	0.030	1.53	55.8
After Utilizing Waste Heat.						
June, 1909	0.008	0.0140	0.412	0.010	2.08	73.8
July, 1909	0.006	0.0157	0.381	0.015	2.17	76.3
August, 1909	0.006	0.0156	0.362	0.013	2.16	77.4
Average	0.007	0.0151	0.385	0.013	2.14	75.8

It will be seen that the average temperature was raised 20° Fahr. An important point is the increased amount of solution that was precipitated per cubic foot of zinc in 24 hours and the smaller consumption of cyanide. The flow was greater and the results better with heat than without. The heated solution also was responsible for the comparative absence of white precipitate in the boxes.

Explosion in Amalgam Barrels.—The danger attending the use of barrels in grinding black sand and other rich products in cleaning up the mill is mentioned by W. R. Dowling in the December, 1910, *Journal* of the Chem. Met. and Min. Soc. of South Africa. It seems that at times gas is generated in the barrel and escapes with considerable violence when the barrel is opened. Several unfortunate accidents have occurred from this cause, even after the barrel had been opened for some time. There was still enough gas left to form an explosive mixture when ignited by a candle used to inspect the barrel and its contents.

The cause of the trouble appears to be that gas is generated when acid material containing partly oxidized pyrite is treated. The finely divided iron produced from the cast-iron barrel and the steel balls being ground in the presence of pyrite results in the formation of ferrous sulphide. Ferrous sulphide and finely divided iron acted on by the acid of the charge form sulphuretted hydrogen and hydrogen, according to the equations:

$$FeS - H_2SO_4\ FeSO_4 - H_2S$$

$$Fe - H_2SO_4\ FeSO_4 - H_2.$$

This mixture of gases is explosive, as has been proved when a barrel was inspected after opening, using a naked light. The remedy to prevent excessive pressure is to use sufficient lime with charges to neutralize the acid. Care should be exercised to open the barrel carefully, and if possible it should be possible to open it at two points so that a current of air could circulate through it and remove any collection of explosive gases. It has been found that when an excess of lime is used there is little or no pressure developed; indeed, in one instance there was a slight vacuum.

Copper.

Copper Blast Furnace Tops.—The introduction of acid making at the copper smelting plant of the Tennessee Copper Company has brought about some interesting developments in the construction of blast-furnace tops. The February, 1911, *Bulletin* of the American Institute of Mining Engineers contains an illustrated history of the developments which have led to the adoption of a special top. The article is by Mr. N. H. Emmons, of the Tennessee Copper Company.

The troubles with the old-style brick-top furnace, supported by structural steel, began when the acid plant was connected with the furnace flue and a damper put in the flue to force the gas into the acid plant. For a few months the tops were tight and worked satisfactorily, but it was then found that they would not stand the temperatures, particularly when dampered back to force the gas into the Glover towers. The steel skeleton of the top warped and became unsafe.

The first new top made was low, with brick-lined flues at each end below the feed floor leading to the main dust chamber. The top was made of 18 ribbed copper castings in the form of a grid in which firebrick were laid. The copper ribs projected above the brick and thus provided for radiation of the heat, which at times reached 1800° Fahr. to 2000° Fahr. The whole top was hung from a structural steel frame. The top of this furnace proved to be too low and allowed smoke, fume and flame to shoot out of the charging doors when opened for charging or barring.

The next attempt was similar, but the top was higher, and the flues larger. Above the floor the flue connections were made of water jackets, but these were not at all satisfactory and a connection was made to the main blast pipe and air at 45 oz. pressure blown through the jackets. This answered very well. Both of these types gave satisfaction as far as concerned the temperature of the gases and making repairs, but the gas flues were so small and low that pieces of the charge would be blown into them and cause accretions, which reduced the flue area so that gas and smoke would issue from the charging doors.

The latest development is a tubular-top furnace. It has cast-iron corner posts and dividers with walls and ends laid up with firebrick. The half-circle top is supported on 20-in. I-beams resting on the columns. All metal parts are exposed to the air for cooling. The area of the flues totals 64 sq. ft., being more than twice the area of the flues of the first furnaces constructed. This top is expensive to build, and it is thought it will be expensive to maintain.

Recording pyrometer charts are shown in the original paper giving the temperatures in the horizontal flues from two of the furnaces and at the base of the Glover towers. A peculiar point noted is that the temperature at the towers is higher than in the furnace flues. Since there must be some loss by radiation in the flues the only way to account for the higher temperature at the towers is that the furnace gases contain some unburned sulphur and carbon monoxide which, in burning in the flue, give rise to the higher temperature at the base of the towers. The temperature at the Glover tower is very uniform, showing the equalizing effect of the dust chamber. For the month of September the average gas at the foot of the Glover tower showed 0.16 per cent more SO_2 than the average of furnaces 4 and 5, and a drop of temperature of 52° Fahr. below the average of the furnaces recorded at the furnace top.

Reverberatory Furnace Hearth Construction.—We take the following description of the construction of a chrome bottom from an article entitled "Engineering in Metallurgy," by Messrs H. Schroeder and W. E. Longworth in the *Australian Mining Standard* of Feb. 1, 1911. The entire article runs through several issues and gives a complete engineering description of the works of the Electrolytic Refining & Smelting Company of Australia.

The reverberatory smelters are 34 ft. x 17 ft. at the hearth. The walls and casing plates are carried on concrete foundations 2 ft. 6 in. deep, a layer of concrete 12 in. deep being laid under the hearth. A course of red brick, laid flat, covers the concrete, and on this is placed a layer of "brasque," 6 in. deep at the center and shaped according to the template of the hearth. The brasque is a mixture of fireclay, crushed coke and sand, moistened and tamped into place. On this a 9-in. course of local firebrick was laid, followed by a 9-in. course of Scotch firebrick. The furnace walls were then built up, the first three courses being stepped out 1.5 in., the following three being stepped back, and the wall then worked straight up to the roof. By thus stepping the walls out a ring was formed to hold down the working bottom, which was a mixture of chrome iron ore and fireclay, tamped to a thickness of 6 in. in the center

and shaped according to the template, the invert being 8 in. at the center.

The chrome bottom was put on thus: Chrome iron ore, crushed to pass a ⅛-in. screen, was mixed with fireclay in the proportion of 3 to 1, by bulk. This was mixed with water to the consistency of a cupel mixture, spread over the bottom in ½-in. layers and tamped with hot irons. The resulting layer was only about ⅛ in. thick, so that 48 layers had to be put on to give the necessary thickness of 6 in. It required seven men, working eight hours per shift, 13 shifts to put in a 34-ft. x 17-ft. chrome hearth, building the sides about 9 in. higher than the center. Seventeen tons of crushed chrome ore and 3.5 tons of fireclay were used. The cost of the labor was about $170. Such a bottom, however, once made is good for all time, and there is practically no absorption.

Electric Furnaces.

The Pinch Effect for Agitation and Proportioning of Electrodes for Electric Furnaces.—A French patent of Mr. Carl Hering (French patent 419,715, June 28, 1910, under international convention, July 6, 1909) is abstracted in the *Journal* of Society of Chemical Industries, February 28, as follows: "The furnace has a bath of melted material and an electric current, proportioned with reference to the transverse section of the material is passed through a mass of molten material in communication with the bath, thus producing in an automatic manner a continuous agitation of the material, owing to the so-called 'pinch' effect. The electrode is provided with a resistance (meaning evidently the dimensions of the electrode are chosen so that it has a resistance), such that the heat developed in the electrode by the current transmitted through it is equal to double the loss of heat by conduction through the electrode when the current is not passed. (In this case practically no heat will pass in either direction through the electrode and within the furnace.) In a modified form of the process one or more arcs are introduced into the furnace between parts of the melted material."

Amorphous Carbon Electrodes and Brushes.—British patent 3336, March 2, 1911, of the British Thomson-Houston Company (communicated by the General Electric Company of this country) is abstracted in London *Electrical Engineering* of March 9 as follows: "Brushes or electrodes of increased density and mechanical strength and of high conductivity are produced by moulding from a mixture of carbon and a carbonaceous binder and baked to a temperature of 480°C. They are then impregnated with pitch or other suitable material and fired to a temperature of about 1,400°C. The following result is given as an example: apparent density, 1.66; absolute density, 2.05; porosity, 19 per cent; resistance, 0.00108 ohm per inch cube; tensile strength, 2,130 lb. per square inch.

A revolving reverberatory copper furnace has been patented by J. A. Brown, of Sydney, Australia. It consists of a barrel-shaped furnace, with firebox or gas producer at one end, and an exhaust gas chamber at the other. The exhaust gases are used to preheat air, which is conducted to the fire end of the furnace, where it is mixed with the fuel gas and directed in a blast onto the ore charge in the furnace. As the furnace revolves, every part of the charge is exposed to the heat. The charge is fed at the fire end, and the slag flows into a settler placed in the exhaust gas chamber at the other end. A tap hole is placed in the side of the furnace at its widest part, and through this the accumulated matter may be tapped.

Bromo-cyanogen as an adjunct to the regular cyanide process is prepared, according to Australian practice, by first dissolving the bromine salts in water, then adding the potassium cyanide and finally, when all the salts are dissolved, sulphuric acid is added and the mixture stirred for several hours. The charge is then ready to be added to the pulp for extraction of the gold.

Smoke Abatement Lectures at Liverpool, England.

A course of evening lectures and demonstrations for persons in charge of the furnaces of boilers and other heating appliances was commenced in Liverpool on Friday, Jan. 27. Dr. Hope, medical officer of health for Liverpool, and Professor Watkinson, professor of engineering at the Liverpool University, co-operated in arranging this course of instruction with reference to smoke abatement. The fee for the course of six lectures is 5 shillings, and 63 students, chiefly working firemen and engineers, have registered their names and are now attending these lectures. The lectures are delivered at the School of Hygiene, and are under the control of the sanitary science instruction committee of the Liverpool Corporation.

The first lecture of the course was delivered by Dr. W. L. Hawksley, one of the assistant medical officers, who dealt with the importance of pure air in relation to the health of the community and with the reforms necessary in order to obtain a purer atmosphere. Commencing his lecture with a reference to the select committees appointed in 1819, in 1843, and again in 1845 to inquire into this question, and to their findings and recommendations, Dr. Hawksley passed on to consider the reports of Dr. Lyon Playfair and Sir H. de le Beche, and the results of the first Smoke Abatement Exhibition of 1881.

The question of domestic smoke was then dealt with by the lecturer, and the connection between smoke and fog was traced and emphasized. The lecturer explained the differences between town and sea fogs as being due to the dirt and soot and tarry vapors suspended in the former. The effects of a smoky atmosphere upon health were next discussed, the differences between healthy and unhealthy persons in this respect being pointed out. The damage done by smoke to the stonework of public buildings was also touched upon, and the monetary loss this entailed was emphasized. Dr. Hawksley closed an interesting and instructive lecture by giving an account of the aims and work of the Smoke Abatement League of Great Britain, and with a summary of the speeches made by the deputation arranged by this league, which waited upon Mr. Burns at the local government board's offices in London on June 29, 1910.

The second lecture was delivered on Feb. 3 by Mr. John B. C. Kershaw, who took as his subject "solid fuels and the scientific principles of combustion."

Dealing first with the origin of solid fuels the lecturer stated that these had been formed by the decay of wood and other vegetable matter in the presence of moisture and heat, great pressure also having been gradually added to these two agencies in the latter stages of the process of conversion. An interesting table showing the chemical composition of wood, peat, lignite, bituminous coal, semi-anthracite and anthracite was exhibited, and it was pointed out by the lecturer that during the successive stages of the gradual change from peat to anthracite the carbon in the fuel increased from 52 per cent up to 94 per cent, while the gaseous constituents—nitrogen, oxygen and hydrogen—diminished in similar proportion. Reference was then made to the formation of the coal fields of southwest Lancashire and to the clear proof which existed that the basin of the Mersey was once a vast inland swamp, with a belt of forest land stretching across what is now the Mersey Bar. The remains of this forest are still to be found on the shore near the surface at Hoylake and at Formby.

The changes which occur when bituminous coal is heated were next discussed, and it was stated that the large volume of hydrocarbon gases which are liberated from this class of coal forms the chief difficulty in burning bituminous coal without smoke, since great absorption of heat occurs during this stage of the combustion process, and the liberated gases require careful and scientific management to avoid smoke formation.

The three essential conditions for obtaining the smokeless combustion of bituminous coal were stated to be: (1) A sufficiency of air supply; (2) a sufficiently high temperature;

and (3) a good mixture of the air and of the hydrocarbon gases.

It was pointed out that, although a Liverpool engineer, C. Wye Williams, clearly demonstrated that these conditions were absolutely necessary 70 years ago, the designers and builders of boiler furnaces to-day still too often neglected them and designed and erected furnaces in which one and often all of these essential conditions of good combustion were absent.

The causes of factory smoke were then stated to be: (a) Too little boiler power, leading to overworked boilers; (b) unscientific management of the boiler fires; (c) neglect or ignorance on the part of the designers of the furnace of the essential conditions of good combustion.

The lecturer closed by stating that all the smoke arising from the use of coal for industrial purposes might be abated, and in time gradually abolished, if those who control our works and factories would study the matter scientifically, and see, as far as possible:

(1) That they possessed adequate furnace and boiler equipment for the work required.

(2) That the furnaces were properly designed and that no cold or water-cooled surfaces came into contact with the hot gases before combustion was completed.

(3) That the boiler or furnace plant was under proper scientific control.

(4) That the firemen were well paid and were given an interest in their work by payment of a bonus for good results.

(5) That the chimney top was visible from the firing places of the furnaces or boilers.

The lecture was illustrated by experiments.

The third lecture of the series was delivered on Feb. 10 by Mr. Kershaw, his subject on this occasion being the examination of the waste gases and control of the combustion process.

Prof. Watkinson, Harrison professor of engineering at Liverpool University, delivered the fourth of the lectures for stokers and others interested in furnace management in the Lecture Theater of the Walker Engineering Laboratories at Liverpool, on Friday evening, Feb. 17, taking as his subjects, the setting and construction of boilers and furnaces, and the methods of producing natural, forced and induced draught.

After a few opening remarks dealing with the general aim and purpose of the lectures, and with the importance of accurate measurements of draught, temperature, pressure and of the composition of the exit gases as an aid to good work in boiler management, the lecturer dealt with the three conditions (see lecture II above) requisite to obtain good combustion of solid bituminous fuels and showed experimentally how essential these were to smokeless combustion.

The details of construction of internally fired or Lancashire boilers were than illustrated by aid of lantern slides to prove the difficulty of obtaining smokeless combustion due to the small furnace and to the arch of water-cooled plates over the furnace grate. The purpose of the bridge in this type of boiler was then discussed, and it was proved that it produced eddies in the gases as they passed over it into the flue beyond and therefore tended to promote mixture of the hot air and hyrocarbon gases given off from the burning fuel. A high bridge was, therefore, better than a low one, if sufficient draught could be obtained to work the boiler fires with it. In the absence of good draught the use of steam jets was often resorted to in order to (1) increase the air supply, and (2) to promote the better mixture of the air and furnace gases.

In the Bellville type of marine boiler air at 30 lb. pressure was employed in place of steam, and the Howden system of forced draught was now generally employed for marine work, but this system has not been adopted to any extent for boiler installations on land.

Natural draught produced by a chimney rarely exceeded ½ in. to ¾ in., measured by a water gage. This low draught limited the thickness of the fires and rendered it exceedingly difficult to keep the fuel on the furnace bars free from holes. For good combustion with thick fires, on the other hand, forced or induced draught was essential, and this was now generally recognized and adopted.

The different methods of obtaining artificial draught were then discussed and the comparative advantages of steam jets, air jets and fans were dealt with by the lecturer. As compared with steam jets, fans were more costly to install, but saved largely in running costs—a good fan requiring only 5 per cent of the steam produced in place of the 10 per cent to 12 per cent used by steam jets. Makers' statements that steam jets only consumed 3 per cent of the steam were said to be absolutely inaccurate.

The different methods of furnace construction and the use of firebrick arches for conserving heat and promoting good combustion were then dealt with. It was pointed out that a firebrick arch, by increasing the heat of a furnace, may actually increase smoke production, owing to the greater rapidity with which the hydrocarbon gases will be evolved from the newly charged fuel, unless precautions are taken to greatly increase the air supply at the same moment.

The Dutch-oven type of furnace construction for Lancashire boilers was condemned by Professor Watkinson, because from 40 per cent to 60 per cent of the heat transfer in this type of boiler is by direct radiation to the plates from the glowing solid carbon lying in the bars of the grate and this radiation can only occur to the full extent when the furnace is inside the boiler. Luminous flames radiate heat also, but not to the same extent as glowing solids. Although steam boilers can be worked efficiently with gaseous fuel they require to be specially constructed for this duty, and no ordinary type of boiler will give high efficiency with external furnaces or with producer gas. The use of economizers was finally discussed and the two chief types were described.

The lecture was illustrated by numerous experiments and lantern slides and was followed by an exhibition of apparatus for making draught, temperature and other measurements incidental to good boiler management.

On Wednesday, Feb. 22, a large number of the members of the class paid a visit to the boiler installation at Messrs. Bibby's Seed-Crushing Works in Liverpool, under the guidance of Mr. Kershaw and Chief Smoke Inspector J. Macaulay.

A Novel Development in Laboratory Burners and Furnaces.

By Chester G. Fisher

No piece of apparatus has become so commonplace in the laboratory as the burner, yet it has gone for years practically unaltered. This is really remarkable, when we consider that thousands of chemists and metallurgists have time and again tried to "push" the burner in an endeavor either to hurry an operation or to obtain a little higher temperature than the design of the burner permitted. It has remained for Dr. Méker, a French scientist, to question the use of the Bunsen burner and to prove it both inefficient and extravagant.

The ideal burner must produce the hottest flame possible with minimum consumption of gas, which flame should also be homogeneous. In order to do this, problems similar to those which present themselves in the gas engine must be solved. First, the valves and passages must fill the cylinder with a homogeneous mixture of gas and air in the proper proportion for perfect combustion. Second, all obstacles which retard the operation must be done away with, in order to allow the burnt gases to quickly pass off and the cycle to be repeated immediately.

A study of perfect mixtures soon showed quite a difference in the proportion of air, depending upon the kind of gas used, which settled the fact that burners of different designs must be used for artificial, natural and gasoline gases. A burner with an adjustable gas supply, such as a needle valve, would

not do, as the proper adjustment would be guesswork; furthermore such a valve would throttle the gas supply badly. Experiments proved that six parts of air to one part of coal gas made a perfect mixture, while the proportion prevailing in the ordinary Bunsen burner was two and one-half parts of air to one part of coal gas.

It might be explained that six parts of air to one of coal gas could not be used in the Bunsen burner, as this is a highly explosive mixture and the flame instantly "fires back." The lack of sufficient air for complete combustion in the Bunsen burner accounts for the cold cone in the center of the flame, as the extra air necessary is only obtained on the outside of the flame.

Dr. Méker, after experimenting with innumerable designs and after much painstaking work, finally perfected a burner using the perfect mixture. In his burner the gas enters the bottom and the air at the sides through many large inlet holes, the

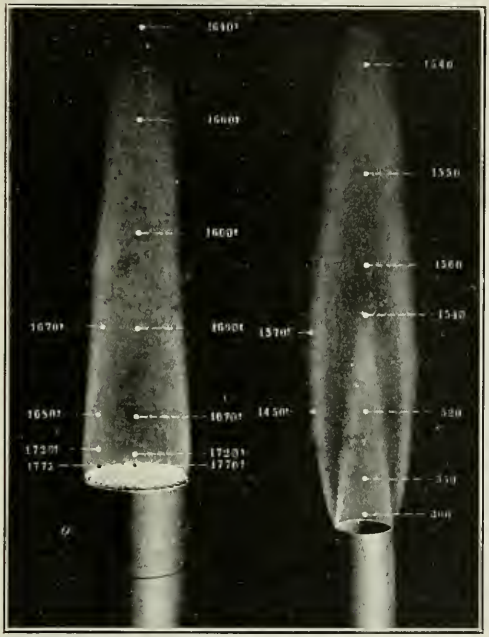


FIG. 1.—COMPARISON OF MEKER BURNER AND BUNSEN BURNER

two are thoroughly mixed in the special shaped tube and pass up at a high rate of speed. The "back firing" of the flame is prevented by a nickel grid at the top of the tube; this grid is about 10 mm thick and has openings about 2 mm square. The openings being large do not impede the flow of gas, yet the flame cannot flash back, as the deep grid is comparatively cool on account of its depth.

The result is a large homogeneous hot flame made up of numerous small flames, and the unusual effect is noted by comparing the temperatures of the different parts of the Méker flame with that of the Bunsen flame, as shown in Fig. 1. The temperatures given in this photograph were measured with a Le Chatelier thermo-couple 2/10 mm in diameter.

It will be observed that the coldest part of the Méker flame is hotter than the hottest point of the Bunsen flame, that the Méker flame is almost uniform, and that the same very high temperature prevails across the large diameter of the flame. The uniform flame produced by the Méker burner has still another great advantage in that it does not produce irregular expansion in glass or porcelain vessels being heated and, therefore, greatly reduces the risk of breakage. From the above photograph one can readily understand why a large-size Méker burner will replace a Bunsen blast lamp.

Dr. Méker was not satisfied with designing the improved type of burner but also designed an equally efficient furnace with which to use these burners. The combination of an ordinary Méker burner and a Méker crucible furnace enables gold, silver and copper to be melted without the aid of compressed air and this is done as readily as water is boiled with an old-style Bunsen burner.

It naturally followed that if such a large increase in efficiency could be obtained without blast an equally large increase could be shown over the Bunsen blast lamp when compressed air was used with the Méker burner. Dr. Méker also designed along the same scientific lines a burner using compressed air, with the result that this burner of ordinary size melted 105 grams of platinum in 17 minutes.

In combining the Méker blast burner with a furnace it was found that special high refractory clay had first to be prepared for use in lining the furnaces.

Melting Gold, Silver and Copper Without Blast.

Fig. 2 shows a plumbago crucible, which has been heated in a specially designed furnace, wherein the flame from the Méker blast lamp is made to circle around the crucible. The plumbago crucible (Fig. 2) shows the blistered effect due to the boiling out of the clay used in its construction. The four

FIG. 2.—TEST OF PLUMBAGO CRUCIBLE.

pyrometric cones shown beside the crucible were in it when it was heated. It is seen that three cones have fused, including one melting at 1770° C., and the only one standing had a fusing point of 1810° C.

The results of official tests made by the Société Technique de l'Industrie du Gaz are as follows:

Test No. 1: Consisted of measuring the consumption of gas necessary to bring to a boiling point 6 liters of water, with a Méker and with an ordinary Bunsen burner, both regulated to about the same gas consumption per hour. With the Bunsen burner the boiling point was reached after 27 minutes, 40 seconds, and the total consumption of gas was 305 liters. The Méker burner heated the same volume of water to a boiling point in 22 minutes, 30 seconds, and consumed 223 liters of gas. This shows a result in favor of the Méker burner of 23.3 per cent in time consumed and a saving of 36.7 per cent in gas by the use of the Méker burner.

Test No. 2: Consisted of bringing a crucible up to a high temperature, at first with a Bunsen blast lamp (gas and air), afterwards with an ordinary Méker burner without air blast. The temperature obtained in both cases was 1060° C. The Bunsen blast lamp consumed 543 liters of gas, while the Méker burner consumed only 273 liters. The temperature obtained was measured with a thermo-electric couple and a standardized

galvanometer. This shows a waste of 98 per cent in gas consumption by the blast lamp over the Méker burner, with considerably less labor for the operator of the Méker burner and greater regularity of temperature, arising from the discarding of the air blast.

Other burners of different shapes and designs are made, in order to meet the conditions ordinarily found.

In view of the large amount of work done by Dr. Méker in perfecting these burners, it was very fitting that the Société Technique de l'Industrie du Gaz concluded its report as follows: "Considering that the burners of Dr. Méker permit of working in ranges of temperature formerly impossible to obtain, and thus contribute, both in the laboratory and in the industrial world, to the increased usefulness of gas, the committee has decided to reward the inventor by awarding him the diploma and silver medal of the society."

These burners and furnaces are now placed on the market in this country by Dr. Méker's American agent, the Scientific Materials Company, Pittsburgh, Pa.

Thermit Repairs of Caustic Pots.

By Thos. M. Skinner, Jr.

In the repairs of caustic pots by the thermit process it is of great importance that an alloy be used which will not be affected by the sodium hydrate which is fused in these pots, and for this purpose I find 50 per cent ferro-silicon to be very satisfactory, as it makes the thermit steel in the weld resemble the cast iron of which the pots are made. In using ferro-

FIG. 1.—CAUSTIC POT SETTING, DIRECT FIRED WITH SLACK COAL.

silicon, 3 per cent based upon weight of thermit has proved the most satisfactory proportion.

Defects in these alkali pots usually develop at a distance about one-third up from the bottom, where the heat is chiefly applied. This position makes it very difficult to place a form to handle the thermit pour. An additional form is necessary on the outside of the pot amply large to cover all possibilities of the thermit spreading in its molten state and thereby missing the form.

A very good method which has been tried and found successful is to build the inside form in which the pouring is made by filling the hole as usual with wax; then as the pot is of circular shape, take a piece of asbestos (½ in. thick) cut about ½ in. larger than the hole to be mended and laid over wax and hole; then places are cut for pouring hole, riser, and preheating hole as wanted and wooden plugs placed where said holes are desired. Then the form is filled with fireclay, pounded firebrick and sand in equal parts. The wax is melted out by a torch and is caught below as it comes out of preheating hole. The sheet of asbestos serves not only to keep the fireclay from making an uneven surface, but gives a clean, smooth job without any slag adhering, which would compel a great amount of chiseling afterwards.

By this thermit process of mending our caustic pots we have been able in many instances to make worthless pots practically as good as new, thereby extending in a very economical way the life and usefulness of these pots.

Green River, Wyo.

The Development and Varied Application of Oxygen Rescue and Reviving Apparatus for Mines, Metallurgical and Chemical Works

By Dr. Henry M. Payne.

When the need of conveniently portable rescue apparatus first became apparent in the mining districts of Europe several types were developed, in all of which great difficulty was experi-

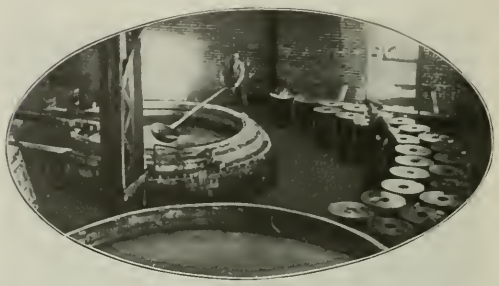


FIG. 2.—CAUSTIC POTS BOILING, SHOWING METHOD USED IN HANDLING HOT CAUSTIC INTO DRUMS READY FOR SHIPMENT.

enced in preparing a regenerator whose chemical action should provide for perfect absorption of CO_2 and at the same time maintain a temperature sufficiently low as to cause no discomfort to the wearer either from exterior contact with the apparatus or from hot air provided for respiration.

In the early forms a fluid solution containing caustic potash was used. The oxygen was exhausted from the bottle by the efforts of the wearer. The next improvement was the use of

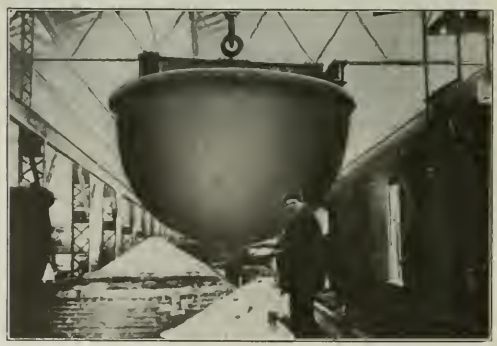


FIG. 3.—CAUSTIC POT BEING LIFTED BY ELECTRIC CRANE IN PREPARATION TO A THERMIT POUR.

flax or cotton waste, which was saturated with the regenerating compound and the compression of the oxygen into iron tanks. The great danger in both these equipments lay in the wearer's inability to regulate the flow of oxygen and to avoid a cumbersome regenerator of irregular and uncertain capacity.

The first type to merit general recognition outside of Austria had a regenerator for which the filler had to be prepared

at the time the apparatus was to be used, a proceeding which entailed great delay and serious loss of life, since it is a well-recognized fact that such apparatus is of the greatest service only when it is immediately available upon the first advice of a catastrophe involving loss of life from asphyxiation of any kind.

Attention was then directed to the preparation of a regenerator whose absorptive capacity should be constant for a given period, and which could be sealed and delivered in perfect condition at any point, ready for immediate use on breaking the seals.

This having been accomplished, effort was concentrated toward providing the maximum area of absorptive surface in order that less heat should be generated and so that a cooler, hitherto necessary, could be dispensed with.

Various shapes of oxygen bottles were tried, but it is now the practice to use the standard cylindrical type so common in all medical apparatus utilizing oxygen.

The elimination of reducing and by-pass valves and the continued simplification of the apparatus has eventually evolved a complete, light weight, "foolproof" and thoroughly reliable form, known as the Westfalia Rescue Apparatus. This type is largely in use by the United States Bureau of Mines, having recently been furnished for the Rescue Station at Pittsburgh and the new station at McAlester, Okla., and on several of the Mine Rescue Cars now being equipped for use in Wyoming, Missouri, Oklahoma, Arkansas and Pennsylvania.

The Illinois Mine Rescue Station Commission are likewise equipping their new stations and three cars with a number of sets of this type.

Until very recently the helmet was supposed to be a necessity, but the Westfalia mouthpiece type now on the market has proved equally safe and possesses many advantages over the helmet.

Earlier and all other types of helmet depend upon the inflation of pneumatic pads for making a tight fit around the face and head, whereas in the Westfalia apparatus an elastic face washer is provided which fits any inequality or irregularity and enables any person to wear the helmet without the usual difficulties of securing a good fit and renders its continued use a matter of no discomfort. These face washers are also provided with an inflating bulb and pneumatic pad, but the safety of the helmet in no wise depends upon the pneumatic pad, since even if it is punctured or the rubber old and weak, the face washer continually conforms to the head and has no angles or creases. It is also easily removed from the helmet and replaced.

The mouthpiece type referred to may be instantly substituted for the helmet and requires no change of apparatus. The nose clip, whose chief danger lies in the laceration of the nose, or its accidental removal by sneezing, and general discomfort, is also obviated with the Westfalia mouthpiece and in its stead cotton plugs, dipped in vaseline, are inserted in the nostrils and an ordinary guard worn over the nose.

A speaking diaphragm attached to the mouthpiece enables the wearer to carry on conversation without difficulty. The regenerators are also furnished in one-hour, two-hour and three-hour types, so that for practice purposes the smaller ones may be used, while in emergencies the three-hour type provides 50 per cent greater capacity than any other. These regenerators being curved to fit the back, the framework leaving an air space between, and the total thickness being reduced to a minimum, the wearer is enabled to crawl through a smaller space than has heretofore been possible. They are also made interchangeable, thus affording owners of older types of other makes of apparatus all the benefits to be derived from the improved regenerator.

The field in which this apparatus is used is not confined to mines, but is also rapidly extending to chemical works, smelters and in extinguishing fires where dense or dangerous gases, smoke or fumes are encountered.

Not less worthy of consideration is the reviving apparatus which is applied to persons asphyxiated by gas or overcome by smoke or water, and which is so commonly used in hospitals for extreme respiration. In this, also, simplicity has been the aim and the "Dr. Brat" system is supplied in a light portable case ready to be instantly and intelligently applied by any person who can read the indices on the operating lever.

Until very recently such apparatus was operated by clockwork, but the chief objection to this lies in the fact that an unconscious or semi-unconscious person breathes in gasps and with extreme irregularity, and any attempt to force oxygen into or exhaust smoke or gas from the lungs with clockwork regularity must inevitably tend to choke the patient. By eliminating the clockwork attachment a light breathing bag is provided for observation and the inhalations and exhalations of the patient are assisted by a slight movement of the lever, which first exhausts the lungs and then injects oxygen, both of which processes are performed under any desired pressure or vacuum up to 2.5 atmospheres.

All this apparatus is on exhibition at the offices of the Westfalia Engineering Company, 42 Broadway, New York City.

Alundum Refractories.

The development of electrically-heated laboratory apparatus has been very great during the last two years, and to meet this demand the Norton Company, of Worcester, Mass., and Niagara Falls, N. Y., has been making use of the refractory qualities of its abrasive product, alundum, made up into the

FIG. 1.—ELECTRIC FURNACE PARTS OF ALUNDUM.

various shapes required in crucible, muffle and tube furnaces. This has led them into other lines, and now this company is offering a general line of alundum refractories of a high grade, comprising not only the articles for winding with high-resistance wire, but also general utility muffles, crucibles, furnace parts and bricks.

The properties which have made it superior for these pur-

FIG. 2.—ALUNDUM TUBES.

poses are high melting point, high heat conductivity, low coefficient of expansion and resistance to chemical action.

Alundum is fused aluminum oxide and is made by melting and purifying the mineral bauxite in a special electric furnace, which, together with the process of manufacture, has been described before in this journal. It is primarily an abrasive material, but the properties mentioned above make it useful as a refractory. There are several grades for abrasive pur-

poses, but only two of these are used in this work. A white product having less than 1 per cent of impurities is used in all the smaller articles, while a brown material with 6 to 8 per cent of impurities is used in larger shapes and in bricks.

The articles so far turned out are mostly porous, consisting as they do of more or less finely-divided grains, held together by a ceramic bond. This porosity renders them unfit for any work in which a gas-tight chamber is required, but on the other hand is very advantageous in another way, for the porosity can be varied to such a degree that crucibles and dishes may be used for filtering purposes. The Gooch type of crucible works perfectly on all precipitates met with in analytical work. It is a great time saver in the laboratory,

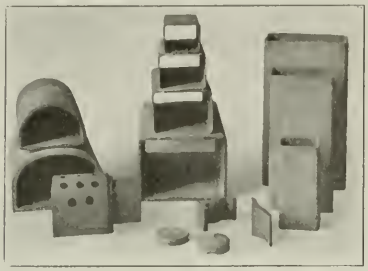


FIG. 3.—ALUNDUM MUFFLERS

as it does away with burning filter papers or preparing asbestos blankets. The crucibles are quite unattacked by aqueous acids or alkalies, so that almost any analysis may be made in them safely.

The mufflers last from four to five times longer than the ordinary fire-clay type, when used in a gas furnace, and when wound with resistance wire are almost the only kind that can be used. The builders of this latter type of furnace use alundum cores exclusively in all shapes and sizes of furnaces.

Combustion boats made of alundum may be used repeatedly in steel-works analysis, as the oxidation products do not stick in the boat or attack it. If other boats are used they may be protected by a layer of carefully-prepared carbon-free alundum grain, which is put up in convenient-sized packages.

Alundum cement may be used for lining crucibles and small furnaces, as it has the same composition and refractoriness as the made-up forms. It is sold, ready for use, except for moistening with water.

The crucibles of denser structure may be used for melting metals but are attacked by slags. The melting point being

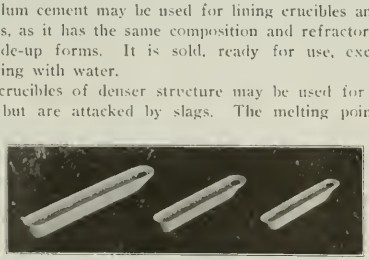


FIG. 4.—ALUNDUM COMBUSTION BOATS

very high, about 1950 deg. C., and also sharp without gradual softening, makes them especially well adapted for melting the metals of the platinum group, and the purity of their composition renders the melt free from contamination.

Bricks have been made experimentally, but the high cost of manufacture and their great weight makes them available only in special cases. For roofs of electric furnaces they are well adapted, on account of high melting point and low coefficient of expansion. Some trials have been made and others are under way to thoroughly test their value for this work. Announcement will be made of the outcome of the trials when they are completed.

Some attempts at casting alundum articles from the molten state have resulted in partial success. On account of the extreme difficulties in the way, only tubes and small crucibles have been prepared, and these are rather crude in appearance, the hardness of the product making it well-nigh impossible to work them into any smoothness after they are cast.

Fig 1 shows various electric furnace parts made of alundum. The high thermal conductivity has here the advantage that the heat generated in the resistance wire is conducted away so rapidly that overheating and consequent burning out of the resistance is prevented. At the same time the chemical inactivity prevents corrosion of the heating wire. By the use of alundum cement, the element can be completely encased, preventing oxidation and corrosion.

Fig. 2 shows a variety of alundum tubes for high-temperature electric-furnace work.

Fig. 3 shows a variety of alundum muffles for furnaces heated by electricity, gas or gasoline. These alundum muffles last from four to six times as long as the ordinary clay muffles.

Fig. 4 shows alundum combustion boats, which, while adapted to any kind of combustion work, were especially designed for the determination of carbon in iron and steel. The pure alundum of which they are composed does not react with the iron oxide of the sample, and the boats can be used repeatedly.

A large display of the shapes already made will be on exhibition at the Chemists' Club, in connection with Mr. L. E. Saunders' paper at the April meeting of the American Electrochemical Society, and some demonstrations of the advantageous properties will then be arranged. The variety exhibited will show the possibilities of moulding, and special shapes may be made to order, of almost any degree of intricacy.

The articles are on sale at the Worcester, Mass., plant of the Norton Company, and descriptive literature and price lists will be sent on request.

Fused Quartz and Quartz Glass.

By Charles Colne.

From time immemorial quartz has occupied a well-known and useful position in the manufacture of glass. In the shape of white sand or quartz crystals it enters in the proportion of about 60 per cent in the composition of the different qualities of glass, for bottles, windows or hollow ware. Lime, potash and soda are used in order to obtain a fusion at a low temperature, bordering on 1300°, to treat the mass in a refractory furnace heated by coal or gas. Such additions, however, alter the qualities of quartz.

It has been discovered long ago that in melting quartz alone at a temperature above 2000° a transparent body is obtained which is insoluble in acids and is an insulator of a superior quality to common glass. Such a product, sometimes erroneously called quartz glass (because it is not glass, but pure silica), has been manufactured commercially for several years with the oxyhydric blow pipe or in the electric furnace. Before these processes reached a practical point inventors found themselves beset by many difficulties. The methods used were very complicated and it is only since the World's Fair of 1900 that anything like success has been reached, as was shown when the firm of Schott exhibited quartz plates produced by fusion.

Inventors were then restricted to the use of the gas blow pipe as the only means of obtaining the needed temperature. Mr. Shenstone's process was to heat Brazilian quartz to a very high temperature, then throw it into water. Crystals would then divide into small fragments; these were agglomerated

in the shape of rods and fused by the blow pipe into thick thread. These threads were wound up spirally around a large platinum thread and again submitted to fusion, producing small tubes of regular shape. These tubes closed at one end, were drawn out and blown into small balls with which small capsules and bulbs were made.

This method later on was improved by preparing directly small, thick, short tubes in an iridium or zirconium capsule, then drawing and blowing them. The cost price of such articles was very high, as high as $110 per pound. Moreover, these objects had very thin sides, were very frail and of necessity of restricted dimensions. They were, however, very transparent—of a transparency that could not he reached with other processes.

Dufour, Callender and Le Chatelier in their experiments have shown that the fusing point of quartz is very near its ebullition. This explains the difficulty of working the melted mass, or more properly, the pasty mass of quartz which has passed from the crystalline state to the amorphous state. A melted mass of this nature shows a tenacity like that of hot asphalt and work must be carried on rapidly, as when the source of heat has been removed solidifying takes place at once. Such extraordinary viscosity almost prevents the expulsion of bubbles which are formed during fusion. In order to get rid of these bubbles a preliminary baking or fritting of the quartz is necessary. Other efforts have been made to obtain fusion under pressure, but apparently with no practical results.

It is only since 1901 that the first attempts have been made to make glass in the electric furnace. A German firm, Becker & Company, of Berlin, appears to have been among the first to make such trials. Since that time these furnaces have been much used for making quartz glass and the result has been to develop the different methods now in use. The active part of a blow-pipe flame is in fact too restricted to allow large quantities of quartz or quartz glass sand to be treated. As soon as the mass to be treated exceeds a few pounds a mass is obtained without transparency looking like porcelain.

The electric furnace did not give satisfactory results either in the first experiments. Trials made to melt glass sand with an arc furnace resulted in getting ingots of glass as large as a fist with a silvery iridescent surface. To bring such ingots to the shape of a receptacle the resistance furnace had to be brought into play. The product, however, was impure owing to the contact of the quartz with the carbon electrodes and the objects made were covered with a coat of silicate of carbon.

Since that time it has become possible to melt white sand completely in the resistance furnace of Borchers. The quartz blocks obtained are drawn so as to make opaque tubes long enough and free from impurities. This process consists in using a furnace in which in a space left vacant between the terminals is placed a tube of thin carbon perforated like a sieve and surrounded by the sand; this tube is heated by the current passing through it and fuses the quartz. Compressed air is then forced through this tube in order to expand the quartz cylinder which has been formed against the sides of an appropriate mold.

The quartz so melted, owing to the high temperature of the incandescent carbon, passes from the crystalline to the amorphous state. It is capable of standing a very high temperature and has a very high insulating power; but it is opaque and slightly inferior, in resistance to sudden changes of temperature, to quartz glass melted with the Shenstone and the Heræus blow pipes. However, the latter process cannot make large pieces fit for industrial uses.

A German company has succeeded in producing transparent quartz glass by the following process in the electric furnace: The sand of the glass maker is introduced into a carbon cylinder, used as a crucible, in the center of which is placed a rod of carbon heated by the electric current at the same time as the outside cylinder. Starting with glass sand, of ordinary quartz, a complete fusion is reached, which, when renewed several times, will produce a transparent mass. To work the latter with a view of getting small objects for laboratory use this can be done either in an electric muffle or with the oxy-hydric blow pipe.

As the fusion of quartz and its transformation into the amorphous state require much heat these two successive operations produce objects more rapidly and more economically; the blow pipe serving only for fining and working the already melted quartz in the shape of tubes. Large pieces are blown in a special manner. Instead of using compressed air or steam, which requires a preliminary preparation, or the introduction of the dose in the quartz bar, which is a difficult operation, owing to the rapid solidifying of the mass, there is introduced in the quartz block by the opening left by the central electrode a moistened body which is transformed into steam—some green wood, for instance, or a substance such as marble, which at the high temperature to which it is subjected decomposes and liberates a gas. During that time the ends of the block are held by pincers. The liberated gases force the quartz against the sides of the molds having the desired shape. The mass when solidified is withdrawn from the mold and the objects are then cut to the desired dimensions with diamonds or by fragments of carborundum mounted on disks of copper or zinc. They are then polished.

To strengthen the objects obtained the tubes can also be blown into metallic woven fabrics. It is with such a process that large capsules of 120 in. are obtained for evaporating, and tubes more than 40 in. long and 10 in. in diameter for use as receptacles for acids. The product thus obtained can be substituted for china and stoneware.

The properties of quartz glass have been investigated by sundry experimenters, especially by Le Chatelier, Shenstone and Gaurties; all have shown how raw quartz not melted is sensitive to variations of temperature; when heated suddenly it breaks in a number of fragments. Fused quartz, on the contrary, which has passed from the crystalline to the amorphous state, resists quite well such changes, especially in a thin shape. Its coefficient of expansion is about 0.0000007 per degree Centigrade, much less than that of platinum. It is interesting to note that the expansion of rods of quartz is quite regular up to 1000° C., but ceases at 1200° C., and at higher temperatures a contraction takes place. The softening point is near 1700° C., the fusion point 2100° C., and at 2200° C. exaporation starts.[1] Its specific weight varies from 2.1 to 2.2.

Quartz is an excellent electric insulator, plates of 1.5 mm to 2 mm thick cannot be pierced except at a tension near 35,000 volts. For a thickness of 7 mm more than 70,000 volts are required.

Being insoluble in acids it makes it applicable to many purposes. It is, however, attacked violently by hydrofluoric acid and slightly by hot phosphoric acid. It does not resist either alkalies. Therefore, laboratory apparatus of quartz glass are sometimes coated with a resisting varnish or a coating of precious metals. The "spray" process for galvanizing, invented by Mr. Schoop (this journal, Vol. VIII, page 404), can be used for such a purpose with economy.

Quartz glass is very translucid for rays of short wave-length. Its great permeability for ultra-violet rays causes it to be used in phototherapy and in the production of ozone for sterilizing water and milk.

Applications of quartz glass are now very numerous and everything points to further extensions. Its great resistance to sudden changes of temperature renders it very precious in chemical laboratories for obtaining capsules and crucibles for making thermometers in which the displacement of zero is nil, etc. For lamp glasses, for globes, for lighting by incandescence, for supports for mantels, for tubes for levels and steam boilers. Mercury-vapor lamps are also made with bulbs of quartz glass.

[1] Others give the melting point as between 1700 and 1800° C and state that softening occurs at 1500° C.—Ed.

Non-transparent quartz can be shaped into vases of large dimensions and takes the place of stoneware with advantage in making pipes used to carry acids, lining for refrigerators and heaters. It also can take the place of porcelain or platinum for making concentration apparatus more durable and cheaper.

Quartz glass is now made by three principal firms, in England by the Thermal Syndicate, Limited, and two in Germany, the Heraeus Campany and the Deutsche Quartz Gesellschaft.

At the factory of the latter objects of large dimensions are obtained by means of a resistance furnace, made of a hollow carbon cylinder of about 200 mm diameter with a central carbon rod. The space left vacant between carbons is filled with white sand and an electric current is passed through the carbons, which are raised to the temperature of 2500°. Each furnace can make an ingot of about 50 kil. The ingot is taken out with pincers; in the hollow part is introduced a gasifying body to do the molding, as described before. The fusion-room contains two large transformers from which 18 furnaces can draw power. Around a third room are distributed the finishing rooms, such as for the treatment with sand blast, grinding with diamonds or carborundum, blowing with the blow pipe, repairing, etc.

Of late years the industry of quartz glass has increased considerably owing to new processes using the electric furnaces. The cost price of manufacture has decreased materially. Although it is yet too high to allow the sale of the objects for the same price as those of common glass of the best quality the qualities of quartz glass give it a preference in a great number of applications where its use is now a necessity.

Large Bronze Casting

The accompanying illustration shows an interesting and rather exceptionally large bronze drum casting. The drum is 12 feet long and 5 feet in diameter, and is used as the drying surface in a large vacuum rotary drum dryer. Such drums are generally made of dense air furnace iron, but in this case it was necessary to use a high quality bronze drum due to the fact that the vegetable extract to be dried on the drum would become discolored if it came in contact with the iron.

BRONZE CASTING

The mold for this casting was swept up in dray sand, the core being swept in loam. When the casting was poured the mold was in a vertical position, thus accounting for the very clean surface free from any blow holes. It required 16,000 pounds of metal to pour the casting and on account of this large quaitity it was necessary to melt the metal in a 48-inch cupola. To melt bronze in a cupola is unusual, but the Buffalo Foundry & Machine Company, who made the casting in question, has been very successful in following this practice where large quanties of metal are required.

An Experimental Arc Furnace for Fine Ore Reduction.

An exhibition of the Reid electric smelting process was given on March 9 at the workshops of the Cobalt Reduction and Refining Company in Newark, N. J. Mr. James H. Reid who some years ago was interested in the promotion of a carbon cell, has developed during the last years an arc-furnace process for the treatment of fine ores, as follows:

The ore fines are fed in a continuous stream by gravity through an electric arc. Within the arc they are reduced (as there is no reducing agent employed, as was exactly stated, the ore must have such a composition or must itself contain such elements as to make reduction possible). The particles of hot reduced metal together with the gangue drop into a box with running water. On the sudden contact with the water the particles of gangue explode and are carried away in the stream of water, while the reduced metal or alloy collects on the bottom of the box. The volatile matter in the ore, when subjected to the heat of the arc is evaporated and the vapors are sucked off from the furnace by a fan and recovered in condensers.

This process was exhibited with an experimental furnace, the capacity of which was given as 1 ton of ore for 24 hours. Exact data of cost, if working on a large scale, are not yet available. The purely electrical and mechanical part of the experimental furnace appeared to work quite nicely. The electrodes between which the arc was produced were of steel. But on a large scale when treating nickel-cobalt ore, it is proposed to use nickel-cobalt electrodes, part of the reduced alloy being used for casting the electrodes. Mr. Reid estimates that the electrode consumption per ton of alloy produced will be 660 pounds of nickel-cobalt electrodes.

The reduced alloy, which collects on the bottom of the water box, is to be treated in a separating kettle, which was shown, but not exhibited in operation. It is provided with electrodes, simply for melting the alloy. Further it is arranged so as to be rotated. The object of this is to separate the constituents of the alloy (perhaps after making a suitable addition) by specific gravity into tops and bottoms.

The company intends to build a larger plant embodying these ideas, near Cobalt, Canada.

Electric Furnace Exhibit.

The high development of the laboratory and smaller industrial electric furnace is well shown at the permament exhibition room of the Hoskins Manufacturing Company in Chicago.

The large muffle furnace for the heat-treatment of high-speed steels, shown in the very center of the photograph, is always of especial interest to visitors, due to the remarkable evenness of the "electric heat" it displays. A temperature of 2300° Fahr. may be produced and this may be regulated to a very fine point

ELECTRIC FURNACE EXHIBIT

by a simple turning of hand-screws in the power circuit. Close to this a crucible furnace, operating on the same carbon-resistance principle, is shown, capable of melting platinum.

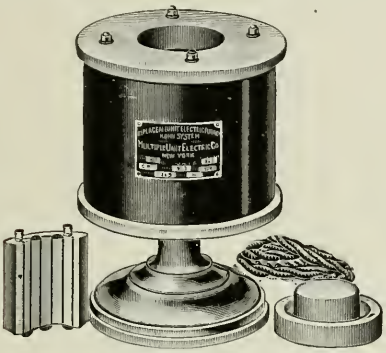
For combustion determinations of carbon in steels the new electric tube furnace, heated by large cross-section metallic elements, is exhibited. This is seen between the muffle and crucible furnaces on the table at the right.

The exact temperature of each furnace operated is clearly shown by means of a full pyrometer installation.

A Novel Laboratory Electric Furnace with Replaceable Heating Units.

We had recently an opportunity to witness a demonstration of an interesting new type of electric laboratory furnace at the workshops of the manufacturers, the Multiple Unit Electric Company, with offices at 47 Maiden Lane, New York City.

This type of furnace is built under patents of Mr. M. M. Kohn. Its chief feature is the replaceability of the electric heating units so that anyone which might burn out can almost instantly be replaced by a new one, held in reserve. The repairs

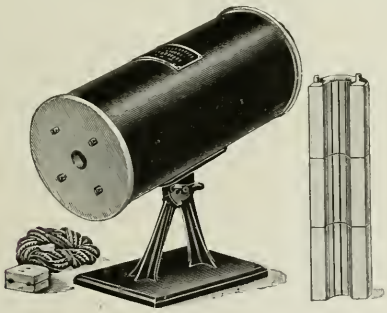


TYPE CR CRUCIBLE FURNACE

of the furnace are thus made exceedingly simple and rapid.

The housing of the furnace is built of a number of sections of a special clay, formed to meet the requirements of each particular case. They are held together by a special cement which will not crack as a result of heating and cooling. This housing which also carries the necessary electrical connections, after being assembled is placed in a metal case, together with the necessary heat resisting material, and is then sealed in permanently, as it is never necessary to remove or change it in any way, unless the furnace housing has been mechanically damaged.

In this housing certain peculiarly shaped openings are molded, into which the replaceable heating units fit, the openings being shaped according to the nature of the furnace, viz.: circular



TYPE CO COMBUSTION FURNACE

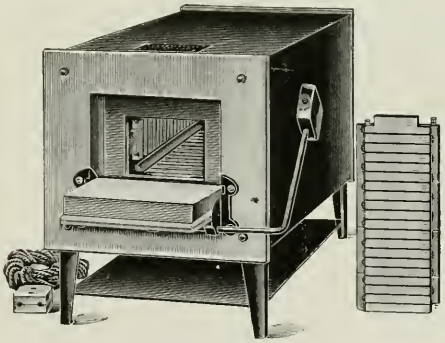
for the crucible and combustion types; and rectangular for the muffle type. The removable units which are replaced in case of a burn-out, carry the necessary heating medium, which may be wire or other material suitable for the purpose. The number of units required, depends entirely on the type of furnace, some requiring two, others four, or more.

In case of a burn-out, all that is necessary is to remove the unit which is burnt out and slide in another one, which can be done by anyone, without any technical knowledge. Units can be replaced, when the furnace is red hot, in less than ten minutes.

These units are made of special clays and other refractory materials in various forms and sizes to meet the requirements of each case. They are made in a number of sections or pieces, each of which is complete in itself and are assembled on rods of special material, which act not only to tie or hold them together, but also as the electrical conductors to carry the current from the housing to the heating medium in which the units are wound. Looking at the front of a unit a number of undercut open slotted grooves are seen, into which the heating medium (wire, etc.), is placed, which while holding it firmly in place still allows for expansion of the medium, while another very important feature of this special groove is the open slot, which allows for the full heat of the medium being utilized in the furnace chamber.

On account of the method of building up the furnace housing of sections it is easy to build the furnace in any size or shape as desired. For instance, a 4-inch furnace may be easily changed into a 5-inch furnace by inserting proper distance pieces.

Further by the method of connecting the heating units it is possible to get a uniform heat within the whole furnace, or to get higher or lower temperatures at different places, according to the requirements of the case. This latter condition must be fulfilled, for instance, in electric furnaces for the automatic heat treatment of blades for safety razors. While the blades pass through the furnace they are subjected to different tem-



TYPE MU MUFFLE FURNACE

peratures according to the requirements of the heat treatment. Quite a number of Kohn furnaces are in successful commercial use just for this purpose.

Another feature of the Kohn furnace is that it is built for direct operation on 110- or 220-volt mains without necessitating the use of a transformer. Any muffle or combustion furnace can be used on either 110 or 220 volts by simply shifting a plug from one set of contacts to another, according to the voltage that is to be used.

Any of the Kohn furnaces may also be equipped with an additional two-heat switch, by means of which it is possible to obtain ¼ heat or full heat, and also with special three-heat switches by means of which 1/3, 2/3 or full heat is obtained, both without the use of a rheostat or external resistance, and with absolute uniformity of temperature.

The safe maximum working temperature of the Kohn furnace is from 1750° to 1850° F. But in emergency cases they will stand temperatures up to 1900° F. for a short time.

The adjoining illustrations show the standard types of the Kohn furnace, the type CR crucible furnace, the type CO combustion furnace and the type MU muffle furnace. In each case a heating unit is illustrated on the side of the furnace, to show its construction.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore-Treatment (*Continued*).

286,208, Oct. 9, 1883, Leon Letrange, of Paris, France.

Zinc. Calcines blende to zinc sulphate and passes the fumes over calamine undergoing calcination, partially converting it into a sulphate. Unsulphated zinc oxide is dissolved by dilute sulphuric acid in vessels of masonry lined with bitumen. The several vessels are successively charged with mineral and acid, drained and refilled. The zinc is precipitated from the acid solution in electrolytic cells of wood lined with glass or lead, the anodes being plates of carbon and the cathodes thin plates of zinc, copper or polished brass from which the deposit is stripped. The liberated sulphuric acid overflows through outlets at the top of the cells, the level of the bath being maintained by the introduction of fresh solution.

291,670, Jan. 8, 1884, Michel Body, of Liege, Belgium.

Gold and silver. The ore is ground fine and either treated in heaps with a ferric-salt solution or mixed with ferric salts and roasted. It is then placed in an electrolytic amalgamator, consisting of a revoluble horizontal cast-iron drum, serving as cathode and containing a number of cast-iron balls. The anodes are plates of compressed coke or graphite, suspended from a stationary, hollow, perforated, insulated shaft passing through stuffing boxes in the ends of the drum. The electshaft.

A steam pipe also does enter the electrolyte. The silver deposits on the drum and balls and is absorbed by mercury subsequently introduced into the drum. The ferric salts are simultaneously converted into the ferrous form.

300,950, June 24, 1884, Henry Benner Cassel, of New York, N. Y.

Dissolves metals, especially gold, by electrolytic chlorine. The cell is a tank containing a semi-annular sheet cathode. The tank receives the lower half of a rotary horizontal drum, containing a series of carbon rod or plate electrodes, arranged peripherally. The ends of the anodes extend through the heads of the drum and are connected by copper wires to a copper rod on one axle, making contact with a fixed copper spiral. The heads, or the cylindrical or polygonal body of the drum, may be of porous clay. Or the body of the drum may be omitted and replaced by a diaphragm secured in the stationary tank.

The ore which is to be treated is placed in the drum or anode compartment with a sodium-chloride solution, and is agitated by the anodes as the drum rotates, the metal being dissolved by the liberated chlorine. It is stated that the dissolved metal is not deposited from the cathode.

300,951, June 24, 1884, Henry Benner Cassel, of New York, N. Y.

Gold. Pyritic and other refractory auriferous ores are electrically dissolved in a sodium-chloride solution. Lime, or equivalent, is added to neutralize the hydrochloric and hypochlorous acids formed and prevent their converting iron in the ore into a proto-salt which would precipitate the gold from its chloride solution.

Or a solution may be employed which will yield a base capable of neutralizing the acids.

317,245, May 5, 1885, Edward P. Thompson, of Elizabeth, N. J., assignor of three-fourths to E. P. Roberts and G. H. Pierce, both of Cheyenne, Wyom.

Gold. Dissolves gold by placing the ore in a tank of lead lined with paraffine, pitch or asphaltum, filled with water and introducing a stream of chlorine at the bottom. The resulting gold-chloride solution is electrolyzed in a cell having anodes and cathode of carbon. Some gold ore may also be placed in this cell to absorb the chlorine freed therein. A number of cathodes coated with gold are packed with fine charcoal in a crucible and heated until the gold melts and runs into a button at the bottom. The chlorine for treating the ore may be produced in an electrolytic cell having an anode of carbon and a cathode consisting of a metal pipe through which steam is introduced into the solution of sodium chloride, acidulated with a few drops of hydrochloric acid. Or a lead-chloride solution may be electrolyzed between a carbon anode and a lead-plate cathode.

BOOK REVIEWS.

Electrische Öfen in der Eisenindustrie. By Dipl-Ing. **W. Rodenhauser** und Betriebsdirektor **I. Schoenawa.** Octavo, 326 pages, 127 illustrations. Price in paper, marks 13.50; cloth, marks 15. (Retail price in New York: $4.50.) Leipzig, 1911; Oscar Leiner.

Electric furnace literature contains many erroneous statements and claims, which contradict the fundamental principles of electrotechnology. The authors have placed before themselves the task of giving complete and accurate information on the electric furnaces used in the iron and steel industry and their operations.

It is not probable that there are in Europe many more metallurgical engineers better qualified to accomplish this task than the designer of the Roechling-Rodenhauser induction furnace and his chief works manager, the above named authors. That two such busy technologists would take the time to write such a treatise for the benefit of the industry and the enlightenment of their colleagues and competitors proves that they are the opposite to narrow technicians—proves that they are broad-minded benefactors of science and industry.

Congratulations are due them in the successful manner in which they have executed their task. To those interested in electric furnaces it may seem that the authors have exaggerated the relative advantages of the induction furnace; but a sober second thought on this question will suggest the possibility that induction furnaces are perhaps better suited for certain metallurgical operations than electrode furnaces, and that their star may be again in the ascendant.

* * *

Herstellung Kolloider Lösungen anorganischer Stoffe. Ein Hand und Hilfsbuch für die Chemie und Industrie der Kolloide. von Dr. **The Svedberg**, Privatdozent an der Universität Upsala. Octavo, 507 pages, 60 illus. Dresden: Theodor Steinkopff.

Kapillarchemie. Eine Darstellung der Chemie der Kolloide und verwandter Gebiete. von Dr. **Herbert Freundlich**, Privatdozent un der Universität Leipzig. Octova, 591 pages, 75 illus. Price, 17.50 marks. Leipzig: Akademische Verlagsgesellschaft m.B.H.

These books represent one of the most rapidly advancing branches of theoretical and applied chemistry; a field practically on the border line between physics and chemistry. The extent to which colloidal chemistry has been proved to enter into the chemistry of every-day life and the vast field opened up for experiment, observation and development of satisfactory theory should, and undoubtedly will, attract many earnest chemical students. Dr. Svedberg's book deals principally with the processes of producing colloids by the methods of condensation and of dispersion, while Dr. Freundlich's deals principally with surface tension, separating surfaces and the detailed properties of suspension colloids, emulsion colloids and gels. There is a wealth of interesting and useful information in these two works.

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New York Meeting of the American Electrochemical Society.

New York, as a convention city, is a somewhat dangerous place, as experience has shown that while it is easy to get a good attendance on paper in form of a long registry list, yet it is difficult to hold the members together for any length of time. No such difficulty was experienced at the recent convention of the American Electrochemical Society and this fact is conclusive evidence that those who attended it came with a purpose and that the society at its meetings gives to its members what they want and need. It was interesting to analyze the personnel of the attendance. There were those typical electrochemists—many of them from Niagara—who have attended practically every meeting of the Society from the beginning. But the nucleus has grown. The new recruits have come both from the chemical and metallurgical fields. Pittsburgh alone, for instance, has sent large and enthusiastic delegations to every one of the last meetings. The circle is larger, but the esprit de corps is the same as in olden times. The growth of the Society reflects accurately the growth of the industry.

Simple Tin Reduction.

In searching the rather meager literature on the metallurgy of tin one is struck by the lack of definite information which would enable a metallurgist ordinarily skilled in his profession to undertake the reduction of tin ore with reasonable prospect of success. In a general way it is understood that both shaft and reverberatory furnaces have been used, but regarding the details of smelting mixtures and compositions of slags contributors generally are silent. Two queries arise in this connection. Is it to be inferred that the reduction of tin is a complex metallurgical process evolved by the highest technical skill and hence held as a trade secret? Or are the conditions of success so variable according to local conditions that the principles of the process are not reducible to terms for general guidance? Perhaps the truth lies partly in the answer to both questions.

Undoubtedly the impurity of some tin ores, contaminated as they are with bismuth, tungsten, arsenic, iron, copper and zinc, has been responsible for the seeming complexity of their treatment. Roasting, leaching and concentrating are preliminary processes in many instances; and the smelting and resmelting of slags, liquating and poling of the reduced tin, and retreatment of impure residues, comprise accessory processes which give an aspect of complexity to the work. Finally, the production of grades of tin of varying purity testifies to the difficulty of securing a uniformly pure product.

It is rather refreshing, therefore, to find that the recent reduction of tin ore in Texas is characterized by simplicity which is in striking contrast to the complex procedure mentioned. True, natural conditions are favorable in that the

cassiterite is quite pure; nevertheless, those in charge of the work were obliged to put their own ideas into practice, despite the predictions of failure from foreign sources. Their success was striking. By mixing the fine tin concentrate with 8 per cent to 12 per cent fine anthracite coal, heating and rabbling the mixture on the hearth of an oil-burning reverberatory furnace, the tin was reduced in a pure condition, 99.8 per cent and 99.9 per cent fine, ready for market. No slag-forming ingredients were added, and no slag made; nor was it necessary subsequent to refine the tin by liquation or by poling with green wood. Later we shall have more detailed information to present regarding this interesting work.

The Pachuca Agitator.

A note of striking unanimity was sounded by all cyanide experts in their annual reviews for 1910 in regard to the use of the Brown or Pachuca tank for agitation. Almost with one accord they hailed it as the best machine for agitating ore pulp in cyanide solution, and recorded its triumphant path through the goldfields of the world. Apparently the only objection which had been raised to the Pachuca tank, viz., its height and the consequent head-room required, was regarded as having been removed by the system of continuous treatment adopted by Grothe and Kuryla. And yet if the literature be searched for information regarding the agitator little will be found except the usual description embodying dimensions, time required and results obtained. All of which suggests that perhaps the Pachuca tank has been accepted at sight, without being subjected to the exhaustive inquiry usually directed toward kindred devices.

It is not surprising, therefore, that recently some investigations have been made regarding the height of the tank, and whether such height is necessary to accomplish to best advantage the primary purpose of the machine, viz., solution of the precious metals. In ordinary practice the central air-lift extends practically the full height of the tank, and pulp and air are discharged at the top, the former settling for further agitation and the latter escaping into the air. The question arises whether, considering the great height of the tank, 45 ft., it offers the fullest opportunity for dissolution of the values throughout the cycle of operations. Aeration is probably effective only in the central column, and at the top of the charge, while dissolution may not be effective in the return downward path of the pulp.

Several experiments apparently have proved that the air-lift should discharge considerably below the surface of the charge in order that better aeration and dissolution may occur. Philip Argall has modified standard practice at Cripple Creek by shortening the central pipe so that the pulp is discharged 8 or 10 in. below the surface of the charge, thus spreading the air through the upper portion of the charge, and improving aeration and dissolution. A. J. Yaeger has gone further. At the Zambona Development Company, Mexico, the tank has been modified until now the central air-lift is but half the height of the tank, and the discharging air rises in small bubbles through the upper half of the charge, materially improving agitation and increasing extraction as compared with former practice.

As stated above, the main object of the tank is to secure solution of one or two of the elements in the mixture. The physical conditions governing rapid solution require that fresh surfaces of the substance to be dissolved be continuously exposed to the solvent, either by changing the position of the substance in the solvent, or allowing the latter to flow over it, or both. A. W. Warwick has suggested that these conditions are not fulfilled in the great mass of pulp falling in the tank outside the central air-lift, and that the machine is consequently inefficient, considering its great height and volume. Granting that agitation and aeration occur in the central air-lift and at the surface of the charge, may it not be probable that on settling the particles of ore and the solution fall at too nearly the same rate of speed to provide the best conditions for solution.

The Pachuca tank is unquestionably a success, but perhaps the chemist has accepted the dictum of the mechanical engineer without due inquiry, and we may look for material improvement as the former makes more thorough investigation.

Steel Production in 1910.

Official statistics show the production of steel ingots and castings in the United States in 1910 to have been as follows, in gross tons:

Bessemer (acid)	9,412,772
Basic open-hearth	15,292,329
Acid open-hearth	1,212,180
Total open-hearth	16,504,509
Crucible	122,303
Electric	52,141
Miscellaneous	3,194
Total, all steel	26,094,919

There is a gain in the total over 1909 of 2,139,898 tons and of 2,700,000 tons over 1906 and 1907, the two best previous years, 1909 having been an "off" year with an output of only 14,023,247 tons.

The total in 1910 comprised 940,832 tons of castings and 25,154,087 tons of ingots. Statistics of rolled steel in 1910 will hardly be completed until late in the year. In 1906, 1907 and 1909 the rolled steel was about 22.5 per cent less than the tonnage of ingots, this representing in small part the unavoidable losses of material, but in large part scrap produced in rolling and which was promptly returned to the Bessemer vessel or open-hearth furnace, chiefly the latter, to be remelted, the same steel thus passing again over the ingot scales. What is customarily given as the "production of steel" is really the record of the ingot scales. The production of steel is more properly the production of rolled steel, weighed after its last hot rolling and after it has been cropped. Applying the 22.5 per cent just derived the production of rolled steel in 1910 promises to be about 19,600,000 tons. We may anticipate a little by observing that in 1909 the production of rolled iron amounted to 1,709,431 tons, so that the total production of rolled iron and steel in 1910 promises to be in the neighborhood of 21,00,000, or 21,500,000 gross tons. In addition we have the 940,832 tons of steel castings, while the iron foundries probably produced not far from 10,000,000 tons of iron castings. The total doubtless exceeds 30,000,000 tons, comparing with 27,298,545 tons of pig iron produced in the year. There is, of course, a large utilization of scrap, by steel works, iron

mills and foundries. A small part of the rolled steel is old material rerolled, the chief products being light rails, bedstead angles and concrete reinforcing bars.

The basic open-hearth steel industry has taken chief rank in our steel processes so suddenly that there is little to say from a statistical standpoint. In 1907 the total open-hearth steel production fell just short of the Bessemer, with 1,270,421 tons of the open-hearth steel made by the acid process; in 1908 the basic open-hearth steel alone exceeded the Bessemer output, but both processes fell far short of equalling their previous records; in 1909 basic open-hearth steel showed a much larger tonnage than Bessemer and acid open-hearth combined, these being the two processes which use pig iron and scrap of limited phosphorus content. In 1910 the production by the basic open-hearth process exceeded that of the Bessemer process by 62 per cent. There is no prospect of the Bessemer process ever equalling its record output, made with 12,275,830 tons in 1906. Since that year no new Bessemer plants have been built, while one efficient plant has been dismantled and another has been turned largely to the practice of the duplex process. How far the electric refining furnace as an accessory to the converter will be able to change the situation it is too early yet to surmise. That the electric furnace will become of decisive importance, however, in this respect, there can be no doubt.

Efficiency and Conservation.

Much has been written of late in both technical and popular magazines on the subject of efficiency and scientific management. It is a rather remarkable fact that although the principles of efficiency engineering are scientific in the highest degree, yet their application has been largely in the field of industry rather than of science. The tradesman, mechanic and factory hand have been its beneficiaries, while the strictly scientific worker has either been purposely neglected or considered able to direct his own efforts to attain the highest degree of efficiency. There may be a subtle compliment in this neglect of the scientist, and if so, fact and conditions ought to show him worthy of it, but we suspect that investigation will find him a transgressor in many simple and almost inexcusable ways.

Efficiency implies system; system implies knowledge of details, and this, in turn, implies brains. We thus come to brains, or a central competent head, as the foremost requirement of efficient work in a laboratory, smelter or plant. Next in point of importance comes the individual ability of the man for his task, and the collective efficiency of the entire corps of men engaged in work to a common end. Finally, the correct design of the plant, the construction of the various buildings, and means for accomplishing the necessary work, form a vital element in the whole system of efficient operation. Given these conditions we have operative conservation in its truest form, as far as it relates to time, money, material and energy.

The directing head of a chemical or metallurgical plant should have more knowledge of the work in hand than any of those under him. This may seem a self-evident proposition, but the instances have been known, especially in the early history of metallurgy in the West, where the smelter boss and the political boss were one and the same, and where technical positions were granted for some real or fancied service in another sphere of activity. Under such conditions metallurgical efficiency and conservation were at a low ebb, and financial success ensued only because the property could stand mismanagement. Those in charge were incompetent to initiate improvements, or pass competently on the suggestions of their employees, many of whom were far superior in technical skill and ability, but without authority to enforce their views.

On the second point—individual fitness and the collective ability of the entire corps—an inquiry often will reveal weak points which do not appear on the surface. We can recall cases where excellent laboratory chemists apparently failed when placed outside, and again where men seemingly poorly adapted to the work of the laboratory made successes in the plant. It is the old problem of the square peg and the round hole, and presents a condition where both individual efficiency and the general efficiency of the plant would be higher if the men were kept where they could work best. The routine work of the laboratory, made up as it is of a few separate operations, each repeated many times during the day, requires a certain temperament for its best performance, and the man who can do it well may not be able to direct those larger operations in the plant which consist in the handling of labor and material, and the economical output of the product.

On the third score, viz., correct design and construction of plant, we find many transgressions of the laws of efficiency and conservation. Perhaps nothing has been so potent in the development of chemical and metallurgical engineering as a profession as the mere fact that those on whom the chemist or metallurgist formerly relied for design and construction of plants, did not fully understand the technical requirements of the plant as a whole, and hence failed to produce the best result. The chemical and metallurgical engineer was a logical development in the demand for efficiency in plant construction. The design and construction of a plant, of the individual buildings, and their grouping together, all exert a wonderful influence on general efficiency. Poor arrangement will nullify the benefits gained through a trained corps of workmen, and illogical design will defeat the ends of the best appliances. A workman, consciously or unconsciously, will neglect a machine placed in an inconvenient place. He will not inspect those parts of the plant which are difficult of access, or which are uncomfortable to visit, with the same regularity or willingness that he will perform the more agreeable part of his duties. Hence the necessity for construction and arrangement which, per se, will not constitute a cause for neglect of duty.

Poor arrangement of laboratory or plant will cause many unnecessary steps to be taken and useless time consumed in transporting material from one part to another. The cost of these wastes of time and energy will amount to a considerable sum per annum, and frequently they will not be appreciated until an investigation has shown where loss can be changed to profit. Fortunately, as said before, the chemical and metallurgical engineer is taking care of these matters as never before in the history of the industry. Much is required of him. He must have mental ability and physical capacity; executive as well as engineering training, and business sense of a high order. With these qualities he will not be guilty of making avoidable wastes and recoverable losses, and will give high efficiency and genuine conservation in all his work.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

Electric Furnaces and Heating.

In the course of the second Cantor Lecture, recently delivered before the Society of Arts, Dr. J. A. Fleming said that the methods of electric heating were classified into resistance heating, arc heating and combinations of the two. He mentionad Sir William Siemens' early experiments on the electrical fusion of steel in 1879, and described the electric-arc furnaces of E. Stassano and Paul Héroult in 1898 and 1899. Héroult in 1900 manufactured steel from pig iron and scrap by means of his arc-resistance furnace; and furnaces were at La Praz now producing six or more tons of steel a day, at an expenditure of 800 to 900 kw-hours per ton. Pig iron could be prepared electrically from hematite by an expenditure of about 2,000 to 2,200 kw-hours per metric ton. The modern blast furnace used only about 16 cwt. of coke per ton of pig, of which 6.5 were required to reduce the ore, and the cost of electric energy must be reduced to 0.1 cent per kw-hour before the electric smelting could compete with the blast furnace. Electric furnaces, however, worked efficiently in the production of high-grade steel from scrap iron and pig in the place of, or as supplementing, the open-hearth process. The Girod arc furnace as developed by M. Paul Girod and used at his works at Ugines produced a high-class steel from scrap iron by an expenditure of 1,000 to 1,100 kw-hours per metric ton. The danger of carbon contamination by the arc furnace electrodes led to the invention of the Ferranti induction furnace, in which a large alternating-current transformer had a secondary circuit consisting of an endless channel lined with magnesite or dolomite, in which the metal was melted by induced currents. The Kjellin furnaces at Gysinge produced excellent steel from pig iron and scrap, consuming about 900 kw-hours per ton of steel; electric energy costing about 0.09d per kw-hour, or £2 per electrical hp-year. In combination furnaces, in addition to the heating of the charge by the induced currents, current induced in a copper secondary circuit was led through the charge from metal electrodes embedded in the furnace lining. The power factor was raised and the combination furnace could be worked at a higher frequency and by three-phase currents. Moreover, this furnace had a roomy and accessible hearth, and was excellent for refining. By using a basic lining sulphur and phosphorus could be eliminated and the occluded gases entirely removed, while the metal was not oxidized as in open-hearth furnaces. Another advantage was the economy of labor compared with the ordinary crucible processes. The electric furnace produced a high class of steel equal to the best crucible steel, and was especially adapted to the production of alloy steels, viz., vanadium, chrome and tungsten steels.

A 10,000-Volt Battery.

Mr. A. A. Campbell Swinton, at a meeting of the Röntgen Society, described a battery consisting of 4800 cells, a number which divided up into convenient sections of 80 cells for charging in parallel off a 200-volt supply. Arranged in series, it gave 10,400 volts on open circuit. Each cell consisted of a glass test tube 4 in. long and ¾ in. bore, and each set of 40 cells was cast solid with paraffin wax into a wooden tray. The battery was of the Planté type, and the plates were made of sheet lead cut into strips which were bent round a shaped piece of wood and pressed to a corrugated form not very liable to deformation. The plates were kept from touching each other by celluloid separators, and were all formed in a nursery battery before being transferred to the battery proper. The total weight of lead in the whole battery was 400 lb., and the total weight of the whole of the battery, exclusive of containing cases, was nearly 1000 lb.

To avoid bubbling the battery was charged at a rate not exceeding 10 milliamperes, and it was capable of discharging at a rate of 150 milliamperes, giving 1½ kw for several minutes. The internal resistance varied between 3000 and 4000 ohms.

With 10,000 volts between polished brass balls 12.5 mm in diameter the current was found to jump 2.5 mm, and the arc could be drawn out to a considerable length, dependent on the resistance in the circuit. That the current was often not entirely continuous was indicated by the arc making a whistling sound. A Leyden jar connected across the arc immediately converted it into a series of discontinuous sparks which, if the jar was of small capacity, followed one another very rapidly, producing a musical note which deepened into a roar when the capacity was increased. The battery showed very plainly the electro-capillary effect discovered by the late Lord Armstrong. At 8000 to 10,000 volts and about 5 milliamperes, and using pure water, the results were exactly similar to those obtained by him with hydroelectric and induction-coil discharges. The addition of a very small quantity of dilute sulphuric acid to the water, which destroyed all movement of the thread when induction-coil discharges were employed, was found, with the battery current, not entirely to stop the thread moving, but increased the flow of water in the opposite direction. Practically all the Crookes and other vacuum-tube effects could be obtained with the battery current.

Rusting of Steel in Concrete.

The draft report of the science standing committee on the "Rusting of Steel Inside a Concrete Covering," which was presented at a recent meeting of the Concrete Institute, stated that a circular was issued in 1909 by the Concrete Institute asking for observations on the question whether rusting of steel takes place when covered by concrete, and was sent to 1000 engineers and others engaged in concrete construction.

Only 111 replies were received. Forty-seven contained results of definite observations, in which 26 cases of rusting had come under notice, and in 43 cases no rusting had been found.

As a result of these investigations the committee has arrived at the following conclusions: Reinforced concrete is as durable as plain concrete in any situation provided that certain precautions are taken in construction.

The cement, sand, stone, etc., must be of good quality, and must be carefully and thoroughly mixed and scientifically proportioned, so as to be practically waterproof and airproof. The mixture must be fairly wet and must be well punned into position to minimize voids. The aggregate should be as nonporous as possible, and no aggregate known to have a chemical action on steel should be used. The aggregate should all pass through a ¾-in. mesh. The concrete covering should never be less than ½ in., and with round or square bars the covering should not be less than the diameter of the bar. In structures exposed to the action of water or damp air the thickness of covering should be increased at least 50 per cent, or the size of the aggregate should be reduced so as to ensure a dense skin. Structures exposed to very severe conditions should have the concrete covered with some impervious coating.

The reinforcement should allow sufficient space between one piece and its neighbor for the concrete to pass and to completely surround every part of the steel. All steel should be firmly supported during the ramming of the concrete to avoid displacement. It should not be oiled or painted, and thick rust should be scraped and brushed off before placing in position.

The scantling of the various members of the structure should be sufficient to prevent excessive deflection. If electric mains are laid down care must be taken that no current is allowed to pass through the reinforced concrete. Fresh water should be used in mixing, and aggregates charged with salt should be well washed.

Engineering Imports and Exports.

The Board of Trade returns for January and February show that the imports of iron and steel, including manufactures, amounted to £1,689,512, or £328,048 more than during the

corresponding period of last year; and the exports were £7,196,410, showing an increase of £609,054. The imports of other metals and manufactures thereof were £4,318,634, and the exports £1,708,186, a decrease of £83,083 in imports and an increase of £96,000 in exports. The imports of electrical goods last month were £231,549, an increase of £9,567, and exports reached £428,724, giving an increase of £75,880. The imports of machinery are valued at £884,912, an increase of £235,957, and the exports at £4,714,446, an increase of £693,878. Imports of new ships were £37,092, an increase of £36,690; exports amounted to £620,847, a decrease of £1,665,134.

Market Prices.

March, 1911.

Copper has not varied much this month. Opening at £54.13.9 it reached £55.1.3 by the 15th, afterward easing off to £54.17.6, and is now (29th) at £54.15.0.

Tin has been over a 10-pound range, but rather less jerkily than before. Starting at £189.10.0 it had dropped by the 15th to £178.10.0, afterward recovering to £182 and now quoted £185.17.6.

Lead has been depressed, dropping from £13.12.6 to £13.10.0 by the 15th and ultimately to £13.5.0.

Hæmatite has also fallen. Opening at 66/2 it was 65/- at midmonth and now shows 64/3.

Pig Iron (*Scotch*) has dropped similarly from 54/9 to 54/4½ and then to 53/7½.

Cleveland Warrants opened at 48/8½ and dropped steadily to 48/3 (15th) thence to 47/7½.

India rubber has been cheaper, falling from 7/1, the opening price, to 6/10 by the 8th and 6/6½ by the 15th; was 6/7½ on the 22d and 5/11½ on the 29th, closing at 5/9½.

Alum, lump, loose, per ton	£5.12. 6.
Antimony, black sulphide powder	22. 0. 0.
Borax, British refined crystal	16. 0. 0.
Sulphate of ammonia, f. o. b. Liverpool, per ton	14. 7. 6.
Sal-ammoniac, lump, firsts, delivered U. K., per ton	42. 0. 0.
Copper sulphate, per ton	19.17. 6.
Camphor, 1-oz. tablets	0. 1.10.
Caustic soda, ash, 48 per cent ordinary, per ton	5.10. 0.
Shellac, standard T. N. Orange spots, per cwt	3.13. 0.
Sulphur, recovered, per ton	5. 0. 0.
Carbolic acid, liquid, 97/99 per cent, per gal	0. 1. 5.
Hydrochloric acid, per cwt	0. 5. 0.
Creosote, ordinary good liquid, per gal	0. 0. 2.
Naphtha, solvent, 90 per cent at 160° C., per gal	0. 0. 9.
Petroleum, Russian, spot, per gal	0. 0. 5.
Copper ore, 10 to 25 per cent unit	9/1½ to 9. 7.
Tin ore, 70 per cent, per ton	£112 to 114.0. 0.
Mica, small slab, in original cases, per lb	6d. to 1. 0.
Litharge, flake, per ton	15.10. 0.
Quicksilver, per bottle, Spanish, 75 lb	£9.7.6 to 12. 0. 0.
Zinc sheets, V. M., f. o. b. Antwerp	27.10. 0.
Red lead, genuine, per ton	16. 0. 0.
Bleaching powder, 35 per cent, per ton	4. 5. 0.
Platinum, per ounce, nominal	8. 6. 6.

On the month the following differences are noted:

Lower.	Amount.
Hæmatite	£0. 2. 0.
Pig Iron (Scotch)	0. 1. 7.
Cleveland Warrants	0. 1. 3½.
Tin ore	11. 0. 0.
India rubber, para	0. 1. 3½.
Tin	3.12. 6.
Lead	0. 7. 6.
Higher.	
Sulphate of ammonia	0. 5. 0.
Shellac, T. N. Orange spot	0. 1. 0.
Carbolic acid	0. 0. 1.
Copper, per ton	0. 1. 3.

The Congress of Technology in Boston.

The Congress of Technology which met in Boston on April 10 and 11 in celebration of the semi-centennial of the signing of the charter of the Massachusetts Institute of Technology was a pronounced success.

The opening session of the Congress was begun with an address by President Maclaurin of the Institute, who spoke on "Some Factors in the Institute's Success," and attributed that success primarily to the method of instruction, now everywhere accepted, but not long ago a debated venture.

"Apart from his appreciation of the value of all sound learning," said Dr. Maclaurin, "Wm. Barton Rogers, the founder and first president of the Institute, saw clearly that the whole controversy as to the relative merits of science and the classics in the field of education missed the mark by placing the emphasis in the wrong place. He understood that when one gets to the root of things in education, the *method* rather than the *subject* is of supreme importance, and his insistence on the value of method in teaching was the cardinal doctrine in his creed and the one that has contributed most to the success of the Institute. His fundamental idea here was not original with Rogers. It has been clearly expressed before, but rarely, if ever, adopted definitely as the basis of educational method and applied systematically throughout. The idea is familiar to us all to-day, the idea of *learning by doing.*"

The daylight hours of the second day were given over to the presentation of papers on various aspects of applied science. These papers were grouped in six divisions, and the times of reading were so arranged that the general public, a large number of whom had received personal invitations, were enabled to hear papers in several divisions on topics in which they were particularly interested.

The result of these sessions amply justified the plan. Every session was very largely attended, and as the papers dealt not with educational or technical abstractions but were on the contrary striking and vital reports from the actual field of industrial work, the large audiences gained a better conception than would otherwise have been possible of the fashion in which applied science has remade the life and habits of the world within the last half century. Some of these papers are printed elsewhere in this issue.

The other main object of the Congress was achieved with equal certainty. The papers were all given by Technology Alumni and members of the faculty and the great range of accomplishment which the papers reported were a splendid certificate of merit and efficiency running to the credit of the men trained at the Institute.

Another feature that is perhaps of more importance than either of the two already mentioned is that the Congress has set clearly before the public the needs and the possibilities of what the speakers on Tuesday evening called the "New Technology." The high standard of the Massachusetts Institute of Technology, and its well deserved fame as one of the very foremost technical schools in the world, hardly need reiteration. But the very success of the Institute has brought it to a point where insufficient endowment and cramped quarters threaten seriously to interfere with its future usefulness unless the obstacles are very soon removed.

The banquet in Symphony Hall on Tuesday evening, April 11, provided a significant sign of deep appreciation of this need both by the alumni of the Institute and by many others outside of its immediate ranks. The enthusiasm of the thousand alumni and guests who filled the floor of the hall was reflected in the words of the speakers, who included two such notable figures in Institute affairs as President Lowell of Harvard, long a member of the Corporation of Technology, and President Emeritus Eliot of Harvard, who was professor of chemistry in the original Institute faculty.

The temper of this great meeting was significant in showing a sharp consciousness of a new era created by applied science;

of the important part that has already been taken by graduates of the Massachusetts Institute, and of the pressing need, especially from the point of view of the interest of Massachusetts, that the Institute should be immediately equipped with the additional means for enabling it to continue in the future as it has been in the past, a leader in technical education.

The practical earnestness of the alumni in doing their full share toward making possible the "New Technology" was shown in three announcements made by President Maclaurin from the platform after the banquet. One alumnus, who wished his name withhheld, has offered to the Institute a tract of 1000 acres of land in Maine, well suited to the uses of a summer school in civil engineering. Other alumni have definitely pledged themselves to give a very large part of the price necessary for buying a new site for the Institute. Edward N. Hagar, Tech '93, president of the Universal Portland Cement Company, has promised as a gift all the cement needed for erecting the new buildings in reinforced concrete. Meanwhile, the question of the site itself, Dr. Maclaurin said, has been narrowed to a choice between three sites, all of which are within a short distance of the present buildings. It is expected that this question of a new site, upon which everything else in the development of the Institute depends, will very soon be settled.

The Western Metallurgical Field

Dredging in South Dakota.

A Denver corporation known as the Castle Creek Hydraulic Gold Mining Company is preparing to carry on dredging operations on Castle Creek, near Mystic, S. D., this summer. About 700 acres of ground were acquired by the company, most of which is amenable to dredging, and over 100 acres of which have been proved by bore holes. The average value of this ground was proved to be $1.30 per yard, most of the gold being coarse and easily recovered. It is estimated that the cost of dredging will be very low, possibly as low as 5 cents per yard, as the ground contains no large boulders and there are no unusual obstacles to be overcome.

The dredge, which is now being built, will be in operation this summer. It is of the type which has proved so successful in California, with 5-cu. ft. buckets, and a hull measuring about 100 ft. x 36 ft. at water line. It will be operated by electricity generated at the company's power plant, which comprises four 100-hp boilers, a 300-hp Corliss engine and generator. The power requirement under full load is estimated to be 240 hp.

Winter operation is to be made possible by a suitable steam heating plant, with an inclosure over the stacker. The depth to bedrock is about 35 ft., which gives a large yardage on the area already proved. If the first year's operations prove successful it is possible that another dredge will be built to work on the other end of the tract. Assays on the fineness of the gold have given results of 947.5 fine.

Copper Concentration.

The work of the Utah Copper Company in concentrating large quantities of low-grade copper ore is generally conceded to be in the front rank of concentration. The report comes from Butte that representatives of the Anaconda Copper Company have been investigating the methods of the Utah company, with a view to making some changes in the Montana mills. The devices particularly attracting attention are the Garfield roughing table and the Janney classifier, the latter being an invention of the mill superintendent of the Utah Copper Company.

The roughing table has proved its value in preparing a lean pulp for treatment on standard tables by getting rid of a large quantity of gangue, thus practically making an enriched ore for the regular tables. The Janney classifier has been an improvement in the treatment of fine pulps, and is used in both the Utah mills.

At Anaconda jigging is the main treatment and is carried to a fine point, the smallest jig concentrate being 1 mm in size. Following this are the Wilfley tables, but a large quantity of slime escapes treatment and is impounded. There are no vanners in the Anaconda plant, while in the Utah plants the vanner is relied on for a large percentage of the recovery. It is said that experiments will be made at Anaconda to test the value of the roughing table between the jigs and Wilfley tables, and that Janney classifiers and vanners will be tried on the finer sizes not treated on the Wilfley tables.

The Nevada Consolidated also is reported to contemplate experiments with various slime tables for the purpose of reducing the present slime losses, which are high on this ore. In the smelting department of the same company alterations are being made to test the basic lined converter. Should this method of converting prove successful, and it probably will be as successful there as at Utah, the old battery of acid-lined converters will be displaced by those of the new type.

Macquisten Tubes in Idaho.

Of the various flotation processes which have been proposed or operated, the Macquisten tube is one of the most interesting. The more common floation processes make use of oil or acid, or both, to make the mineral particle buoyant, but the Macquisten process makes use of a different principle, viz., the varying action of mineral and gangue particles to the surface tension of water. Sulphide minerals are apparently unable to overcome the surface tension of water when delivered onto its surface, and, therefore, float, while gangue particles promptly sink. Thus for sulphide mineral particles, gravity is a lesser force than the surface tension of water.

The Macquisten tube is a cast-iron tube about 6 ft. long and 1 ft. in diameter. On its inner surface it has a helical groove and on its outer surface two tires which rest on supports. It is revolved in a horizontal position, and the ground pulp is delivered at one end. Being revolved in the direction of the helical groove in the interior, the pulp is screwed through the tube to the discharge end. Naturally in the revolutions of the tube the pulp in any groove is raised out of the water, which is maintained at a definite height in the tube, and on sliding down the side of the tube the mineral particles tend to float while the gangue sinks and is carried to the next groove. This separation depends somewhat on the angle of emergence of the pulp from the water, as the greatest floating effect is obtained when the mineral is directed onto the surface of the water at a small angle.

The latest important installation of this process is in the Morning mill of the Federal Mining & Smelting Company, Wallace, Idaho. Its object will be the treatment of low-grade zinc middlings. The first unit will be of 50 tons capacity and will use about 120 tubes. If this proves successful, additional tubes will be added to give a capacity of 250 tons per day. The plant should be in operation in May or June.

Cyanide Tailings Treatment.

Another interesting plant which will be in operation about the same time as the one just mentioned is that of the United States Reduction & Refining Company, at Florence, Col. At this place are the large tailings dumps of the old Union plant and the Metallic Extraction Company. There are upward of 300,000 tons of material in each dump, representing the tailings from the early application of the cyanide process on Cripple Creek ores.

The success which the company has met with in cyaniding old tailings at the Standard plant at Colorado City has led to the erection of the present plant at Florence. About 400 tons of tailings will be treated daily at the new plant, and a good net profit is anticipated.

There has been some talk about starting the smelter owned by this company at Canon City. This plant formerly treated zinc ores and produced pigment, but it has not been operated for several years. Several proposals have been made by prospective

leasers to again operate it on zinc ores, but no definite results have transpired. Experiments also have been made by outsiders on the old dump of the smelter to see whether a profitable extraction of zinc could be made from it, but for some reason or other no further action was taken.

Tungsten from Nevada.

The first shipment of tungsten concentrates from White Pine County, Nevada, was made in March by the United States Tungsten Corporation. The mill is operated by water-power when water is plentiful and by an auxiliary steam plant at other seasons. The latter plant is inadequate for the needs of the mill, and consequently operations have been on a small scale during the winter. It is expected that this summer's work will result in a steadier output.

The grade of the concentrate is up to the standard of 60 per cent tungstic acid.

Mason Valley Smelter.

The construction of this plant in the Mason district, Nevada, is progressing rapidly and satisfactorily. Some time ago the contracts were let for the electric equipment, and more recently the bid of the American Bridge Company for the structural steel work was accepted. The contract calls for 400 tons of structural steel at a cost of about $40,000. This provides for the construction of blast furnace and converter building, sampling mill, power house and ore bins. The initial capacity of this custom copper smelter will be 800 tons per day, but this can be increased easily to 2000 tons on demand.

American Mining Congress.

The fourteenth annual session of the American Mining Congress will be held in Chicago, Oct. 23-28, 1911. This probably will be the last session of the congress at which the present delegate system will prevail. At present delegates to the congress may be appointed by the president, governors, scientific societies, etc., but in the future it is planned to have a congress of members only, and to have this congress represent the different sections of the United States. The reorganization scheme also includes the establishment of State chapters to carry on the particular work of the State and to send delegates to the annual congress. In this way the congress will not be so unwieldy and cumbersome as it is now, and better work can be done than before.

The Non-Ferrous Metal Market.

The non-ferrous metals were without feature on the market during April. Copper has tended to decline, as has spelter, but lead has showed a slight spring improvement. Purchases seem to be only for immediate needs.

Copper.—The market has been marked by dullness, and large business has been consummated only at some concession in price. Lake copper is quoted lower than in our last report, being 12¼ to 12⅜ cents. Electrolytic is quoted at 11.95 to 12.05 cents.

Lead.—The April market showed an advance over March, and this improvement seems to hold. St. Louis lead is quoted at 4.25 to 4.27½ cents, and New York, 4.40 to 4.42½ cents. The demand for Missouri lead has shown improvement during the month.

Spelter.—Although reports are to the effect that consumers are not well supplied, the business in spelter is quiet. Prices have shown a tendency to decline from day to day, and the market is generally lower than in March. St. Louis spelter is quoted at 5.22 to 5.27, and New York, 5.37 to 5.42.

Tin.—Transactions abroad indicate that the foreign syndicate dominates that market, but developments in the domestic market during the month showed that the influence of the London market was not strong here. April tin showed a steady increase in price as compared with March, and was quoted at 42⅛ cents, New York.

The Iron and Steel Market.

The event of the month, as regards iron market movements, was the settlement of Lake Superior iron ore prices for the season, at 50 cents below the 1910 schedule, making this season's prices as follows compared with prices in the past 10 years:

	——Old Range——		———Mesabi———	
	Bessemer	Non-Bessemer	Bessemer	Non-Bessemer
1901.....	4.25	2.85	2.75	2.35
1902.....	4.25	3.00	3.00	2.60
1903.....	4.50	3.60	4.00	3.20
1904.....	3.00	2.60	2.75	2.35
1905.....	3.75	3.20	3.50	3.00
1906.....	4.25	3.70	4.00	3.50
1907.....	5.00	4.20	4.75	4.00
1908.....	4.50	3.70	4.25	3.50
1909.....	4.50	3.70	4.25	3.50
1910.....	5.00	4.20	4.75	4.00
1911.....	4.50	3.70	4.25	3.50

Since 1907 the base guarantees have been 55 per cent iron for Bessemer ores and 51.50 per cent iron for non-Bessemer ores, in the natural state, with the phosphorus basis for Bessemer ores .045 per cent, dried at 212°. Prior to 1907 the basis for several years was 56.70 per cent for Bessemer ores and 52.80 per cent for non-Bessemers. Prices are for ore on Lake Erie docks.

The settlement of season prices was unusually late, since the season has nominally commenced, although on account of the light demand there will be relatively little ore moved before July. Occasionally season prices have been fixed before the end of the preceding calendar year. This season found a complicated situation. The advance for the season of 1910 was determined upon late in December, 1909, when the pig iron market had shown dulness and a declining tendency for more than a month. At the moment the condition was regarded as only temporary and the market was expected shortly to resume the steady upward movement which had characterized it from May to the beginning of November. Instead it declined steadily almost throughout 1910, leaving ore prices quite out of line with prices of pig iron, and even of finished steel products. The very badness of the market, however, constituted one argument for maintaining ore prices, inasmuch as merchant furnaces were left in many instances with large stocks of pig iron or ore, or both, upon which a reduction in ore for the new season would necessarily write a depreciation. The steel interests, furthermore, were intent upon preserving the existing level of finished steel prices in face of a very light demand, and decried a reduction in ore on account of the sentimental influence. A meeting of ore interests was held in Cleveland, April 11 and another of ore interests with some of the steel producers in New York, April 18. On Friday morning, April 21, the Shenango Furnace Company, commonly known as the Snyder interest, announced that it had sold two lots aggregating 200,000 tons of Mesabi Bessemer ores and four lots aggregating 550,000 tons of Mesabi non-Bessemer ores, at $4.25 and $3.50 respectively. This informal method of opening the selling season, supplanting the method of a few years ago of holding a meeting of ore interests and then announcing the result of the deliberations, is due to the desire of the ore interests to avoid the appearance of conflict with the later interpretation of the Sherman law of 1890. The ore interests are satisfied to have ore prices for the season on the basis thus indicated.

Total sales of merchant ore for the season are not likely to be heavy, and may even fall short of 5,000,000 tons. In the three years of heaviest movement, 1907, 1909 and 1910, the total ore shipped from the Lake Superior region was between 42,000,000 and 44,000,000 tons, nearly a fourth being merchant ore sold in the open market, the balance being made up of a relatively small tonnage moved on long term contracts, and moved by the producer-consumer class. Various estimates of the

prospective season movement for 1911 range from 25,000,000 to 35,000,000 tons.

April opened with pig iron production at the rate of about 26,000,000 tons a year, there having been practically no change during March, after the sharp increase in January and February. The current production was passing into steel mill, foundry and other consumption, but new orders for finished product were not being booked at a rate to support the production, and new buying decreased until the middle of the month, when the state of the iron and steel market as a whole was one of stagnation. Production began to be curtailed by the middle of the month, the United States Steel Corporation leading, as it firmly follows a policy of regulating its pig iron output to the shipments of finished steel products. Independent steel interests curtailed somewhat more slowly, and probably accumulated some pig iron during the month. The country's production rate at the close of the month was probably well under 25,000,000 tons, while shipments of finished steel products in the second half of the month were at a lower rate than this output would support. There has been little change in production by merchant furnaces, as the demand for merchant pig iron had not grown as brisk early in the year as the demand for finished steel products.

The position at the beginning of May is that buying of finished products is at a much smaller rate than early in the year, and falling quite short of justifying the rate of production, which is declining in consequence. An improvement in buying is to be expected this month, on account of the progress of the season, involving outdoor work and the depletion of stocks due to the extreme timidity of buyers late in March and during April, but it is questionable whether there will be sufficient increase to arrest the decline in production, which has been forced by the deficiency in buying already noted.

Prices of finished steel products are unchanged, and producers do not regard the reduction in ore as cause for any revision. As to both pig iron and finished steel products, the reduction in ore is held to have been discounted long ago by the declines in the more finished products.

Pig Iron.

The market has been extremely quiet, the month's turnover being the lightest for months. Prices have not changed appreciably. Early in the month the premiums demanded for second half delivery began to disappear, the south leading as usual, so that it became possible to buy southern iron for deliveries to the end of the year on the basis of $11, Birmingham. At the same time it appeared probable that for prompt delivery the price could be shaded 25 cents, but this has not been very seriously tested. The reduction in ore prices brought out considerable enquiry for northern iron, but this enquiry seemed to arise purely from a desire to test the market and upon furnaces quoting former prices business did not result. The furnaces insist that the ore reduction was discounted months ago by declines in pig iron, but the matter will ultimately depend upon the competition. The lake front merchant furnaces have a margin above cost which they could if necessary shade, while the Mahoning and Shenango Valley furnaces, whose selling prices have been practically at cost, based on the reduced ore prices, have large stocks, and the competition may force them to accept losses. We quote at valley furnaces, 90 cents higher delivered Pittsburg: Bessemer, $15; basic, $13.75; No. 2 foundry, $14; malleable, $13.75 to $14.

Steel.

Shipments of billets and sheet bars on contracts have decreased somewhat. There has been no new business of importance, but on a few odd lots of open-hearth billets and sheet bars regular prices have been shaded. We continue to quote billets at $23 and sheet bars at $24, f.o.b. maker's mill, Pittsburg or Youngstown, with rods nominally at $29, Pittsburg.

Finished Steel.

The United States Steel Corporation's unfilled orders decreased 3,252,274 tons during 1910, and increased 436,162 tons in January, 289,624 tons in February and 46,758 tons during March, standing at 3,447,301 tons on March 31. A decrease during April is expected to be shown by the report to be made public May 10.

Finished steel products are practically unchanged in price. There has been somewhat more shading in sheets, but the former basis is at least nominally in force. The advance of 5 cents per 100 lb. scheduled for April 1 in spikes became effective in Pittsburg territory, but not in the West, so that the spike market stands at $1.60, Pittsburg, and $1.65, Chicago, base, subject to extras according to the new list adopted February 1 last. Regular price f.o.b. Pittsburg, except where otherwise stated, are as follows:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, 1.40 cents for tank quality.

Shapes, 1.40 cents for beams and channels, 3 in. to 15 in. inclusive, zees and angles, 2 x 3 and larger.

Steel bars, 1.40 cents, base; iron bars, 1.35 cents, Pittsburg; 1.20 to 1.25 cents, Chicago.

Wire nails, $1.80, base, per keg; plain wire, 1.60 cents, base; galvanized barb wire, 2.10 cents; painted barb wire, 1.80 cents.

Black sheets, 28 gage, 2.20 cents; galvanized, 3.20 cents; blue annealed, 10 gage, 1.65 cents; painted corrugated roofing, $1.55 per square; galvanized, $2.75.

Tin plates, 100-pound cokes, $3.70 per box.

CORRESPONDENCE.

Separation of Oxygen by Cold.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Mr. J. J. Smith's letter deals with two points: one theoretical, the other practical. Taking the theoretical first, I have nothing to add to your note at the end of his letter. Mr. Smith has evidently taken Clausius' equations for reversible processes and applied them to irreversible processes. The increase of entropy due to the mixture of two inert gases is not an invention or fallacy of my own; it is quite well known, and Mr. Smith will find it discussed in the works of his own eminent countryman, the late Professor Willard Gibbs.

As to the discrepancy between my figures and those of M. Marchis: the first proofs of my paper had a slip in arithmetic and a few copies were sent out before I corrected it. I merely mixed up watts and kilowatts, a small matter which sometimes gives unexpected results.

At any rate the result astonished and disappointed me until I realized its absurdity and looked for the error. The figure ought to be 120 kilowatts, not 120,000; the prices coming out at about 25 cents a ton. The error justified Mr. Smith in attacking the theory on the ground that the result proved its error.

As this error of mine may have got about I am especially glad of a chance of correcting it.

London, England. James Swinburne.

[The sixth and fifth paragraphs from the end of Mr. Swinburne's paper, that is, the last three lines in the first column and the first nine lines in the second column of page 36 of our January issue, should read as follows, according to the corrected copy of the paper which Mr. Swinburne has kindly sent us:

"Assuming the ideal energy needed to separate a kilogram of air into its gases to be 40,000, the question is: What would it cost in practice? On a large scale it may be safe to say 100,000 joules, or a ton would need 120 kw hours. Taking the cost of a kilowatt hour as low as 0.2 cent, this means 25 cents a ton.

"It certainly looks as if oxygen at this order of figure could be used for blast furnace work. Taking a blast furnace as using four and a half tons of air per ton of iron, to increase the oxygen content by 10 per cent of the oxygen content comes out at 2½ cents per ton of pig. This shows that the Linde process may be useful on a large scale."—Editor.]

Metallography and Its Industrial Importance.*

By Albert Sauveur.

This paper deals with a subject which was not taught at the Massachusetts Institute of Technology during my stay at that institution, which was not even mentioned in the classroom. Indeed, metallography at that time had just been born and had not attracted greater attention than is generally bestowed on infants outside the family circle.

If I have contributed a mite to the growth of the child—to the progress of metallography—why, then, should my Alma Mater be entitled to any credit? Why should I not claim for myself and retain undivided whatever merit be attached to the contribution of my mite?

The reply is obvious. While I did not acquire even the rudiments of metallography in passing through the institute I entered it with an untrained mind and left it with a mind satisfactorily trained for scientific pursuit. This is my greatest debt of gratitude to my Alma Mater. It is why the institute is entitled to the lion's share of the aforesaid credit.

This incident is only a passing illustration of what, to me, is becoming more and more evident as the years go by, namely, that it is not the knowledge acquired at the school or university which is later of vital importance to us, but rather the training of the mind which the acquirement of that knowledge implies. It is this training—this peculiar attitude of the mind—which distinguishes an educated from an uneducated man.

Have not most of us forgotten a large portion of the knowledge obtained at college 10 years after graduation? Do we cease then to be educated men? Are we less educated? Clearly not, for we retain, and possibly have developed further, that mental attitude—that way of looking at things—which distinguishes the educated man.

If this be so does it matter then what we do study while at the university or technical school? While I am not prepared to say that this is immaterial provided we study enough to train our mind, I do not believe that the selection of courses has the importance generally attached to it. I do believe that the educational value of a subject depends more upon its mind training power than upon the specific kind of knowledge which it aims to impart.

Hence the value of mathematics, even for those who never expect to be called upon to perform, after leaving college, more than the four elementary operations. It is, of course, evident that the natural bent of the mind should be followed in the selection of studies—that persons with decided scientific inclination should take up chiefly scientific studies, those with literary taste, classical subjects, etc.—but the value of greater specializing for the purpose of education is, to me, becoming daily more doubtful.

A scientific education should be obtained in four years, during which a sound knowledge of the pure sciences and the rudiments of the most important applied sciences should be acquired, thereby securing the desired mental attitude and scientific way of thinking. If the student then desires to take up as his life's work the application of science in some well-recognized field, such as one of the various engineering professions, his general scientific training should be followed by a course of two years of as practical a character as possible and devoted exclusively to applied science in his chosen field.

During these two years specializing could hardly be carried too far. The student, for instance, should make up his mind not only to specialize in mining and metallurgy, but in mining *or* metallurgy, and in the latter case whether his special training will be in the metallurgy of iron and steel or of the non-ferrous metals. Should he select iron and steel, his two years of post-graduate work should be devoted exclusively to that one subject, or practically so.

In short, for the purpose of education and general fitness to take up a man's burden, let us provide a program of study as broad as possible, avoiding specialization, while for preparation in any one field of applied science let us frankly face the need of specializing; at this stage it could hardly be too excessive.

The prevailing practice is, of course, very different, the young man being made to specialize upon entering college, or possibly upon beginning his second year, and, therefore, at a period when very few can make an intelligent choice. Should he then decide to qualify himself for useful work in the manufacture of iron and steel he is required to follow for three or four years a number of prescribed studies of mining methods, ore dressing, the metallurgy of the non-ferrous metals, while learning relatively very little of his chosen subject: iron and steel. His general scientific training is narrowed while the specialized knowledge acquired is not the kind he wanted.

He leaves college a less broad man than what he might have been with specific information in his chosen field too limited for immediate useful application. Nor can it be reasonably argued that six years is too long a period of time to devote to preparation for engineering work, for it would be an admission that professional work in applied science does not call for as broad and careful training as that now required by the best schools for the practice of law or of medicine.

Twenty years ago the science of metallography was practically unknown and it is only within the last fifteen years that it has been seriously considered by metal manufacturers and consumers as a valuable method of testing and investigation. That so much has been accomplished in so short a time is highly gratifying to the many workers, practical or scientific, who have contributed by their efforts to the progress of metallography.

To realize the practical importance of metallography it should be borne in mind that the physical properties of metals and alloys—that is, those properties to which these substances owe their exceptional industrial importance—are much more closely related to their proximate composition than to their ultimate composition, and that microscopical examination reveals, in part at least, the proximate composition of metals and alloys, whereas chemical analysis seldom does more than reveal their ultimate composition.

It will bear repeating that from the knowledge of the proximate composition of a certain industrial metal or alloy we are able to infer its properties and, therefore, predict its adaptability with a much greater degree of accuracy than if we knew only its ultimate composition.

The analytical chemist may tell us, for instance, that a steel which he has analyzed contains 0.50 per cent of carbon, without our being able to form any idea as to its properties, for such steel may have a tenacity of some 75,000 lb. per square inch or of some 200,000 lb., a ductility represented by an elongation of some 25 per cent, or practically no ductility at all; it may be so hard that it can not be filed or so soft as to be easily machined, etc.

The metal microscopist, on the contrary, on examining the same steel will report its structural, i.e., its proximate, composition, informing us that it contains approximately 50 per cent of ferrite and 50 per cent of pearlite, and we know at once that the steel is fairly soft, ductile and tenacious, or he may report the presence of 100 per cent of martensite and we know that the steel is extremely hard, very tenacious, and deprived of ductility.

Which of the two reports is of more immediate practical value, the chemist's or the metallographist's? Surely, that of the metallographist.

Nor is it only in the domain of metals that we find such close relationship between properties and proximate composition, for, on the contrary, it is quite true of all substances. How many organic bodies, for instance, have practically the same ultimate composition and still are totally unlike in properties because of their different proximate composition, i.e., different grouping

*A paper presented at the Congress of Technology at the 50th anniversary of the granting of the charter of the Massachusetts Institute of Technology.

and association of the ultimate constituents. If we were better acquainted with the proximate composition of substances many unexplained facts would become clear to us.

Unfortunately the chemist too often is able to give us positive information in regard to the proportion of the ultimate constituents only, his reference to proximate analysis being of the nature of speculation. Ultimate analysis has reached a high degree of perfection in regard to accuracy as well as to speed of methods and analytical chemists have built up a marvelous structure calling for the greatest admiration. Their searching methods never fail to lay bare the ultimate composition of substances. But how much darkness still surrounds the proximate composition of bodies and how great the reward awaiting the lifting of the veil!

The forceful and prophetic writing in 1890 of Prof. Henry M. Howe, M. I. T., '71, naturally comes to mind. Speaking of the properties and constitution of steel, Professor Howe wrote:

"If these views be correct, then, no matter how accurate and extended our knowledge of ultimate composition, and how vast the statistics on which our inferences are based, if we attempt to predict mechanical properties from them accurately we become metallurgical Wigginses. * * *

"Ultimate analysis never will, proximate analysis may, but by methods which are not yet even guessed at, and in the face of fearful obstacles.

"How often do we look for the coming of the master mind which can decipher our undecipherable results and solve our insoluble equations, while if we will but rub our own dull eyes and glance from the petty details of our phenomena to their great outlines their meaning stands forth unmistakably; they tell us that we have followed false clues and paths which lead but to terminal morasses. In vain we flounder in the sloughs and quagmires at the foot of the rugged mountain of knowledge seeking a royal road to its summit. If we are to climb, it must be by the precipitous paths of proximate analysis, and the sooner we are armed and shod for the ascent, the sooner we devise weapons for this arduous task, the better.

"By what methods ultimate composition is to be determined is for the chemist rather than the metallurgist to discover. But, if we may take a leaf from lithology, if we can sufficiently comminute our metal (ay, there's the rub!) by observing differences in specific gravity (as in ore dressing) in rate of solubility under rigidly fixed conditions, in degree of attraction by the magnet, in cleavage, luster, and crystalline form under the microscope, in readiness of oxidation by mixtures of gases in rigidly fixed proportions, we may learn much.

"Will the game be worth the candle? Given the proximate composition, will not the mechanical properties of the metal be so greatly influenced by slight and undeterminable changes in the crystalline form, size and arrangement of the component minerals, so dependent on trifling variations in manufacture as to be still only roughly deducible?"

The above was written before the days of metallography, or at least when metallography had barely appeared in the metallurgical sky and when no one yet had fancied what would be the brilliant career of the newcomer. Metallography has done much to supply the need so vividly and timely depicted by Professor Howe, precisely because by lifting a corner of the veil hiding from our view the proximate composition of metals and alloys it has thrown a flood of light upon the real constitution of these important products. Has the game been worth the candle? Will any one hesitate to answer in the affirmative Professor Howe's question?

Professor Howe with his usual acumen was conscious of the fact that proximate analysis, while likely to reveal a great deal more of the constitution of metals than ultimate analysis ever could, might still leave us in such ignorance of their physical structure as to throw but little additional light upon the subject. His fear was certainly well founded and surely if the proximate composition had been obtained by chemical analysis it would indeed have told us little of the structure or anatomy of the metals. In the domain of proximate composition chemistry can not do more for the metallurgist than it does for the physician.

Invaluable information chemistry does give without which both the physician and the metallurgist would be in utter darkness, but it throws little or no light upon the anatomy of living or inanimate matter. Its very methods which call for the destruction of the physical structure of matter show how incapable it is to render assistance in this, our great need.

The parallel drawn here between metals and living matter is not fantastic. It has been aptly made by Osmond, who said rightly that modern science was treating the industrial metal like a living organism and that we were led to study its anatomy, i.e., its physical and chemical constitution; its biology, i.e., the influence exerted upon its constitution by the various treatments, thermal and mechanical, to which the metal is lawfully subjected; and its pathology, i.e., the action of impurities and defective treatments upon its normal constitution.

Fortunately metallography does more than reveal the proximate composition of metals. It is a true dissecting method which lays bare their anatomy—that is, the physical grouping of the proximate constituents, their distribution, relative dimensions, etc., all of which necessarily affect the properties. For two pieces of steel, for instance, might have exactly the same proximate composition—that is, might contain, let us say, the same proportion of pearlite and ferrite and still differ quite a little as to strength, ductility, etc., and that because of a different structural arrangement of the two proximate constituents; in other words, because of unlike anatomy.

It is not to be supposed that the path trodden during the last score of years was at all time smooth and free from obstacles. Indeed, the truth of the proverb that there is no royal road to knowledge was constantly and forcibly impressed on the mind of those engaged in the arduous task of lifting metallography to a higher level.

Its short history resembles the history of the development of all sciences. At the outset a mist so thick surrounds the goal that only the most courageous and better equipped attempt to pierce it and per chance they may be rewarded by a gleam of light. This gives courage to others and the new recruits add strength to the besieging party. Then follows the well-known attacking methods of scientific tactics and strategy, and after many defeats and now and then a victorious battle the goal is in sight, but only in sight and never to be actually reached, for in our way stands the great universal mystery of nature: what is matter? What is life?

Nevertheless there is reward enough for the scientist in the feeling that he has approached the goal, that he has secured a better point of vantage from which to contemplate it. The game was worth the candle. And if scientific workers must necessarily fail in their efforts to arrive at the true definition of matter, whatever be the field of their labor, they at least learn a great deal concerning the *ways of matter,* and it is with the ways of matter that the material world is chiefly concerned. Hence the usefulness of scientific investigation, hence the usefulness of metallography.

Like any other science with any claim to commercial recognition, metallography has had first to withstand the attack and later to overcome the ill-will and reluctance of the so-called "practical man" with a decided contempt for anything scientific. He represents the industrial philistine clumsily standing in the way of scientific application to industrial operations. Fortunately, while his interference may retard progress, it can not prevent it. Had he had his own way neither the testing machine, nor the chemical laboratory, nor the metallographical laboratory, nor the pyrometer would ever have been introduced in iron and steel works.

In metallography, as in other fields of research, American workers, with very few exceptions, have been quite willing to let Europeans perform the arduous and generally unrewarded task of the pioneer, being content to wait before entering the

field until practical results were fairly in sight. Such course, which is never to be commended, becomes intolerable when accompanied, as it so frequently is, by the boasting attitude of the man believing himself smarter than his neighbor, whom he regards in the light of the cat drawing the chestnuts from the fire.

America, barring brilliant exceptions like Richards at Harvard and Noyes at Tech, does not as yet do her share of the pioneer's work in investigations which do not give evident indications of quick commercial returns. The unselfish, nay, self-sacrificing, spirit of the true scientist is of far rarer occurrence in the United States than it is in Europe, and especially in France. America has not yet produced a Pasteur nor a Berthelot, intellectual giants, profound scientific thinkers, whose conception of the duty of the scientist as a man is so lofty that they have despised the wealth within their easy reach to devote themselves unreservedly to the betterment of their country, or rather of the world, for they are morally so great that the entire world becomes their fatherland. Humanity claims them.

Speaking in 1904 of the practical value of metallography in iron and steel making, I wrote the following which it may not be out of place to reproduce here: "History, however, must repeat itself, and the evolution of the metallographist bids fair to be an exact duplicate of the evolution of the iron chemist; the same landmarks indicate his course; distrust, reluctant acceptance, unreasonable and foolish expectation from his work, disappointment because these expectations were not fulfilled and finally the finding of his proper sphere and recognition of his worth. The metallographist has passed through the first three stages of this evolution, is emerging from the fourth and entering into the last. For so young a candidate to recognition in iron and steel making this record is on the whole very creditable."

We may say to-day that he has definitely entered the last stage and that the adverse criticisms still heard from time to time, generally from the pen or mouth of ignorant persons, are like the desultory firing of a defeated and retreating enemy.

In the United States alone the microscope is in daily use for the examination of metals and alloys in more than 200 laboratories of large industrial firms, while metallography is taught in practically every scientific or technical school.

Harvard University.

The Florence-Goldfield mill was operated 365 days during 1910 and treated an average of 143.3 tons of ore per day. The total recovery of gold was 92.11 per cent; 58 per cent of this was shipped to the United States mint as bullion. The average cost of milling for the year was $2.085 per ton. The distribution of the total cost of milling for the year follows:

Amalgamating	$6,140.39
Assaying	1,740.60
Concentrating	12,490.45
Concentrating repairs	2,734.79
Crushing	16,704.57
Crushing repairs	9,716.57
Cyaniding	44,067.83
Cyaniding repairs	8,013.01
General superintendence and office	2,922.61
Mill superintendence	3,669.66
Power and lighting	37,038.38
Tailings	1,390.19
Watching	1,387.21
Water	6,934.24
Water repairs	377.02
Total	$155,327.52

In percentages of the total cost, the different items of cost are as follows: Cyaniding and cyaniding repairs 33.5 per cent, power and lighting 23.8 per cent, crushing and crushing repairs, 17 per cent, concentrating and concentrating repairs, 9.8 per cent, etc.

Modern Ore Dressing Practice in Gilpin County, Colorado.

By H. C. Parmelee.

A new standard has been set for concentrating mills in Gilpin County, Colorado, which is noteworthy because styles have not changed much there in the past forty years. The old milling methods, which were the result of varying conditions as to quantity and quality of ore, were just as simple and efficient as could be devised by the pioneers. Hence the development of the venerable slow-drop, high-discharge stamp mill, with inside

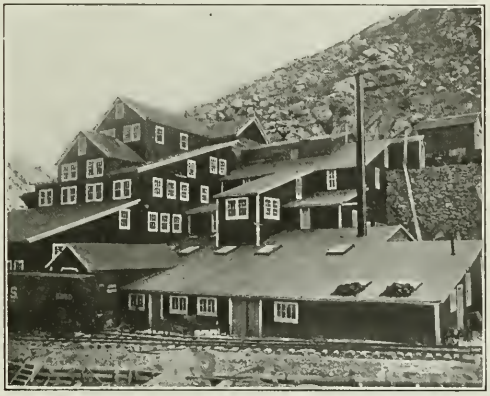


FIG. 1.—NEW 250-TON CONCENTRATING MILL OF THE FRONTENAC CONS. MINES, LTD., BLACK HAWK, COLO. OLD IRON CITY MILL IN RIGHT FOREGROUND.

amalgamation, for the treatment of surface quartz-gold ores and the Gilpin County bumper for the concentration of sulphides.

These devices probably were efficient in their day and for the class of ore they were designed to treat, but they do not adequately meet the demands of present practice. The inevitable change in other conditions has not been accompanied by a general disposition to abandon old styles of treatment and adopt new, even though the latter are of proved advantage. The tendency has been to hold to old methods, without always using the discrimination that marked their early use; the frank expression of some of the millmen being that "they are good enough." Under such conditions those who are progressive enough to depart from the old lines, even in the face of sceptical criticism, are entitled to credit and it is to be hoped that their example will be more widely followed.

The Frontenac Mill.

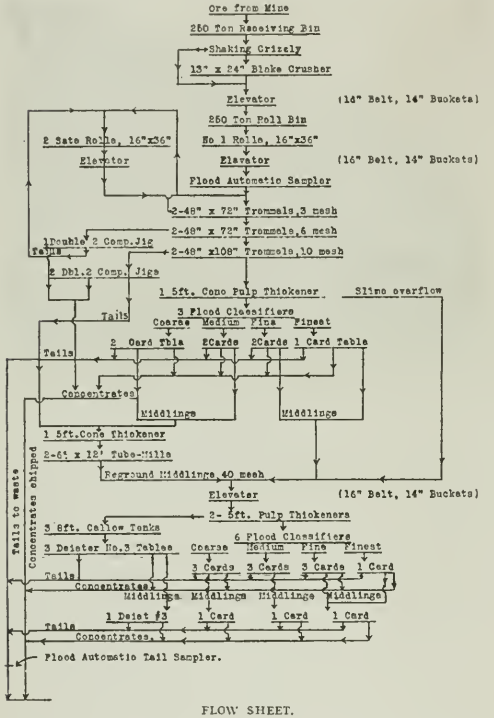
The Frontenac Consolidated Mines, Limited, an English corporation operating in Gilpin County, is represented locally by Mr. Henry P. Lowe, managing director. At his instigation the present Frontenac mill was built after two years' experimenting on methods of ore dressing to determine the best procedure and the most efficient machines. The mill, which is shown in Fig. 1, was designed by Mr. S. B. Tyler, general superintendent of the Frontenac company, and built by Peter McFarlane & Sons Iron Works Company, Denver. The view of the Frontenac mill also shows the adjoining Iron City mill in the right foreground. It was in the Iron City that Mr. Tyler conducted the preliminary experiments on Frontenac ore, testing several types of the various machines for long periods of time. The result is believed to be a rational system of treatment showing a marked advance in efficiency over former methods.

Frontenac ore is a mixed sulphide of iron, lead and copper, carrying gold and silver and occurring in a quartz and granite gangue. The minerals are mainly pyrite, with some galena and

tetrahedrite. The gold occurs mainly in the pyrite and the silver in the tetrahedrite. The pyrite and galena occur massive and are liberated from the gangue in condition for coarse concentration, but the tetrahedrite exists in thin seams and is not released save by fine grinding. This condition makes the recovery of the copper a difficult matter, as it is not until the ore is slimed that the highest saving can be made. The total value of the ore ranges from $8 to $10 per ton. The gold assays from 0.30 oz. to 0.35 oz. per ton; silver from 4 oz. to 6 oz.

Crushing, Screening and Jigging.

In general the system of treatment consists of crushing in stages, with intermediate concentration, beginning with the coarsest size at which mineral is liberated from the gangue, and ending with slime. Specifically concentration begins in coarse jigs on —3+6-mesh material and is continued in fine jigs, —6+10-mesh, and from there on three sets of tables with one intermediate regrinding.



FLOW SHEET.

The accompanying diagrammatic flow sheet shows the steps in the treatment. Ore is delivered at the head of the mill by the district tramway and dumped into a 250-ton receiving bin, from which it flows through ordinary rack-and-pinion gates to the coarse crusher. This machine is of the Blake type, and the largest in the district, having a jaw opening 13 in. x 24 in., and a capacity of 750 tons in 24 hours. All coarse crushing, therefore, is done on one eight-hour shift, which is ample to provide the tonnage for 24 hours.

The crushed ore, reduced to a maximum size of 2 in., is elevated to a bin of like capacity to the crude-ore bin. From here it flows to a hopper fitted with a plunger feeder which discharges to the roughing rolls. These are 16 in. x 36 in. in size, making 60 r.p.m., and set ¼ in. apart. Their product passes to an elevator which feeds the trommel screen line.

The trommel line is in duplicate, three trommels in each train; the first two are 4 ft. x 6 ft., and the third 4 ft. x 9 ft. Diagonal slotted punched screens are used, the first having 9 mm opening, the second 3 mm and the third 1½ mm, corresponding to 3, 6 and 10-mesh respectively. The oversize from the first trommel is returned to the regrinding rolls, of which there are two sets, 16 in. x 36 in., making 80 r.p.m., and set close together. The reground pulp is returned to the screen line.

The undersize of the first trommel passes to the second, and that of the second to the third. The oversize of the second and third trommels comprises the feed for the coarse and fine jigs respectively. The final undersize of the trommel line is classified for table treatment.

Three double two-compartment Traylor jigs receive the screen products as stated above. All jigs make hutch concentrates—a pyrite-galena mixture. The coarse jigs treat —3+6-mesh material. They are run at 220 r.p.m. with a ¾-in. stroke. They are fitted with 4-mesh screen of No. 15 wire. The fine jigs treat the —6+10-mesh pulp, have a speed of 280 r.p.m. and a stroke of 5/8 in. Their screens are 6-mesh of No. 13 wire. Coarse jig tailings are reground in rolls with the first trommel oversize and returned to the screen line, while fine jig tailings are reground in tube mills and treated in the slime department.

The jig concentrates are dewatered and delivered to a bin by means of a shaking launder having a perforated screen bottom for a portion of its length. The launder is suspended from above by 1-in. boards, and has a fall of about 1 in. to the foot. A knocker movement at the upper end throws the concentrates toward the discharge, dewatering them as they pass over the screen bottom. This is a device of Mr. J. J. Fitzgerald, mill foreman, who first installed it some years ago at the Humphreys mill, Creede, Col.

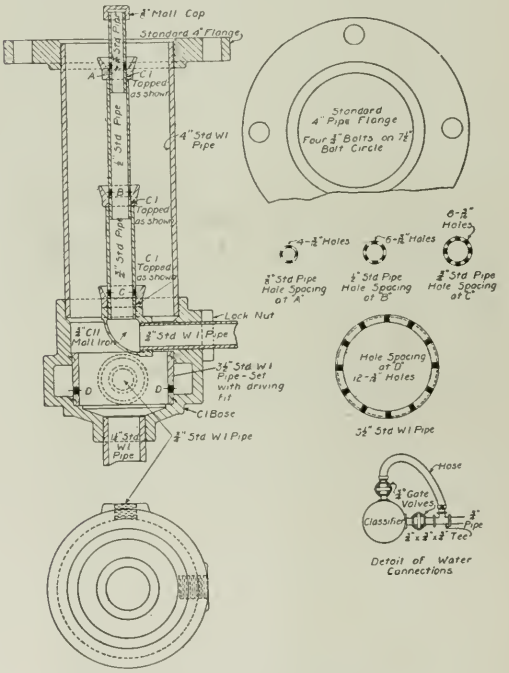


FIG. 2.—DIAGRAM OF FLOOD CLASSIFIER.

Classification and Table Treatment.

The undersize of the 10-mesh trommel is classified for table treatment by Flood classifiers. This device, which is illus-

trated in Fig. 2, is a hydraulic tubular classifier, several of which may be arranged in a launder to receive successively that part of the pulp not taken out by the one immediately preceding. It consists of a vertical tubular sorting column, with a centrally located water pipe entering from below, capped at the top and perforated with three annular series of holes for the escape of the hydraulic water.

As the pulp flows into the sorting column it encounters the upward current of water from the annular jets, and each particle of pulp sinking to the bottom of the tube must pass each series of jets before it can escape through the discharge spigot. That portion of the pulp which is too light to sink passes on to the next classifier, where the water pressure and discharge spigot are regulated to permit a further classification. Additional water is added through a circular chamber surrounding the bottom of the sorting column. This equalizes the pressure due to the height of the column and enables the classifying water to work without interference from counter currents.

These classifiers require only a small quantity of water, which is a consideration at the Frontenac mill. Further, they are not easily choked with foreign matter, nor do they permit the accumulation of a bank of solids at the top which, by becoming suddenly dislodged, may fall into the sorting column in a mass and escape classification.

Taking up the flow of ore from the last trommel, it passes to a thickening box, the overflow of which goes to the slime department and joins the tube mill discharge. The thickened underflow passes to three Flood classifiers, which give three classes of sand pulp and a fine overflow. These four products are treated on seven Card tables, each classifier discharge on two tables and the fine overflow on one.

Fig. 3 shows this section of the mill. This set of tables makes

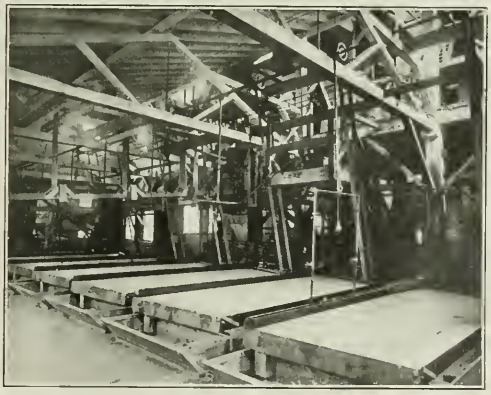


FIG. 3.—VIEW OF CONCENTRATING FLOOR FRONTENAC MILL, BLACK HAWK, COLO.

two concentrates, lead and iron. The former is rich in gold which has been liberated from the pyrite, while the latter carries some of the copper mineral, the specific gravity of which approaches that of pyrite. The slimes and middlings from these tables are retreated as shown in the flow sheet, the coarsest being first reground in tube mills with the fine jig tailings.

The use of the tube mill as a regrinder of middlings is still new in practice, although the idea seems to have been accepted as sound. By regulating the various elements which contribute to the net result it seems possible to get a wide range of products from the tube mill. Consistency of pulp, quantity of pebbles, size and speed of the tube can be so regulated as to make the mill adaptable to comparatively coarse grinding for concentration or to fine grinding for agitation in cyanide solution.

The tubes at the Frontenac mill, shown in Fig. 4, are 6 ft. x 12 ft. in size and are run at a speed of 21 r.p.m. They have silex lining and carry a load of about 7000 lb. of Norway pebbles. Only one tube is now in operation, and it is grinding about 60 tons of solids in 24 hours. The tube feed, as already stated, is the tailings from the jigs treating $-6+10$-mesh pulp and coarse middlings from the Card tables. The tube product is about 40-mesh material, with only a small percentage of -100-mesh. The machines have not been in regular operation long enough to give average screen analyses of the product.

The combined slimes and reground middlings are elevated, as shown in the flow sheet, to pulp thickening boxes, the underflow of which is classified in six Flood classifiers for

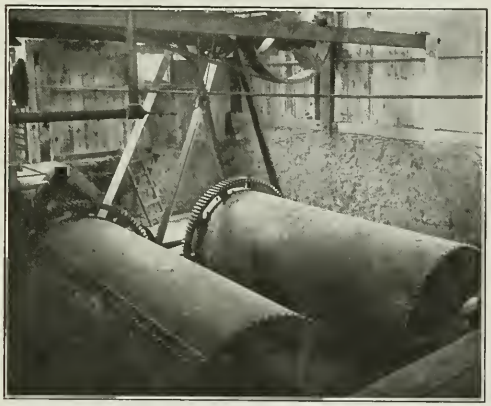


FIG. 4.—TUBE MILLS USED FOR REFINING TABLE MIDDLINGS AND JIG TAILINGS, FRONTENAC MILL, BLACK HAWK, COLO.

treatment on 10 Card tables. The overflow of the thickeners is further thickened in three 8-ft. Callow tanks for treatment on three Deister No. 3 slime tables.

Fig. 5 gives a view of the Callow tanks, with tube mills in the pit in the background, Card tables on the right and Deisters on the left. Owing to temporary water shortage the overflow of the Callow tanks is further clarified in thickening boxes and used for table wash water. The tables in this section make but one class of concentrates, practically all the galena having been removed in the jigs and coarse tables. Fig. 6 gives a view of part of the first division of the slime department.

Retreatment of the middlings from the Card and Deister slime tables is accomplished on additional tables of the same kind on a lower floor. These also make but one concentrate, which is richer in copper than the others owing to the naturally fine state of the copper mineral and its tendency to form slime.

Assays.

The following table gives average assays of the five classes of concentrate. It will be observed that the silver is nearly uniform throughout. Practically all the lead is removed in the jigs and first set of Cards, while the copper is recovered principally in the slimes. Each class is shipped separately on account of the varying percentage of silica which, according to smelter contracts on concentrates, is subject to a penalty when it exists in quantities above, say, 15 per cent.

	Gold oz. per ton.	Silver oz. per ton.	Lead per cent.	Copper per cent.	Silica per cent.
Jig concentrate	0.70	12.00	10.00	1.20	10.00
Card iron concentrate	0.60	11.00	.....	2.00	9.00
Card lead concentrate	5.00	12.00	37.00	....	.....
First slime department	0.45	12.00	.....	3.00	20.00
Second slime department	0.40	12.00	.....	3 to 4	30.00

The ratio of concentration is 4:1, and the percentage recovery, figured on shipments, is 85 per cent on gold, 78 per

cent on silver, and 75 per cent on lead and copper. This compares very favorably with recoveries of 70 per cent on gold and 60 per cent on silver when the same ore was treated by stamp crushing to 20-mesh and simple concentration in the Iron City mill.

Power and Water.

The mill building is of frame construction, steam heated and electric lighted. Electric power is received from the lines of the United Hydro Electric Power Company, Georgetown, Col.

FIG. 5.—CALLOW THICKENING CONES SHOWN IN RELATION TO TUBE MILLS AND ELEVATOR, FRONTENAC MILL, BLACK HAWK, COLO.

Six General Electric 440-volt induction motors are distributed as follows:

50 hp for crusher and elevator.

100 hp for three sets of rolls and two elevators.

25 hp for three double jigs, six trommels and seven Card tables.

20 hp for 13 Card and 5 Deister tables.

75 hp for two tube mills and elevator.

15 hp for Deming three-plunger pump, size 9 in. x 10 in.

The total power requirement has been found to be 176 hp at peak load and 150 hp on running load.

Water is obtained from Clear Creek, but can be used only after passing several settlement basins for the removal of sand and slime. It is then pumped to storage reservoirs on the hillside. Good water supply is a problem in this district, but the Frontenac company is making arrangements for an independent supply, which will be available probably this year.

The mill has been in operation since the middle of February, and at the time these notes were obtained had been running only five or six weeks. Hence it has been impossible to present some data, such as screen analyses, etc., which will be available later when those in charge have time for such details. Enough has been presented, however, to show a marked departure from former milling methods in this district, and acknowledgment is due to Mr. Lowe, Mr. Tyler and Mr. Fitzgerald for courtesies extended and information given for publication.

The Determination of Manganese in the Presence of Chromium and Tungsten.

By Thos. C. Watters.

The accurate determination of manganese in high-speed and air-hardening tool steel, or in any alloy steel containing chromium, has given the metallurgical chemist considerable trouble, both in the volumetric and color methods, owing to the fact that in oxidizing the manganese to permanganic acid part of the chromium present is oxidized to chromate and has an action similar to that of the permanganate upon reducing agents used in volumetric determinations and in the color method the yellow color of the chromate interferes with the pink produced by the manganese and destroys its accuracy. We are

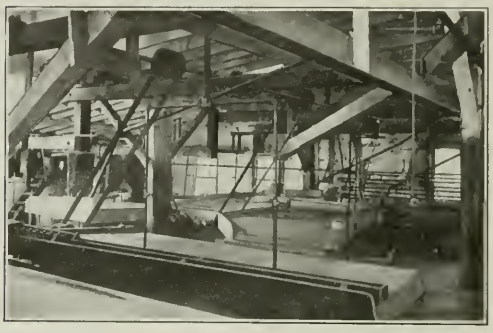


FIG. 6.—SLIME CONCENTRATING DEPARTMENT, SHOWING CARD AND DEISTER TABLES, FRONTENAC MILL, BLACK HAWK, COLO.

using a method in our laboratory which is a combination of the Volhard and bismuthate methods and are inclined to think the application original.

Two grams of drillings are dissolved in 20 c.c. of sulphuric acid (1.2 Sp. Gr.), enough water being added from time to time to keep the iron sulphate in solution. The solution when complete is oxidized by adding 5 c.c. of nitric acid (1.42 Sp. Gr.) and then evaporated until fumes of sulphuric acid are given off and the carbonaceous matter is destroyed; 100 c.c. of water are added and the solution of the ferric sulphate effected by boiling. The liquid and residue are washed into a graduated 500-c.c. flask and a solution of sodium carbonate added until the solution becomes dark in color and the precipitate formed by the addition of the carbonate dissolves with difficulty.

An emulsion of zinc oxide and water is then run into the flask a little at a time and the flask shaken after each addition until all the iron and chromium have been precipitated. The volume is then made up to 500 c.c. with water and the precipitate allowed to settle; 250 c.c., equivalent to 1 gram of sample, of the supernatant liquid is decanted, through dry filter and transferred to an Erlenmeyer flask. The solution is acidified with 25 c.c. of nitric acid (1.42 Sp. Gr.) and one gram of sodium bismuthate added.

It is well to shake the flask for several minutes after the addition of the bismuthate and then allow the residue to settle for a few minutes, after which it is filtered through asbestos, the filter pump being used. The flask and filter are washed with water containing a little nitric acid until free from the pink color of the permanganic acid. The filtrate is transferred to a beaker and a measured volume of ferrous sulphate solution, in excess of the permanganic acid, is dropped into the solution to reduce it. Its excess is titrated with a standard solution of potassium permanganate until a delicate pink color is obtained.

The ferrous sulphate solution should be made equivalent in strength to the permanganate. The difference between the num-

ber of c.c. of ferrous sulphate solution added and the number of c.c. of permanganate used to titrate the excess will give the number of cubic centimeters of permanganate which are equivalent to the manganese in the steel. By multiplying the number of c.c. of permanganate found by its manganese value the weight of manganese in the sample is found.

A solution of convenient strength for steel of low manganese content is made by dissolving one gram of potassium permanganate to a liter of water. It is standardized in the usual way, either with iron wire or ammonium ferrous sulphate, to find its exact value per c.c. in iron. Its iron value multiplied by the factor 0.1968 gives its value in manganese.

The ferrous sulphate solution may be made of such a strength as to exactly equal the permanganate solution, or, if not quite equal in value, a blank titrate is made and its value in c.c. of the permanganate solution found. Fifty c.c. of sulphuric acid to the liter will keep the ferrous sulphate from crystallizing out of solution.

I trust that this method of determining manganese in the presence of chromium may prove of value to others who desire to combine accuracy with reasonable speed, as it has to me.

Columbia Tool Steel Company,
Chicago Heights, Ill.

The Grinding and Screening of Clays.

The inherent difficulties in grinding and screening substances of a clayey nature have given rise to the development of machinery specially adapted to this purpose. The success of these machines in the various branches of the clay industry has led to their adoption also in some metallurgical plants where the conditions are exceptional and where the usual devices are not applicable. There has just been issued by the Geological Survey of Ohio *Bulletin* No. 11, entitled "The Manufacture of Roofing Tiles," by W. G. Worcester and Edward Orton, Jr., containing the latest data on actual practice in grinding and screening clays, from which we take the following excerpt:

The purpose in grinding clays preliminary to manufacture by the plastic process is twofold: First, to break up any obnoxious minerals which occur in coarse size and disseminate them through the mass; and, second, to create plasticity in clays in which that property lies dormant or undeveloped. While both purposes are probably accomplished in grinding any given clay, only one may be, and usually is, in view. Thus in soft, plastic clays there is need only of crushing and disseminating mineral impurities, while in homogeneous shales the principal requirement is the development of plasticity..

Grinding is practised both wet and dry. Wet grinding is used where defective plasticity is found, as it is the best known method for developing latent plasticity; but at the same time the method is inefficient in securing physical homogeneity. Hence this treatment is usually applied to hard shales and fire clays. In general, dry grinding is practised where the material is unhomogeneous naturally, and where its lack of homogeneity would cause trouble if not overcome. It is followed ordinarily by screening, which establishes the limit of size of the particles and insures uniformity. The dry screened clay is then converted to the plastic state by a subsequent process.

Thus it is seen that the wet and dry grinding processes are more than alternative routes to the same goal. The choice of treatment lies mainly in the judgment of the operator. The wet treatment is slower, more costly, and has less control of the homogeneity of the product, but produces far the best plasticity. In general any clay *can* be treated by this method, but it is uncommon to use save where necessary for the reasons given. The dry treatment is faster, cheaper, produces a more homogeneous paste, but of deficient plasticity. It can be used with clays of good or moderate plasticity, and wherever possible will be used for the reasons given.

The following observations on machines are from actual installations in roofing tile plants in the United States:

Rolls.—The roll is one of the simplest and oldest forms of grinding machines. Essentially it consists of two cylinders or cones whose surfaces are in close contact or held closely parallel to each other. Their axes must be in the same plane, and in the case of cylindrical rolls are parallel, but in the conical rolls the axes are inclined to each other by an amount proportional to the taper of the cones. Rolls are designed for two different ends: First, as a wet grinding machine for crushing stony, plastic clays, and for rejecting the rocks or hard portions which are too large to fall within the angle of nip; second, as a dry grinding tool for pulverizing large quantities of nearly uniform size of grains. The commonest use of rolls in this way is as a tailings grinder to finish the grinding of small particles which are partly reduced, but not enough so to pass the screens.

Wet Grinding Rolls.—At the plant of the Detroit Roofing Tile Company, Detroit, Mich., differential rolls are used for grinding a fine grained alluvial clay which is so plastic as to be difficult to grind in a strictly dry grinding machine like a dry pan. One roll is small and driven at a high speed, and the other is larger, driven at lower speed. They are set within a fraction of an inch of each other. The small roll, about 9 in. or 10 in. in diameter by 2 ft. long, runs at approximately 500 r.p.m., while the large roll, about 18 in. in diameter, makes but 150 r.p.m. The peripheral speed of these two rolls is quite different, so that when clay is fed into the machine it not only squeezes and crushes it, but also shreds and tears it apart as well. This machine is able to take clay either dry or wet and carry it through. As the clay falls from the rolls it is put through a pug mill and stored in aging bins. At another plant both rolls are of the same size, 2 ft. long and 30 in. in diameter, running at the same speed. These rolls also are followed by a pug mill, the combination being admirably suited to the needs, which are principally a thorough mixing of the different layers as they come from the pit and a thorough incorporation of an added ingredient which is foreign to the clay itself.

Dry Grinding Rolls.—The grinding of glacial clays presents a problem not found in alluvial clays and shales on account of the occurrence of boulders, gravel, sand, etc., which have been picked up by glacial action. Glacial clays are thus very heterogeneous in character, and their preparation offers a special problem. At one plant treating this kind of clay a pair of conical dry crushing rolls, smooth surface, set 1 in. apart, and running at 75 r.p.m., handles about nine tons per hour. In size they are 2 ft. long by 12 in. at the large end and 7 in. at the small. These rolls crush the lumps of clay and smaller pebbles; any stones that are too large for the angle of nip gradually work their way to the large end of the cones, where they fall out into a wheelbarrow and are removed. The use of these rolls is chiefly for their qualities as stone separators, but the next step in the process would hardly be successful if the clay had not first been crushed and shredded.

Dry Grinding Pans.—No other tool wholly takes the place of this as a rough-and-ready grinder, in which the material comes to it in widely varying sizes, degrees of hardness, degrees of plasticity and in rates of feeding. Irregularities in all these points may occur in any works and on every dry pan. No machine that cannot give a good account of itself under such conditions will ever displace the dry pan as a clay-grinder's main reliance. The step bearing which carries the weight of the vertical shaft is one of the most essential features about a dry pan. Usually a load of five to ten tons is carried on a bearing not as a rule exceeding 5 in. in diameter. This bearing tends to heat very easily and unless proper oiling facilities are provided it will be very short-lived. The principal thing is to secure positive and copious lubrication. Nearly

any step, if of adequate size, will stand well if kept thoroughly lubricated and clean. In selecting a pan one should try to secure one that is strongly and well built. By this is meant one that has every joint and bearing planed and fitted, every bolt hole drilled and machined bolts that properly fit the holes. There is not another machine about a clay plant that is expected to stand the rough usage that the dry pan is called upon to endure. It is entirely impossible to say that all the virtues are found in any one design, and there is no best pan in the sense of one filling all situations best.

Disintegrators.—A method of grinding not used in any other roofing tile plant in this country is found in the works of the National Roofing Tile Company, Lima, Ohio. Here a Williams pulverizer is used. The machine is not uncommon in other branches of industry, such as coal and cement. It consists of a horizontal shaft to which are keyed a series of disks, through the circumference of which bars are run, and on the latter are hung swinging hammers, which, by centrifugal force, fly straight outward on a line from the center of the shaft when the machine is rotated at high speed. The hammers strike a very powerful blow, but not being rigidly fixed are free to move backward when they encounter a body heavy enough to overcome the centrifugal force. Clay being subjected to such severe treatment as this is completely broken up before leaving the machine. It is found advisable to bring the clay to the disintegrator as dry as possible, and even then provision is made to steam-heat the jacket of the mill in order to prevent the clay from packing in it. For clays of a hard, compact nature, this machine will unquestionably break up massive lumps and reduce the whole to a powder, but if the clay be fat and sticky it would be impossible to work it in such a machine.

Screening.—After a clay has been through a pan or equivalent, and is in the form of a coarsely granulated powder, the screening still remains difficult if the moisture exceeds 5 per cent or 6 per cent. Caking on the screen, covering the holes, agglomerating and rolling on the surface without passing through, sticking in elevator cups, choking spouts and runways are troubles incident to dry clay screening.

The purpose of screening is to set a limit on the maximum size of particles. As the coarsest particles in a ground mixture also are likely to be the hardest, and also of different chemical composition from the matrix of fine aluminous material, it follows that these coarse particles are both difficult to grind finer and are the most injurious to the clay if not reground and made to blend well into the matrix. Screening is therefore necessary to establish a limit beyond which particles cannot enter the clay body except by accident. It is a very necessary part of the process of homogenization, of which grinding and tempering are the two principal steps.

On account of the troublesome nature of clays in the screening process, the actual size limit in clay plants is often much greater than is desired by the manufacturer, because without predrying he cannot either grind or screen effectively. Roofing tile clays in general are passed through from 12-mesh to 20-mesh screens or equivalent. Some form of the inclined screen, either stationary or movable, is most generally used.

Stationary Inclined Screens.—These consist of an inclined chute, the bottom of which is covered with punched plate, over which the pulverized clay passes. The size of the perforations and the angle at which the screen is hung govern the size of grain that will pass through it. Should the screen be horizontal particles the same size as the openings would pass, but upon elevating one end the particles attain a momentum and those passing through will be smaller and smaller as the angle of elevation increases. Hence, inclined screens are made adjustable as regards angle of elevation.

The fixed inclined screen gives considerable trouble through its tendency to blind, especially when the clay is damp. This means constant attention to secure effectiveness. Only one roofing tile plant in the country is using the fixed screen, and the material is a mixed sandy clay, screened through a 16-mesh screen, which is a little coarser than the average mesh used in this work.

Piano Wire Screen.—A recent form of inclined screen is made of heavy steel piano wires, strung on a strong substantial iron frame about 30 in. wide by 6 ft. long. The wires are drawn to a high tension and provision is, or should be, made to prevent slipping. At each end of the frame the wires are drawn over a rod threaded to suit the space required between the wires. By varying the pitch of these threads the mesh can be changed. The constant rigidity of the wires is an important requirement in order to insure regularity of the product. When properly constructed and kept in condition this screen has large capacity, does not blind easily, requires no power and little head room. In the roofing tile industry it has been found that on soft, easily slaking shale good results are obtained, while on hard, sandy shales the results are not all that could be desired.

The principal trouble lies in keeping the wires rigid enough to give a uniform product.

Traveling Inclined Screens.—The main difference between this and the stationary type is that the screening surface is composed of punched metal plates, constantly in motion on an endless chain. The object of the device is to bring automatically every part of the screen surface over a rotary cleaning brush, instead of depending on its being done by hand, as is the case with stationary inclined screens. The moving screen is made up of sectional screen plates, attached to sprocket chains on each side. These chains travel upward on the frame, over sprocket wheels at the top and down on the under side. A spreading board at the top delivers the clay onto the screen, after which it slides down over the plates which are moving in the opposite direction. Particles of proper size drop through the screen and the larger tailings flow over the end to be reground.

The screen plates may be made of piano wire or screen cloth instead of punched plates, if desired. The entire screen is hung so that its inclination can be changed in a few minutes. The rate of travel is not over 3 ft. or 4 ft. per minute. There are a number of advantages in this type of screen over others; it requires almost no attention, has a large capacity and will screen finer clay because it is always clean.

It has been found quite successful in screening ore as well as clay.

Centrifugal Screens.—One of these devices is in use at a plant at Alfred, N. Y. It consists of a large funnel-shaped receptacle, 4 ft. high, with a diameter of 9 ft. at its upper or large end. The screen is supported on a central shaft with spiders and suitable bearings at top and bottom, and revolves with the shaft. Clay is fed into the center of the rapidly revolving cage, and upon striking the bottom of the cone begins to move up the sloping sides of the cone by centrifugal force. The fine clay passes through the perforations, while the coarse material is carried up and over the top, whence it is returned to the rolls for regrinding.

It is claimed that this screen is giving excellent satisfaction in practice.

Visit of Pittsburg Railway Club to Ellwood City Works National Tube Company.—On Friday evening, April 28th, at the monthly meeting of the Pittsburg Railway Club, Mr. F. N. Speller, metallurgical engineer of the National Tube Company, read a paper on locomotive boiler tubes. In the afternoon of the same day the National Tube Company had invited the Pittsburg Railway Club to visit the Ellwood City plant where the members of this club had an opportunity to observe the manufacture of Shelby cold drawn steel tubes and Shelby hot rolled seamless steel tubes. The party was taken to the works by special train and luncheon was served going, in two large baggage cars. All members of the Pittsburg Railway Club had been invited.

Improvements in Cotton Bleaching.*

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Bleaching is the whitening or removing of color from a substance, but in the textile industry the term is broadened to include the removal of all foreign bodies and coloring matter adhering to or included in the original fiber. In the case of fabrics it also covers the elimination of all materials, such as starch, oil and the like, added either intentionally or accidentally in the process of manufacture.

Cotton was used for the production of cloth thousands of years before the Christian era, and bleaching of the fabrics so made must have been more or less crudely performed even at that early age. While interesting, the scope of this paper will not permit the tracing of the early history of the art. Ancient processes will therefore be cited only as they may have some bearing upon modern practice.

An efficient process of bleaching requires a thorough understanding of the nature and chemical composition of the fiber and of the substances to be removed, as well as a knowledge of the effect produced by the different agents to be employed. It is a remarkable fact that very few complete studies of the process as a whole exist, and that the few careful investigations of different steps in the procedure have not received the attention and consideration they deserve at the hands of the practical manufacturers. There is no doubt that great progress will be made in the improvement of both processes and results by careful and scientific research in this field.

At the very outset of our study we are met by the fact that the chemistry of cellulose, which composes 83 per cent to 90 per cent of the crude cotton fiber, is still, after many years of investigation, far from being clearly defined. Two series of analyses conducted by the Department of Agriculture gave average results as follows:

FERTILIZING CONSTITUENTS.

Water	6.07
Ash	1.37
Nitrogen	0.34
Phosphoric acid	0.10
Potash	0.46
Soda	0.09
Lime	0.19
Magnesium	0.08
Ferric oxide	0.02
Sulphuric acid	0.6
Chlorine	0.07
Insoluble matter	0.05

PROXIMATE CONSTITUENTS.

Water	6.74
Ash	1.65
Protein	1.50
Cellulose	83.71
Nitrogen-free extract	5.79
Fat	0.61

ASH FROM COTTON.

Potassium carbonate	44.8
Potassium chloride	9.9
Potassium sulphate	9.3
Calcium phosphate	9.0
Calcium carbonate	10.6
Magnesium phosphate	8.4
Ferric oxide	3.0
Alumina and loss	5.0

Similarly, the precise change produced by alkalies and acids has not been determined and the extent to which such treatment may be carried without injuring the strength and lasting properties of the fabric is now rather a matter of observation and practical experience than an actual science.

For a knowledge of the impurities present in crude cotton we are indebted to a study of these bodies by E. Schenck. About 5 per cent of the original fiber is removed by boiling in soda ash for eight hours and may be thrown down from the resulting dark-brown liquor as a copious, light-brown precipitate by adding an excess of dilute sulphuric acid. If dried, it produces a brittle, horn-like mass which leaves upon incineration an ash varying from 2.3 per cent to 6.9 per cent.

From this residue Schenck succeeded in isolating substances which he classifies as follows:

1. A wax-like body which he named *cotton wax*. It is lighter than water, is soluble in ether and alcohol, but insoluble in water and only slightly dissolved by boiling dilute caustic soda.

2. A *fatty acid* which melts at 55.5° C. and may be either margaric or a mixture of stearic and palmitic acids.

3. Two nitrogenous, amorphous, brown *coloring matters*, one of which is easily soluble and the other sparingly soluble in alcohol.

4. *Pectic acid*, obtained in the form of a light-yellow amorphous substance resembling gum or gelatine. This body forms much the larger part of the precipitate, the other constituents being present only in extremely minute quantities.

5. Albuminous matter.

Many others have studied more or less thoroughly the nature of the substances extracted by water, ether or other solvents, as well as the nature of the ash present in the raw cottons, but without more definite results.

In cotton bleaching we are confronted with the problem of the removal of these various impurities and of the decolorizing or whitening of any yellow of brown tint still remaining. Broadly speaking, the process consists of boiling with alkili liquors to remove all impurities made soluble by such treatment and a subsequent oxidation for the removal of residual tint.

Cotton may be bleached in almost any form from loose or raw cotton to cotton piece goods, the difference in treatment required in each case being purely mechanical. As the bleaching of cotton piece goods far exceeds the importance of the bleaching of any other form of cotton, we will confine our attention to that form. We will also assume in the succeeding remarks a knowledge on the part of the reader of the terms and processes of practical bleaching.

The complete process used with slight modifications for the past 50 years may be outlined as follows:

1. Singeing.

2. *Gray Wash.*—Wet out in washing machine, allow to lie for 24 hours and wash.

3. *Lime Boil.*—Pass through milk of lime and boil for 12 hours under pressure.

4. *Lime Sour.*—Wash and pass through hydrochloric acid at 3° Twaddle, squeeze and allow to lie in sours for six to eight hours. Wash thoroughly.

5. *First Lye Boil.*—Boil six to eight hours with soda ash and resin soap. Wash in kier and through washer.

6. *Second Lye Boil.*—Boil in kier as for first boil, using soda ash and a small amount of caustic. Wash in kier and through washer.

7. *Chemic.*—Pass through hypochlorite of lime solution at ½° to 1° Twaddle. Allow to lie in chemic from six to eight hours.

8. *Sour.*—Wash and pass through sulphuric acid at 1° to 2° Twaddle.

9. *White Wash.*—Wash until all traces of acid are removed, open and dry. This process in practice will occupy from five to ten days.

The first step in the process of bleaching cloth for printing or other purposes requiring a smooth face is the operation of singeing, which has for its object the removal of all loose

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hairs from the surface of the cloth. This step is largely mechanical, but requires close attention to produce satisfactory results without damage to fiber or undue cost for gas or fuel. The most approved method, known as gas singeing, passes the fabric at speed several times through the flame of Bunsen burners extending the width of the cloth. Improvements in this step must come from proper regulations of the gas mixture and mechanical arrangement of rollers and speed of travel to produce the most efficient results.

The next operation is known as steeping, or gray washing, and consists in passing the goods through a washer or otherwise saturating, preferably with warm water, and allowing it to remain piled in a warm room from 18 to 24 hours. This treatment removes any bodies soluble in water and, as a result of fermentation, the starchy matters added as sizing.

Modern practice has made the mistake of considering this operation unnecessary and omitting it entirely even when the caustic boil is used. Lime probably removes the starchy matters completely, but it is extremely doubtful if more than a relatively small portion is extracted in the caustic boil. Steeping may be replaced by a malt treatment, using by preference one of the diastase preparations now on the market. The solution is prepared by using 15 lb. to 20 lb. of diastace or similar product to 100 gal. of water and warming to 140° Fahr. to 150° Fahr. The goods are wet out in this solution and the action allowed to continue for 15 to 30 minutes, then washed and passed to the next step in the process. Solubility of the starch may also be produced by wetting out with very dilute hydrochloric or sulphuric acid and piling for a short time. This process must be carried out with extreme care, owing to danger of the acid injuring the fiber.

Lime Boil.—The cloth is saturated with milk of lime of such a strength that it takes up about 4 per cent of its weight of CaO, run into one of the standard kiers and boiled with constant circulation of the liquor through the goods. This operation decomposes the fats and similar matter contained in the cloths and forms as a result insoluble lime soaps. These latter are broken up by the scouring which follows and completely removed by the subsequent operations. In addition, the other natural impurities of the raw cotton, as well as starch, are so changed that they either become soluble or are more easily acted upon by the chemicals employed in the succeeding processes.

The gray sours remove the excess of lime and other metallic oxides, if present, besides breaking up the insoluble soaps formed on the fiber.

The *lye boils* continue the reactions already begun in the lime boil and further saponify and remove any fatty acids resulting from the decomposition of the lime soaps in the previous steps.

Resin soap is employed by some bleachers and is supposed to remove certain indefinite constituents of the fiber which are said to have a deleterious effect on the whites of prints and are not fully removed by the other agents employed.

Bleaching or Chemicing.—The destruction of the coloring matter now left in the fabric is the object of this final step. Previous to the nineteenth century this was always accomplished by the exposure of the moist cloth to the action of air and sunlight, and extensive bleaching greens formed an important part of every establishment. The use of chlorine for this purpose was first proposed by Berthollet, about 1785, but its use even after the introduction of the hypochlorite of lime was very slow in becoming general. To-day, chlorine, either as calcium or sodium hypo, far exceeds all other agents in importance and in amount used. With proper regulation of strength of solutions and duration of action this bleaching agent may produce the most satisfactory whites without appreciable injurious action on the strength or other desirable properties of the fiber.

Large quantities of chlorine are annually produced electrolytically either as a gas to be absorbed by lime to form ordinary bleach or as hypochlorite of soda solution to be used direct. On a large scale the latter form cannot compete successfully with the cheaper and more common product, but it finds extensive use in laundries and in small establishments.

The sodium hypochlorite solution as made in this form of apparatus contains an excess of sodium chloride which causes the chlorine to be given off freely either to the fabric or the atmosphere. The solutions are, therefore, unstable and cannot be made in advance. The cost of production is also higher except in the case of cheap electric power and low freight rates for salt.

The much-claimed superiority of electrolytic bleach seems to have been based on the increased rate at which the chlorine was given off in the presence of the chloride and may be produced, if desired, in bleach from other sources.

The use of hypochlorite of soda made from bleaching powder and soda ash is highly to be recommended and in many cases will more than repay the increased cost. The following receipt for its manufacture is given by Thies and Herzig:

Two hundred and ninety kilograms of bleaching powder of 33 per cent available chlorine are ground to a paste with 1100 liters of water, and to this are added 175 kg of soda ash dissolved in 500 liters of water, and the resulting solution is immediately made up in 2000 liters. After stirring for one-half hour the mixture is allowed to settle over night and the clear liquor is decanted. The sediment is mixed with clear water, the clear liquor being decanted and this process repeated until in all 5000 liters of liquor are obtained, which should stand at from 6° to 6½° Twaddle. This solution is alkaline and is said to keep well.

In order to neutralize the free alkali and start the liberation of the hypochlorous acid, sulphuric or hydrochloric acid is added to the liquor just before using. Thies and Herzig recommend 6 grams to 7 grams of sulphuric acid of 60° Beaume for every 10 liters of the above bleaching liquor.

The kiers used in bleaching have had much to do with the progress of the art. Starting from open kettles heated by direct fire the natural tendency led to a large open kier, but heated by steam. A great improvement in this kier was the introduction of a central vomit pipe taking the liquor from the bottom of the kier and throwing it by means of a steam ejector over the goods at the top, thus producing a circulation through the goods. The next improvement was the introduction of the closed kier, by means of which the pressure and consequent temperature of boiling was increased.

The first distinct system was that of Barlow, whose kier was introduced about 1870. This system employs two kiers working in pairs and may use steam up to 40 lb. pressure. The liquor circulates from the bottom of one kier to the top of the other. In working with the Barlow system both kiers are filled with cloth as usual, the requisite quantity of liquor admitted and steam blown through both to expel the air through vent pipes. The kier is then closed and steam let into one only, so that the pressure at the top forces the liquor through suitable connection to the top of the other, where it is sprayed over the cloth.

When most of the liquor has been forced into the second kier the valves are changed so that the pressure is led to that kier, changing the circulation to the top of the first kier. This reversing process is continued until the goods are sufficiently boiled, when the kiers are blown out, washed and the process continued as usual.

The next great step in advance was the advent of the Mather kier, which was especially constructed by Messrs. Mather & Platt to carry out the process devised by Horace Hoechlin. In this system the lime boil is entirely dispensed with and this, as well as the two lye boils of the old process, are replaced by a single boil in caustic soda and resin soap. The kier used for this boil is a large horizontal cylinder with convexed ends, one of which may be removed to allow the entrance of the goods. Suitable means are provided for raising the door and for making the same steam-tight when closed. The pieces to be bleached are piled on wagons made of sheet iron with perforated

bottoms and running on tracks leading inside the kiers. When ready the wagons are run into the kier, the door is closed, the air driven out by steam and circulation started. While the kier is boiling, other wagons may be loaded and those already boiled unloaded, thus increasing greatly the production of a single kier.

The process aims to subject the fabric to the action of caustic liquor and steam at the same time and for this purpose the amount of solution used is small compared with the size of the kier. The circulation is maintained by means of a centrifugal pump, which takes the liquor from the space under the wagon and showers it over the top of the goods. It then percolates through the pieces and collects in the bottom of the kier, where it is heated by closed steam pipes and is again taken up by the pump, which maintains a continuous circulation. The kier may also be heated by the introduction of live steam, but the resulting dilution of the liquor is more objectionable in this than in other kiers.

Several ingenious continuous processes have been proposed for boiling cotton piece goods, but as yet no system of this character has succeeded in prolonging the treatment long enough to produce results comparable with those obtained by the more usual method.

The Thies keir and the process of which it forms a part are the result of long, careful research by Thies and Herzig. By designing the kier in such a manner that no steam comes in direct contact with the liquor and thus avoiding the action due to the oxygen contained in the air always present in steam, it is impossible to use strong caustic at a high pressure without fear of tendering the fabric. The latest form of kier consists of three parts—the kier proper, the vacuum boiler and the superheater. The vacuum boiler consists of an upright cylindrical vessel similar in size and shape to an ordinary kier, while the superheater is of the same height as the kier, but much smaller in diameter. The latter is provided with closed coils for heating. For the circulation and transfer of liquors use is made of a Grindle pump. The boiling consists of two parts and the process is described by the authors as follows:

The goods saturated with alkali (liquor blown off from the previous bowking) are packed into the kier and then by means of the Grindle pump the old kier liquor is drawn from the vacuum boiler (which is quite full to start with, and closed at the top) and sent in at the bottom of the kier until the latter is quite full, the air escaping through a blow-off valve at the top. As soon as the kier is full the air valve is closed.

By pumping the liquor from the vacuum boiler into the kier a vacuum equal to about 10 lb. is produced in the former vessel. A pipe provided with a throttle valve connects the top of the kier with the top of the vacuum boiler. This valve is now slightly opened, so that a circulation of the liquor takes place, while any air contained in the kier rises with the liquor, passes through the throttle valve into the vacuum boiler, where the liquor falls to the bottom and expands into the vacuum.

The pressure in the kier is now gradually increased to three atmospheres by choking the valve, when any air remaining in the goods is mechanically dissolved out of the goods (air being more soluble in water under pressure) and transferred to the vacuum boiler. After the air in the vacuum boiler has been expelled, steam is turned on in the superheater and the bowking continued for two hours at 45 lb. pressure.

The pump is now stopped, the steam turned off and the liquor blown off under its own pressure. Fresh lye, consisting of caustic soda at 6° to 7½° Twaddle and the resin soap, is now run into the kier (which is free from air), the pump set in motion again, steam turned on in the superheater and the bowking proper continued for six hours at 45 lb. pressure.

As free hypochlorous acid is much more efficient as a bleaching agent than any of its salts we find many attempts to make use of this property.

Mather and Thompson pass the cloth continuously through closed chambers in which the pieces are run through an atmosphere of CO_2 after being saturated with the ordinary bleaching solution. Acetic acid vapors have also been used in the same manner. The addition of small amounts of sulphuric or hypochloric acid or of acetic or formic acid to the chemic solution will produce similar results, and the improved action of the liquors will be at once apparent.

Chlorine in cotton bleaching may be replaced by other oxidizing agents and will no doubt be superseded at some later date by some agent acting less injuriously on the fiber itself. Such a body is found in sodium or hydrogen peroxides and it is only the higher cost which prevents its more general use. For cotton and silk goods, laces and other special fabrics, it finds extensive application.

Drawing freely from all published information, especially the results of the researches of Koechlin, Thies and Scheurer, the following process has been evolved as a working compromise of the theoretical and practical:

The goods are marked and sewed as usual and piled on trucks. They are then run through the gas singer and directly through a lukewarm solution of diastafor or other diastase and into a continuous piling chute—a more or less complicated inclined trough so arranged that while cloth is entering at the upper end and leaving at the lower a large amount of slake cloth fills the length of the incline and allows time for the action to take place. To increase the time of action, several of the chutes may be arranged in series, the cloth passing over a reel between each.

From this step the goods pass directly to a small washer and are then squeezed and piled in a bin. As wanted they are drawn through a saturating machine, where they are impregnated with caustic soda at 3° Twaddle, containing 0.5 per cent sodium bisulphite solutions and packed directly in the boiling kier.

This kier should be of the vertical pressure type with outside vomit pipes and pump circulation. The perforated false bottom should be conical in shape and so arranged as to give a much larger space at the bottom than is usually allowed, and should be provided with both closed and open steam pipes. The closed coil should be sufficient to maintain the kier at boiling during circulation after that temperature has been reached, and must be provided with a suitable trap to remove the hot water and return it to the hot well or feed-water heater.

The goods are evenly plaited and tramped in the kiers by boys in the usual manner to within 2½ ft. or 3 ft. of the top and anchored with nets or chains.

The air is replaced by admitting steam and a charge composed of 3° Twaddle caustic containing 3 per cent of resin soap run in to cover the goods. The kier is brought to a boil with the open coil at a pressure of 15 lb., a small vent pipe being open to allow the escape of air.

When the goods are boiling the open pipe is nearly closed and the boiling is continued for six to eight hours with the use of the closed coil as far as possible.

The charge is washed in the kier, care being taken to prevent access of air to the hot fabric. The pieces are then run out of the kier through a washer, squeezed and returned to another kier, where the boil is repeated, but with the omission of the resin soap.

Experience seems to indicate that two boils are necessary to produce a thorough bleaching without over treatment in the subsequent steps.

The goods are next run through the chemic and piled in open bins for two to six hours. The best results are produced at the greatest economy by the use of hypochlorite of soda prepared as already described, but with experience and care very satisfactory results are obtained by the use of hypochlorite of lime solution at ½° to 3° Twaddle.

Continuous processes which subject the fabric for a short period to a very strong chlorine solution or which sour the cloth after chemicing without previous washing are necessarily injurious to the fiber and unsatisfactory in other respects.

The pieces after lying exposed to the air for a suitable length of time are washed, soured through sulphuric at 1° to 2° Twaddle, washed twice, and are then ready for opening, drying and finishing as desired.

Bleaching to-day is not an exact science and depends for success upon experience combined with close observation and understanding of the various processes. If the heads of the industry in this country could be brought together for an interchange of experience and frank discussion of the most promising lines of research, a great advance would be produced in the art, but in any case we may expect sooner or later more economical and efficient processes to replace many of those now in use.

Boston, Mass.

American Foundrymen's Convention at Pittsburgh.

The annual convention of the American Foundrymen's Association, the American Brass Founders' Association, and the Associated Foundry Foremen will be held in Pittsburgh from May 23 to 26. More than 3000 delegates are expected to attend the convention.

At the same time the great exposition of the Foundry & Machine Exhibition Company will be held. The exhibits will fill more than 40,000 square feet of area.

The Pittsburgh committee in charge of the preparation for the convention has been very active. It consists of Mr. Joseph T. Speer, chairman; Mr. F. W. Zimmers, secretary; Mr. J. S. Seaman, chairman of the Finance Committee; Mr. H. E. Field, chairman of the Convention Committee; Mr. W. A. Bole, chairman of the Excursion Committee; Mr. George P. Bassett, Jr., chairman of the Ladies' Entertainment Committee; Mr. E. D. Frohman, chairman of the Ball Game Committee; Mr. Eliot A. Kebler, chairman of the Reception Committee, and Mr. W. B. Robinson, chairman of the Press Committee.

On Monday, May 22, registration of delegates will be started at the exhibition buildings. The formal opening will be on Tuesday, May 23, and at the joint meeting papers on production cost, economic foundry insurance, and uniform contracts will be presented.

In the afternoon of the same day there will be sessions of the American Foundrymen's Association (papers on unloading methods, on the use of borings in the cupola, on the effect of alloys in cast iron, and on permanent molds) and the American Brass Founders' Association (papers on vanadium in non-ferrous metallurgy, on determinations of nickel in bronzes, on pouring high-grade bronzes, etc.

A joint meeting will be held on Wednesday morning, May 24, with papers on rotary blowers, on foundry construction, on heating and lighting systems, on pattern making, and on molding machines, while separate meetings will be held in the afternoon by the American Brass Founders' Association and the American Foundrymen's Association.

On Thursday morning, May 25, another joint session will be held with papers on molding sand, on the use of alloys, on shot in castings, etc., while for the afternoon a great excursion to foundries and steel works around Pittsburgh has been arranged.

On Friday morning, May 26, separate sessions will be held by the different associations, with election of officers. In the evening of Friday the large official subscription dinner will be held.

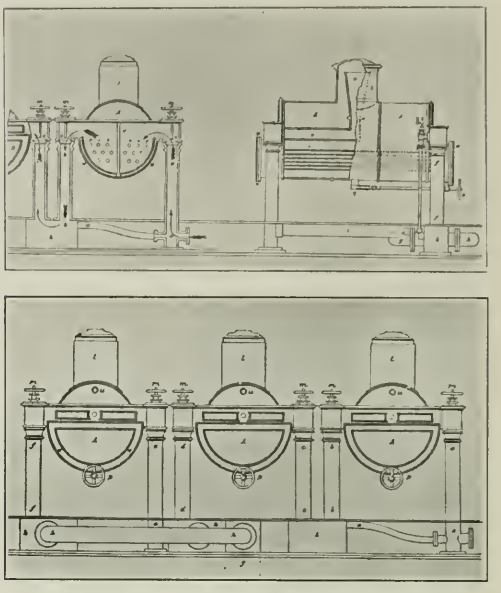
Evaporators and Vacuum Pans.

By B. Viola, M.E.,
Member American Society of Mechanical Engineers.

(*Continued from page 209.*)

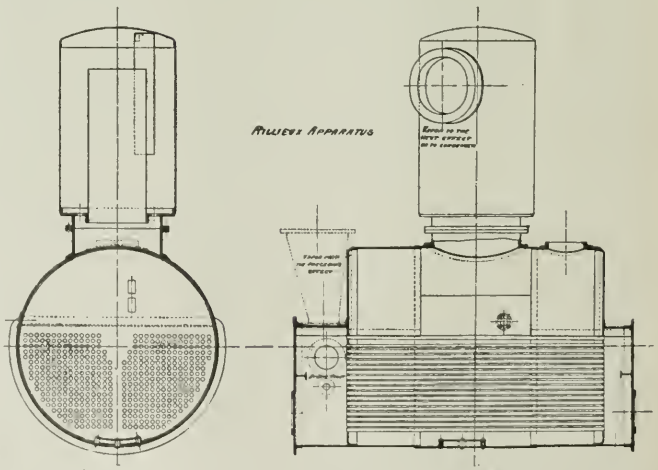
Evaporators with Horizontal Heating Pipes.

The originator of the present system of evaporation was Norbert Rillieux, who in Paris in the year 1830 first proposed



FIGS. 11, 12 AND 13.—ORIGINAL CONSTRUCTION OF RILLIEUX EVAPORATOR.

the construction of the multiple effect evaporator. It took, however, many years before the first one could be built and used by Rillieux himself in Louisiana in the year 1843.



FIGS. 14 AND 15.—MODERN CONSTRUCTION OF RILLIEUX EVAPORATOR.

According to the report to the Thirtieth Congress in 1848 this was a triple effect with wrought-iron heating tubes and

is shown in his original construction in Figs. 11, 12 and 13. It was composed of three cylindrical vessels made of sheet-iron plates 10 ft. long and 3 ft. 6 in. in diameter, made parallel and supported by cast-iron columns which are placed at each end.

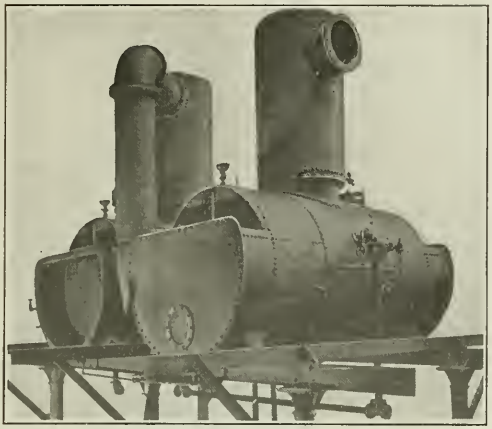


FIG. 16.—DOUBLE-EFFECT RILLIEUX EVAPORATOR.

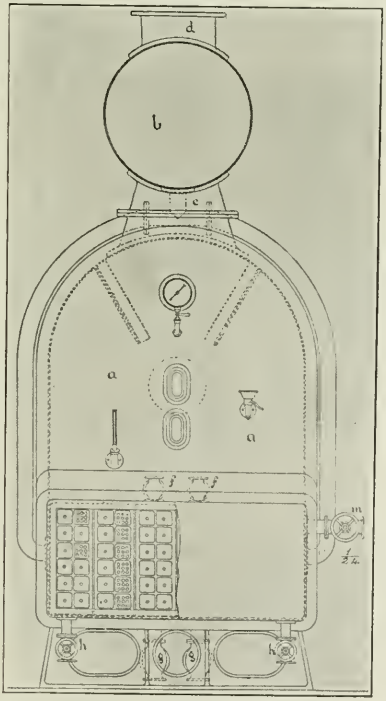


FIG. 17.—JELINEK EVAPORATOR—CROSS-SECTION.

The steam or vapor circulates in the interior and passes from one vessel to the other by means of a system of valves and pipes and each vessel is provided with a dome in which a cone-shaped connection is inserted for taking up the vapors of the juice and leading it to the next effect. The vapors from the last vessel pass to a condenser, where they are condensed by an injection of water. The vacuum is maintained by means of a large vacuum pump.

A modern construction of a Rillieux evaporator is shown in section in Figs. 14 and 15. Fig. 16 shows a complete double effect manufactured in this country. Rillieux's apparatus was introduced in Europe with the help of his foreman, Brami André, and in 1849 was constructed by Tischbein with horizontal, and later by Robert, in his sugar factory, with vertical heating tubes.

All the evaporators had disadvantages in that the steam chamber and the space for boiling were very small and naturally the surface of the liquid also, especially in the vertical constructions. Under these conditions, when the liquids were in the boiling state, the vapors carried away great quantities of small drops, thus causing immense loss.

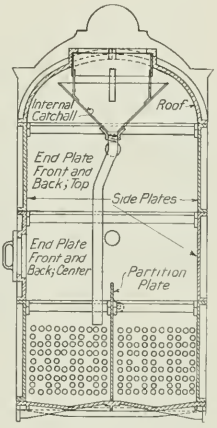


FIG. 19.—SWENSON EVAPORATOR.

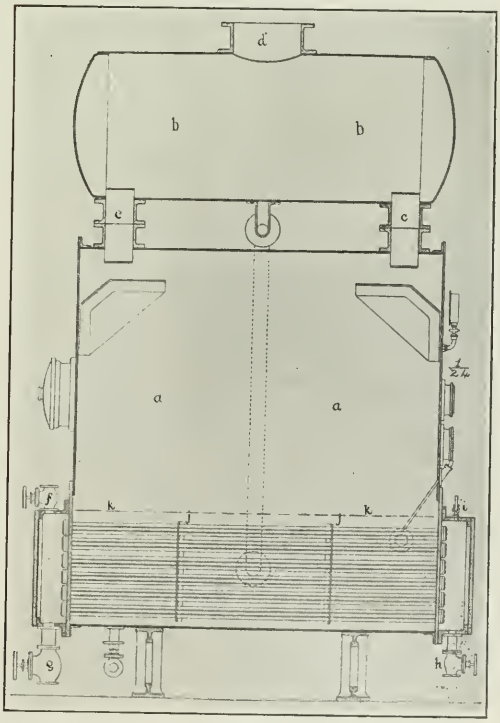


FIG. 18.—JELINEK EVAPORATOR—LONGITUDINAL SECTION.

The first improvement to avoid this loss was made by Hugo Jelinek. He constructed his evaporator with a rectangular

bottom and haf-cylindrical top, filling the lower part with a large quantity of heating tubes and making the apparatus very high on account of the large space above the boiling liquid. The liquid was always kept low, hardly covering the heating pipes, and in this way getting a high efficiency, Figs. 17 and 18.

We can see on both sides the so-called drop catchers which prevented the drops raised by the dome vapors to pass in the T on top of the apparatus. These catchers covered the vapor outlets and the drops raised by the vapors were knocked against them and fell back, the vapors passing out through the two outlets into the next effect or condenser.

A very efficient construction of the Rillieux type is the Swenson evaporator built in the United States and shown in section in Fig. 19. Fig. 20 shows how the heating tubes are made tight to the shells and Fig. 21 the disposition of a quadruple effect.

Film Evaporators.

As mentioned before steam bubbles in an evaporator rise to the surface with greater difficulty if a large body of liquor covers the heating pipes and naturally the boiling point is much higher in the bottom than at the top. To make the evaporation more efficient different evaporators were con-

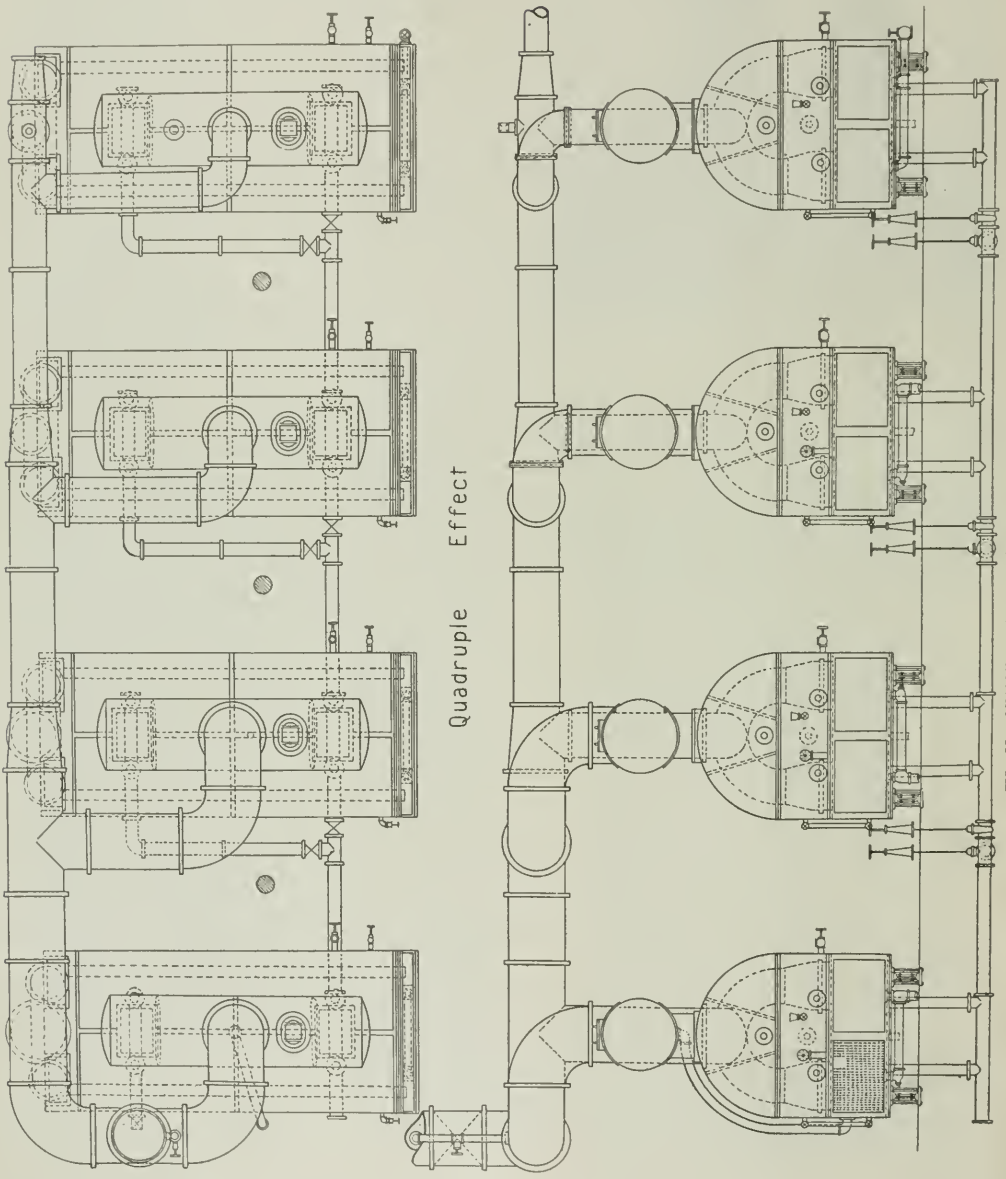


FIG. 21.—QUADRUPLE-EFFECT SWENSON EVAPORATOR.

structed to let the liquors run on the outer surface or the inner surface of the heating tubes in thin films or fine sprays.

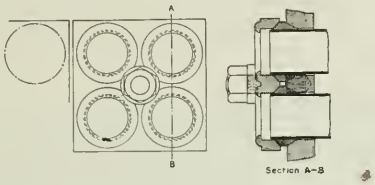


FIG. 20.—ARRANGEMENT OF HEATING TUBES.

We can classify them:

a. Film evaporators with horizontal heating tubes.

b. Film evaporators with vertical heating tubes.

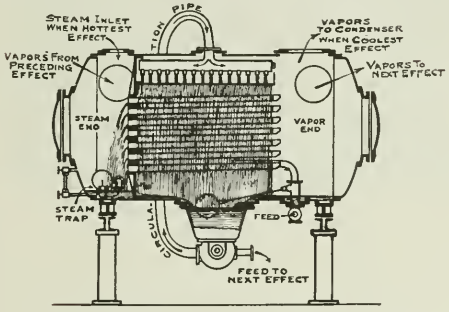


FIG. 22.—FILM EVAPORATOR WITH HORIZONTAL TUBES.

I will describe shortly those most in use at the present time having high efficiencies.

The Lillie Evaporator.

Fig. 22 shows an evaporator with horizontal tubes and mechanical circulation in longitudinal section. The evaporating tubes incline slightly downward toward the steam end and open through the heavy tube plate partition, in which they are firmly expanded (without annealing) and by which they are supported.

The other ends of the tubes are closed, save for a small air vent, Fig. 23, in each. They are not fastened or supported in any way, and the tubes are quite free to expand or contract independently of the shell of the effect.

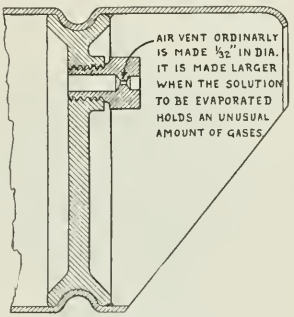

AIR VENT ORDINARLY IS MADE 1/32" IN DIA. IT IS MADE LARGER WHEN THE SOLUTION TO BE EVAPORATED HOLDS AN UNUSUAL AMOUNT OF GASES.

FIG. 23.—AIR VENT OF A LILLIE EVAPORATING TUBE.

On the under side is a specially designed centrifugal circulating pump located midway between the ends of the evaporator and bolted to a well, which, however, is sometimes omitted, in which case the liquor rises sufficiently in the body to float the feed-valve ball float which regulates the feed into the body.

The condensation of the steam inside the tubes flows back into the steam end and thence through a steam trap into the steam end of the next cooler body and finally from the coolest body to the atmosphere, in the case of a multiple effect.

The circulation of the solution over the tubes is a deluging shower, maintained by the centrifugal pump. The circulation is independent of ebullition and there is no depth of solution on the tubes through which vapors must force their escape, as in all submerged-tube evaporators.

The small quantity of liquor in the body is indicated. It is comprised of that in the well beneath the effect plus the liquor flowing over the tubes.

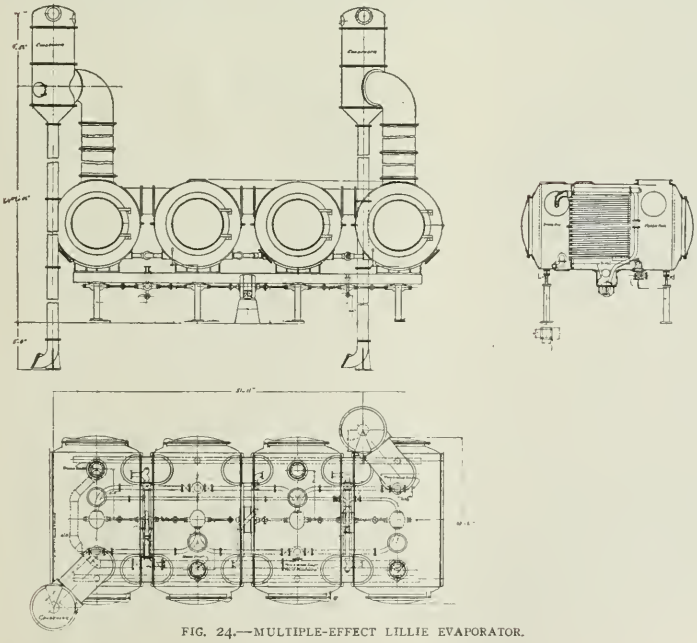


FIG. 24.—MULTIPLE-EFFECT LILLIE EVAPORATOR.

The perforated distributing plate above the tubes is formed in sections for convenience in handling. It has one row of holes 3/8 in. or more in diameter above each vertical row of tubes. There is a 1-in. space between the distributing plate and the tube plate, through which there is a heavy flow of solution upon the surface of the tube plate, which prevents matter caking on the tube plate.

There is also a 1-in. space on each side of the tube plate. The flow of the solution over the side edges of the plates carries with it the particles of scale that may come up with the liquor and prevents them from clogging the perforated plate. These matters pass away when the liquor is discharged from the body and in the case of a multiple effect, finally leaving the apparatus from the last effect.

The feed to the effect is automatically regulated by the feed-float valve to correspond with what passes away from the effect and consequently the liquor level is held practically constant. This is the usual method of arranging the vents for the escape of incondensable gases from the tubes. They are, however, differently arranged to meet different conditions.

These vents should occasionally be tried with a fine wire to see if they are open. A closed vent stops evaporation in its tube.

When combined to form multiple effects (Fig. 24) bodies are placed in line side by side and turned alternately end for end, so that the vapor end of a body is next to the steam end of each adjacent body; this permits short vapor connections between the bodies and the peculiar valve system employed for reversing vapors. The circulating pumps come in line and one shaft is employed common to all the pumps.

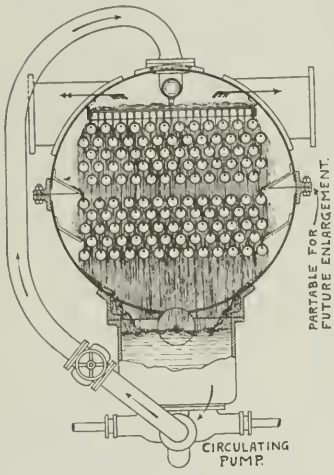


FIG. 25.—LILLIE EVAPORATOR BUILT FOR FUTURE ENLARGEMENT.

In addition to the circulating pumps the shaft usually carries a centrifugal pump for discharging the thick liquor and another for the condensation. All the latter comes away from the coolest body through this one pump. This shaft is driven by a motor or by a belt from an engine.

The multiple effects are arranged either for not reversing the direction of vapors through the system (the usual manner of working multiple effects) or to permit of reversing the direction of the vapors or heat through the system, which is strictly characteristic of the Lillie multiple effect.

Its operation is automatic. The feed of liquor into the first effect and from effect to effect and away from the last effect is automatically regulated by the feed-float valves, no matter what may be the number of effects, with the exception of the regulation of the density of the product by means of the valve in the discharge pipe for the latter. The water of condensation is automatically trapped from effect to effect and all passes away from the coolest effect through the centrifugal condensation pump.

A valuable feature of the Lillie evaporator is its capacity for enlargement, when bodies are constructed with that end in view. It is accomplished by parting the two semi-cylindrical halves of which an effect is composed when constructed for future enlargement and inserting between them a section of tubes carrying the tube surface which it is desired to add (Figs. 25 and 26).

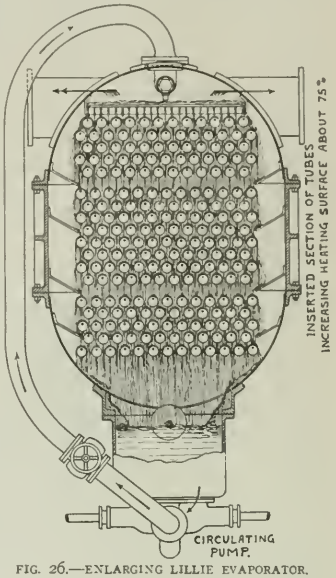


FIG. 26.—ENLARGING LILLIE EVAPORATOR.

The general arrangement for the reversal of vapors of the Lillie multiple effect or cutting out bodies in times of emergency are simple and almost automatically effected. By reversing the direction of the vapors scale does not form on the evaporating tubes, or if it does form it tends to crack and fall off from the tubes on account of the difference in expansion between the scale and the metal of the tubes under the changing temperatures. The time required for reversals is from 5 minutes to 15 minutes each.

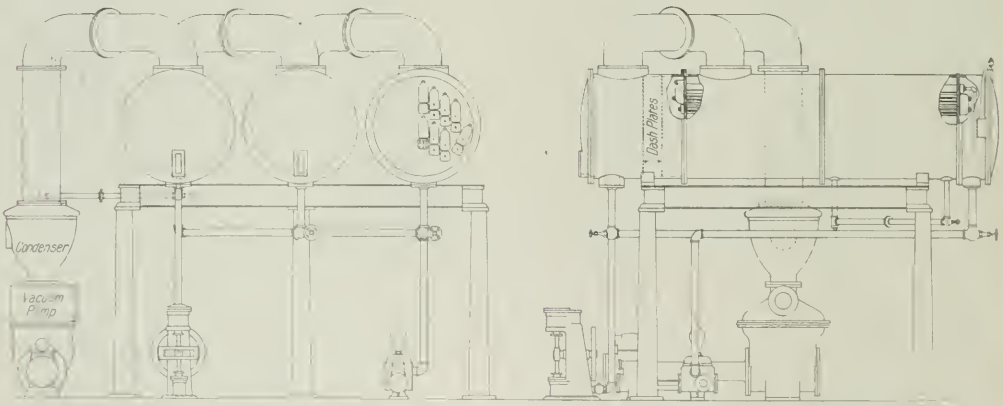


FIG. 27.—YARYAN EVAPORATOR.

The course of the liquor being concentrated is, as a rule, reversible. That is, it may be sent through the effects in succession in either direction, no matter what may be the direction of the vapors through the series. This permits taking the diluted solution into the coolest effect and delivering it concentrated from the hottest effect in cases when it is desirable to do so (when a liquid is to be brought to specially high densities).

Any effect can be cut out of the multiple effect and a quadruple effect can be arranged to run as two double effects.

The efficiency of the heating surface of the Lillie evaporator according to practical experience varies with conditions, but taking hot clarified cane juice and concentrating it to, say, 30 deg. Beaumé, it can be assumed that an evaporation of about 39 lb. per square foot in one body of a multiple effect, working with, say, 5-lb. pressure of steam to the first effect and an average vacuum in the coolest effect. This would be equivalent to about 10 lb. evaporation per square foot of total heating surface in the quadruple effect and about 13 lb. evaporation per square foot of total heating surface in a triple effect. The Lillie evaporator is used for water distilling and many plants are equipped with it.

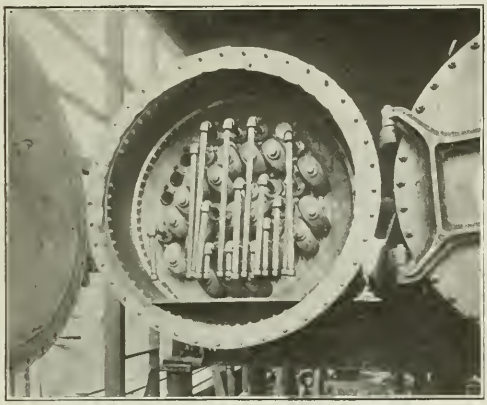


FIG. 28.—YARYAN EVAPORATOR. FEED CHAMBER.

The Yaryan Evaporator.

Under this name (Figs. 27 and 28) it is very well known and is now one of the oldest and very much improved evaporator systems with high efficiency. It consists of horizontal, cylindrical bodies with swinging doors at both ends and tubes packed into the tube sheets by return bends and packing glands, which makes it very easy to give the evaporating tubes a thorough cleaning, if necessary. Inside of these doors, as shown in Fig. 28, is a chamber in the front or liquor inlet side, and another chamber at the end or vapor side, where liquors and vapors leave the tubes.

The Yaryan evaporator is a combination of groups of evaporating tubes, each group working for itself. Between the two shells is the steam chamber and the evaporating tubes are there surrounded and heated by the steam.

Since the feed chamber (Fig. 28) for the vertical inlet pipes through which the liquor enters a pipe of larger diameter is surrounded by steam, the liquor begins to boil immediately and rushes with great velocity until it reaches the return bend (Fig. 27) and enters the pipe group with small diameters. By the rapid evaporation in the tubes great quantities of steam are formed which by their expanding force rush the liquid in broken-up streams of films through the pipes.

After these films pass the second return bend in the front they empty their contents with great force against the baffle plates, where the liquid falls to the bottom and continues its way to the next effect or the receiving tank, while the vapors pass over into the steam chamber of the next effect or to the condenser.

The Yaryan evaporator can be used to great advantage for water-distilling purposes as it concentrates any liquid that has a tendency to foam and also, on account of the ease with which the tubes can be removed and cleaned, it can be advantageously used for any liquid that scales the heating surface.

(*To be continued.*)

New York Meeting of the American Electrochemical Society

The nineteenth general meeting of the American Electrochemical Society, held in New York City on April 6, 7 and 8, was as successful and enjoyable as any that preceded it. For the first time the splendid new Chemists' Building was the gracious host of a national chemical society convention.

The attendance was good, more than 350 members and guests having registered. All the professional sessions, which were held in the spacious Rumford Hall of the Chemists' Building, were well attended. The papers presented were followed with unflagging interest and the discussions were lively and to the point.

No less interest was shown in the excursions which had been arranged for the three afternoons, including two of the largest and best-equipped copper refineries. At the works of the Nichols Copper Company, on Long Island, on Thursday afternoon, the smelting of copper ores and the bessemerizing of copper matte were shown, this being now the largest copper refinery in the world. Of even greater interest was the extended excursion on Saturday afternoon to the Perth Amboy works of the American Smelting & Refining Company, where the processes of electrolytic copper refining and of electrolytic silver refining were also shown, while in the purely metallurgical plant the latest departure of metallurgical practice—the bessemerizing of leady copper mattes in basic-lined converters—attracted naturally the closest attention.

For Friday afternoon a number of optional visits had been arranged, including an excursion to the new works of the Hanson & VanWinkle Company, of Newark, N. J., where the visitors were very cordially received and entertained, and the thoroughly modern and up-to-date demonstrating electroplating plant was shown which is used chiefly for demonstrating the company's mechanical electroplating apparatus for plating large quantities of work in bulk.

Another party visited the new interesting electric-furnace laboratory of Dr. F. J. Maywald, where both direct current and alternating current is available for experimental work with various types of electric furnaces, and the exceedingly well-equipped plant of the Moss Photo-Engraving Company, where the latest methods of photo-engraving and electrotyping could be watched in commercial practice.

Other parties visited the New York Testing Laboratories, which have already been described in this journal, and the laboratories of Columbia University and of the College of the City of New York.

A subscription dinner "en famille" on Thursday night was very enjoyable. Mr. Maximilian Toch distinguished himself as a most resourceful and witty toastmaster. On this occasion the secretary of the New York Section of the society, Mr. H. B. Coho, presented on behalf of the section a watch fob to Mr. Alois von Isakovics in recognition of his seven years' service as secretary-treasurer of the section and as chairman of the membership committee of the society. The fob was a copy—the first one struck off—of the new enameled gold emblem of the society. Other after-dinner speakers were Professor Walker, Dr. Whitney, Professor Richards, Mr. Sperry, Dr. Baekeland and Professor Bancroft. The dinner, which was a very happy, harmonious and informal affair, was attended by a large number of ladies.

Not less enjoyable, though in a different way, was the smoker on Saturday night. Mr. H. B. Coho, as master of the revels, was simply grand. He was ably assisted by Mr. Frank A. Pattison, whose acting was clear-cut art, and by Mr. J. M. Muir,

who managed the affair with great success from the outside. There were songs, presents, speeches, a raid, a newspaper, a love story, and refreshments.

Business Meeting.

The annual business meeting of the society was held on Thursday morning, when the reports of the secretary and treasurer were presented.

In the report of the board of directors, presented by the secretary, Prof. J. W. Richards, it was stated that the increase of the number of members was quite satisfactory, being 145 net during the year. The number of members is now 1294. The financial status of the society during 1910 is also very satisfactory. The excess of receipts above expenditures for the year was $1,176.40, from which the permanent investment fund was increased by the purchase of another $1,000 bond.

The result of the annual election of officers was announced as follows: Dr. W. R. Whitney, chief of the research laboratory of the General Electric Company, is the new president of the society. Mr. F. A. J. FitzGerald, Dr. E. F. Smith, and Mr. A. von Isakovics were elected vice-presidents. Dr. E. G. Acheson, Dr. W. D. Bancroft, and Mr. Carl Hering were elected managers. Dr. Joseph W. Richards was re-elected secretary and Mr. Pedro G. Salom treasurer.

Dr. Willis Rodney Whitney, the new president, was born in Jamestown, N. Y., on Aug. 22, 1868. He graduated from the Massachusetts Institute of Technology with the degree of B.S. in 1890 and obtained the degree of Ph.D. at the University of Leipzig in 1896. He was for a time instructor and associate professor of chemistry at the Massachusetts Institute of Technology. In 1904 he accepted the directorship of the research laboratory of the General Electric Company. In this position he has been directing an enormous amount of research work, two brilliant results of which may only be mentioned—the graphitized-carbon filament and the ductile tungsten filament.

Dr. Whitney made a short address, starting with a poem by Browning and ending with a few impressive and characteristic words on universal optimism. Optimism is a wonderful tool in bringing about results. Optimism leads a man to try things. About the carbon ion, if we can get carbon to go as ion into solution, we can make a carbon cell which would revolutionize all methods of generating electrical energy. The ordinary man will say this would be such an enormous result, hence it will probably not work. The optimist goes ahead and tries it. He knows that he has an even chance all the time and he concludes that he had better take the chance.

In the following we give a concise report of all the papers presented at the meeting.

Dry Cell Testing.

Instead of the report of the committee on standard tests for dry cells, which was not yet ready, a paper was presented by Mr. W. B. Pritz, of the National Carbon Company, of Cleveland, Ohio, on dry-cell testing. The author pointed out the nonuniformity in the tests of dry cells now being employed and emphasized that there is in no sense a relationship between service and short-circuit current. It is true that the amperage reading of a cell, coupled with a familiarity with the particular brand, does serve as a good indication of the age of the cell or the presence of any serious defect; however, the customer who judges solely from the short-circuit current is very apt to obtain very inferior quality, and still dealers state that at least 90 per cent of the customers who buy cells off the shelves demand that they be so read and in most cases select that brand giving the higher current.

The author emphasized the need for informing the customer and for standardizing the methods of carrying on dry-cell tests. In general (other than for purposes of research) there are but two reasons for desiring a test upon dry cells: 1. To ascertain what life may be obtained from a brand of cells in a certain service. 2. To ascertain which one of several brands will give the longest life in that particular service.

With the former object in view the knowledge is best obtained by actual use of the cells in connection with the appliance. The great majority of tests are carried on, however, with the second object in view, viz., the comparison of two or more brands of cells for use in a particular service. Where the amount of testing is large it is impossible, even where it is expedient, to use the actual appliances for testing cells, and it becomes necessary to devise testing methods and appliances such that results obtained therefrom shall be comparable to the results obtained from the cells when placed in actual service.

It is even questionable whether it is wise to go further and make the operating conditions not only comparable, but as similar as may be to the operating conditions of the service for

W. R. WHITNEY,
PRESIDENT AMERICAN ELECTROCHEMICAL SOCIETY.

which the test is intended since it is not certain that results of greater meaning are obtained from such strict adherence to service conditions.

In general Mr. Pritz advises against test of various brands of cells by connecting them in series and discharging them simultaneously. There are occasions, however, when this is the only method to employ. Especially is this true when test conditions cannot be held constant for the separate testing of various brands. By test conditions are meant length of contact, strength of current and temperature of the battery. It is much preferable, for instance, to use the series method of test upon an automobile than to test the various brands upon different machines or upon the same machine at different times.

Mr. Pritz then discussed the effect of temperature upon dry-cell testing. This effect is three-fold. Firstly, temperature affects the initial current. When cells have been standing in a very cold place they may give such a low amperage as to be unsalable, but when brought to normal temperature the initial current is quite satisfactory.

Secondly, the influence of temperature upon service is greater than is generally supposed. Where comparable results are desired it is of extreme importance that the two batteries be kept at the same temperature. The effect of temperature also varies with the nature of the service, high temperature giving longer service where the drain is heavy, while low temperatures are favorable for light-drain tests lasting over a considerable period.

Thirdly, of prime importance in dry-cell testing is a knowledge of the effect of temperature upon the rate of deterioration of cells when left on open circuit. To determine just what this

effect might be, a number of cells were stored at seven different temperatures. The following table gives the initial currents and the drop in amperage after 10 weeks, expressed as a percentage of the initial values.

Temperature.	Initial Amperes.	Per Cent Drop in Amperes in 10 Weeks.
5 deg. C.	18.1	4.4
25 deg. C.	22.0	10.0
35 deg. C.	21.0	19.0
45 deg. C.	22.8	25.0
55 deg. C.	23.0	52.0
65 deg. C.	20.5	71.0
75 deg. C.	21.0	98.0

The greatly increased rate of deterioration on open circuit at high temperatures no doubt accounts for the poor service obtainable at such temperatures when long periods of time are necessary for its completion. The cells should be stored in a dry, cool place, and not in the corner behind the stove.

The rating of dry cells was next discussed by Mr. Pritz. In regard to the terms in which the results of dry-cell tests should be reported there is some difference of opinion, some authorities contending that the ratings should be expressed as the number of amp-hours given by a cell under specified conditions to a certain working voltage value. Others claim that more practical meaning is attached to a statement of the length of time during which the cell is able to maintain its working voltage above the specified limiting value.

It is the opinion of the author's company that that method of rating should be used which gives to the consumer the exact information which he desires, hence they favor the rating of cells by the length of service of a given kind which they are capable of giving.

The consumer is interested in the length of service which he is able to obtain from a battery and not in the amount of energy given. At first thought it might appear that the amp-hour capacity of a dry cell bears such a relation to length of service that either method of rating would give the same information. Such is not the case, however. Differences in discharge rate cause vast differences in the number of amp-hours obtainable from the cell. For research purposes the watt-hour or amp-hour output of a cell is very useful, but as a practical rating for dry cells it is not satisfactory, especially from the standpoint of the consumer.

Mr. Pritz finally urges the adoption of standard tests and recommends the following two tests for telephone service and gas-engine ignition service respectively:

For telephone service he subjects three cells in series to a discharge through 20 ohms resistance for a period of two minutes each hour, during 24 hours per day and seven days per week, until the working voltage reaches the limiting value of 2.8 volts, the results being reported as the number of hours' service to this cut-off point.

For gas-engine ignition service a battery of six cells is connected in series with a 16-ohm coil, which permits an average drain on the battery throughout its life of approximately 0.5 amp. Readings of the working voltage are taken at intervals throughout the test, the most important reading, however, being the impulse of current which the battery is able to force through a ½-ohm coil connected in series with the ammeter and in parallel with the 16-ohm resistance. When this value at the end of a period of contact falls below 4 amp the battery is considered unfit for service and discarded. The test batteries are discharged for two hours each day, one hour in the morning and one in the afternoon. As a test for lighter service the battery may be subjected to two drain periods of one-half hour each.

Since it is often the case that quite an interval of time may elapse between the manufacture and installation of a cell in service, a knowledge of what may be expected during this period becomes of no little importance, hence an open-circuit or "shelf-life" test is quite essential.

The method for determining shelf life generally consists of reading the initial voltage and short-circuit current of a representative sample of cells, followed by current readings at intervals of one or two months, depending upon the nature of the cell in question. The ammeter for reading short-circuit current should be dead-beat, and with its leads should have a resistance of 0.01 ohm. Two 30-in. lengths of No. 12 lamp cord make very convenient leads.

The results are merely indicative of internal resistance and bear no definite relation to the service which the cells may give. However, this information, coupled with familiarity with a brand of cells, becomes a very good indication of its quality. It also serves to indicate any serious defects of manufacture.

The cells are kept on the shelf until the short-circuit current has fallen below 10 amp. This point is arbitrarily chosen, as it represents a point below which it would be difficult to market the cell. For practical purposes the results are expressed as the number of months during which the short-circuit current remains above this cut-off point.

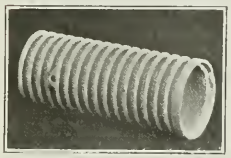


FIG. 1.—ELECTRIC FURNACE CORE OF ALUNDUM.

Much more meaning, however, is attached to the rate at which the current falls, generally reported as the drop in amperage expressed as a percentage of the initial amperage. This is especially true when investigation of the quality of cells is the object. For practical purposes, however, the first rating given, i.e., months to 10 amp, is perhaps preferable.

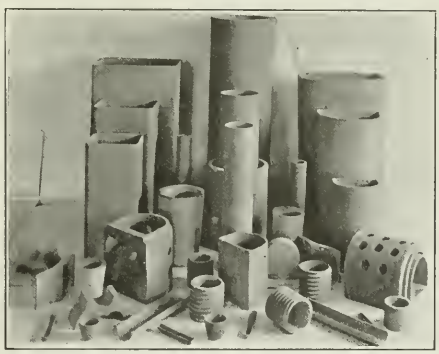


FIG. 2.—MUFFLES AND TUBES OF ALUNDUM.

The above paper having been printed in advance was read by title. There was no discussion.

Alundum Refractories.

Mr. L. E. SAUNDERS, of the Norton Company, of Niagara Falls, then presented a paper on electrically fused alumina (alundum) as a refractory. Alundum was primarily developed for abrasive purposes, but its heat-resisting properties were so often urged that it led to its introduction into this field.

There are two forms of alundum of interest in this respect: First, a white crystalline product, having less than 1 per cent of impurities, and, second, a reddish-brown vitreous material having from 6 per cent to 8 per cent of impurities, these consisting in both cases of oxides of iron, titanium and silicon. They are both electric furnace products from the mineral bauxite, having undergone different degrees of purification. The purer white product has a melting point between 2050° C. and 2100° C.; the brown product not more than 50° C. lower. The coefficient of expansion is linear and is, for the white product, 0.0000078 and for the brown material 0.0000085. The

thermal conductivity is about twice that of Marquart porcelain and three to four times that of most fireclay. The hardness in Mohr's scale is above nine, but lower than ten. The specific gravity in both products varies between 3.93 and 4.00. Both kinds of product are quite unattacked by aqueous acids and alkalies and very superficially by fused alkali carbonates. Basic or acid slags dissolve it with some difficulty, acid slags having a readier action.

Some attempts at molding it from liquid condition have met with partial success, although the difficulties involved are so great that only very simple and crude forms have been made.

FIG. 3.—ALUNDUM FILTER.

The method of application which has been found easiest consists in mingling the crushed grains (the purer form is mostly used) with a binding agent of a refractory and ceramic nature, molding the desired pieces after the manner of the potter and burning them in a ceramic kiln. The resulting articles are more refractory than the binding agents employed, and in nearly every case than amorphous alumina. This is also true of the unbonded product, even in fine powders, and it is a fact for physical chemists to explain. The amount of bond is varied in different forms, but in none of these articles does the melting point drop below 1950° C.

Other properties have changed but slightly. For instance, the temperature coefficient is in the more refractory bonded bodies 0.0000059. The thermal conductivity is still twice that of Marquart porcelain. The strength, both tensile and compressive, is very high even at high temperatures. The charge of electrical resistance with temperature as determined by Dr. M. deKay Thompson is as follows: at 535° C., 476×10^5 ohms; at 721° C., 49×10^5 ohms; at 908° C., 24×10^5 ohms; at 1040° C., 7.5×10^5 ohms.

Alundum muffles are distinguished by high thermal conductivity and high melting point. They have four to five times the life of the ordinary clay muffle in gas-furnace work. They are especially adapted to the electrically heated, wire-wound type of laboratory furnace. They have the advantage over quartz of being more refractory and of higher tensile strength. They are porous and cannot be used where entirely gas-tight furnaces are required.

Alundum tubes are similar to the muffles and have found chief use along the same lines. They are also porous and cannot be used for pyrometer protection tubes.

Alundum crucibles are made of varying density, but are still porous in all cases. This prevents their use in fusing slags or salts, but metals, even platinum, may be melted without injury to the crucible. This seeming disadvantage of excessive porosity has been turned to good account in the laboratory sizes. They answer all the purposes of a Gooch filtering crucible without the trouble of preparing the asbestos blanket, and the porosity can be so controlled that the most finely divided substances may be cleanly filtered. The usual analytical precipitates are quickly and surely separated and may be ignited quickly in the crucible on account of its high heat conductivity. Even deflocculated graphite may be separated from its suspending medium.

Large filtering dishes of alundum are also made and have been found useful in making organic preparations and in bacteriological work. An extraction thimble of alundum for the Soxhlet or similar apparatus is very useful as it may be cleaned simply by ignition over a burner and used repeatedly.

Alundum combustion boats of various sizes are valuable on account of good heat conductivity, mechanical strength and chemical inactivity. If it is desired to use a boat of some other material, some carefully prepared alundum grain free from carbon sprinkled over the bottom prevents the sample from sticking in an oxygen combustion.

A useful modification is alundum cement, which may be used to coat the inside of crucibles and furnaces for high-temperature work. It does not melt or combine with carbon below 1950° C. and may be used for lining larger sized crucibles than it has seemed practicable to make entirely of alundum.

There is a wide field of application, and it is possible to produce almost any shape that may be desired. Indeed the greatest difficulty so far encountered is that each user desires a different shape or size so that it has been impossible to prepare the usual stock.

Mr. Saunders had on exhibition a large collection of alundum articles. Mostly all of the forms shown were of laboratory or semi-commercial types. With respect to the larger commercial uses, Mr. Saunders said that by the very nature of the process of manufacture such a substance is costly and in excuse for the early development of the small articles, rather than more commercial forms, he stated that the cost of the material entering into their composition is relatively small, while in the case of bricks it is relatively great.

Some work has been done, however, on testing out alundum bricks. A Héroult furnace in the experimental department of one Niagara plant was equipped with a roof of these bricks in April last year. Since that time the furnace has been used at intervals totaling about three months' actual running time, with some forty or fifty shutdowns, and the roof is still in usable condition. The temperatures used are decidedly higher than in the steel furnaces, so high, in fact, that a regulation silica roof previously tried went down in five or six hours. A trial is now under way in the Héroult furnace of the Halcomb Steel Company, and so far the roof has been in use for twice the life of a silica brick one and is still in fine condition. Until this and other trials are complete no claims will be made for alundum brick. They cannot economically be used, however, in contact with slags, either acid or basic. Nor can they be usefully employed where the present refractory life is very long.

The paper of Mr. Saunders was discussed at some length by Messrs. Tucker, Lidbury, Hering, Hansen, Beckman, Amberg, Richards and Queneau. In reply to questions Mr. Saunders stated that he had determined the comparative melting points of seven commercial refractory bricks; they are in the order of their melting points from lower to higher temperatures as follows: silica brick, chrome brick, bauxite brick, brown magnesite A, brown magnesite B, Grecian magnesite, alundum.

As to the attack of alundum articles by ordinary laboratory reagents, Mr. Saunders stated that first treatment with H_2SO_4 and HCl takes off a little in weight from the alundum article, but never more than three hundredths of a per cent; later treatment takes away nothing. For this reason a preliminary wash of the alundum article is recommended.

As for certain purposes it is important to know the softening temperature rather than the melting point, Mr. Saunders stated that solid alundum does not soften within 100° of its melting point. Bonded articles hold together at 1950°, but would change their form under pressure at that temperature.

With respect to the use of alundum brick for roof construction of electric furnaces, Mr. Hansen pointed out the disadvantage of silica roofs in that they are liable to spoil the bath. The silica roof absorbs basic fumes which reduce the melting point Alundum is not thus affected, and alumina dripping into the furnace from the roof would not have the same disastrous effect as silica dripping into it.

Dr. Richards pointed out that the high thermal conductivity of alundum would make it an excellent material for zinc retorts. To prevent heat loss through the alundum roof of electric furnaces the roof would be jacketed.

New Resistance Furnace.

An account of a new resistance furnace, designed originally by Mr. John Thomson and Mr. Francis A. J. FitzGerald for carrying out the Imbert electric zinc smelting process, was the subject of a paper of Mr. F. A. J. FITZGERALD, of the Fitz-Gerald & Bennie Laboratories, Niagara Falls, N. Y. This furnace has already been noted in our Vol. VIII, pages 289 and 317 (1910).

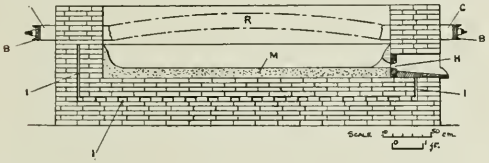


FIG. 1.—LONGITUDINAL SECTION OF RESISTOR RADIATION FURNACE.

If zinc sulphide and metallic iron are heated together to a sufficiently high temperature, the following reaction occurs:

$$ZnS + Fe = Zn + FeS$$

It appears, however, that this reaction is incomplete unless there is a large excess of iron present, or unless the temperature of the reaction is very high. Imbert discovered[1] that by using suitable "dissolvents" this objection to the process is overcome. For example, he found that ferric oxide and iron sulphide mixed together in the proportion of one part and three parts respectively formed a very fluid bath at a temperature between 1000° and 1100° C., and that this bath would "dissolve" six parts of blende. Now, when the zinc blende is "dissolved" in a bath in this way and treated with iron, the reaction given above is complete and works very easily, the zinc being given off as vapor and a residue obtained, which consists of two parts: a slag formed of the gangue of the ore and iron sulphide, which may be used for the regeneration of iron, etc.

The experiments made by Imbert were satisfactory, but the process is obviously not suitable for the ordinary zinc retort, and it was thought that an electric furnace might be used with economy. Experiments with various electric furnaces were tried, and finally the kind which forms the subject of this paper was devised.

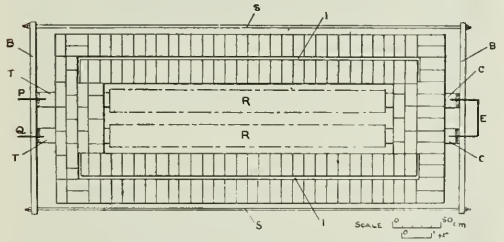


FIG. 3.—PLAN OF COVER OF FURNACE.

The conditions to be met by the furnace are as follows: It is desired to obtain a bath of molten iron, to which can be added a mixture of the zinc ore and the "dissolvent." The furnace must be gas-tight and must be constructed with sufficient ruggedness to withstand high temperatures and rough usage over prolonged periods. It must also be so designed that the temperature can be readily controlled, and finally the heat losses must be reduced to a minimum.

The first furnace constructed had an oblong reaction chamber, which was roofed over by the resistor. This was built of carbon rods laid transversely across the top of the reaction chamber, the ends of the bottom row of rods resting on ledges built of magnesite brick. The resistor was in contact with the furnace terminals, which were situated at either end. A charging door was provided under one terminal, while the tap-hole was under the other. The melting chamber of this furnace had a capacity of 250 kilograms (550 pounds) of iron.

Several experiments were made, but the most important was one lasting 3 days, 5 hours. During this experiment a quantity of pig iron was melted in the furnace, and it was found that with a temperature in the bath of 1300° C. the radiation loss was about 18 kilowatts. Since the full working capacity of the furnace was 40 kilowatts, the efficiency at 1300° C. was 55 per cent, which is not bad considering the small size of the furnace. This experiment was ended by keeping the current at 40 kilowatts without charging anything, the object being to find out how the furnace would finally fail under the increasing temperature. As was expected, failure occurred through the fusion of the magnesite bricks which supported the rods of the resistor.

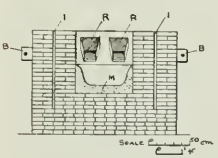


FIG. 2.—CROSS-SECTION OF FURNACE.

After the furnace had cooled it was taken down and examined, with the result that an interesting observation was made which gave a good notion of the temperature finally obtained in the bath. The reaction chamber was lined with Dixon's graphite-clay mixture, used in making crucibles, and it was found that this had been largely converted into amorphous silicon carbide.

Various critics of a furnace built according to this plan have pointed out that an unsatisfactory feature is the risk of fusing the cover. Presumably the heat radiated from the top of the resistor, which, of course, is just underneath the roof, would be so great as to be very severe on that part of the furnace. This objection, however, is not serious, for the current density at the lower part of the resistor is greater than at the top. The carbon rods forming the lower part of the resistor are in much closer contact than those in the upper part, on account of the weight of the upper carbon rods, consequently the resistance of the lower layers of carbon rods is less than that of the upper layers. The fact that the magnesite bricks below the resistor were fused at the end of the experiment, while the roof, which was built of a good grade of ordinary fire brick, was unharmed, shows clearly the difference in temperature between the upper and lower parts of the resistor. Moreover, by means of pyrometers actual determinations were made simultaneously of the temperatures in the bath and in the space between the top of the resistor and the roof. These showed that when the bath had a temperature of 1500° C. the temperature in the space above the resistor was only 1250° C.

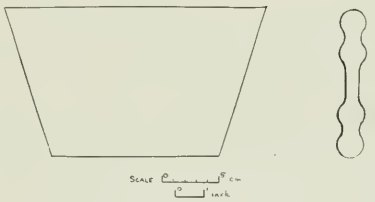


FIG. 4.—RESISTOR CONSTRUCTION.

In view of the fact that the final failure of the furnace was due to the excessively high temperature reached at the end of the experiment, it was concluded that for the lower temperatures required in working the zinc process, no difficulty of this kind would be encountered. Accordingly, a furnace having an estimated capacity of 150 kw was designed. In order to obtain a sufficiently high resistance in the resistor it would have been necessary to give this furnace a length considerably greater than its width, which would be objectionable on account of the relatively large radiating surface that would result.

[1] U. S. Patent, 875,579; Dec. 31, 1907.

Consequently, the furnace was constructed with both terminals at one end and the resistor in two sections connected with each other at the end of the furnace opposite the terminals by a graphite connector. One side of each section of the resistor was then supported on a ledge running along the side of the furnace, while the other sides were supported on a series of arches spanning the center of the reaction chamber in a longitudinal sense. By this arrangement a nearly cubical form was given to the furnace, and consequently the radiating surface reduced to a minimum. The inside dimensions of the reaction chamber were approximately 1400 mm (55 inches) square and 250 mm (10 inches) deep. The resistor was built of rods 19 mm (¾ inch) in diameter and 500 mm (20 inches) long. There were seven layers of these carbon rods in the resistor, each section of which was approximately 1140 mm (45 inches) long.

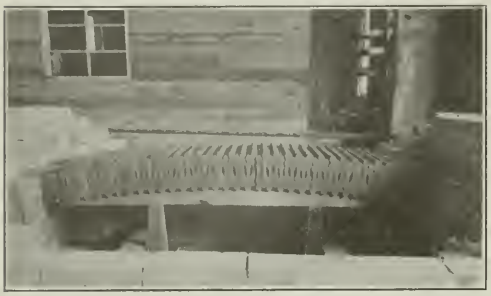


FIG. 5.—RESISTOR PLATES ASSEMBLED.

The experimental work on the zinc process was carried out at the plant of the Hohenlohe Works A. G., Hohenlohehütte, Upper Silesia, where the electrical apparatus was installed and the furnace constructed under Mr. FitzGerald's supervision The electric current was supplied to the plant at 6000 volts, and was transformed by a special transformer. The construction of the plant and the carrying out of a long series of experiments occupied over three months, and the results obtained with the furnace were satisfactory. The highest temperature actually measured in the furnace was 1460° C., but higher temperatures were certainly obtained.

Very interesting experiments were made on the heat losses of the furnace. During one part of the experiments the furnace was held at a constant temperature of 1250° C. to 1260° C. for over three days. During this time the furnace contained a bath of pig iron, and was absolutely idle, so that the energy generated was used simply for supplying the heat lost by conduction and radiation. The rate of generation of energy, when an absolutely constant temperature both in the bath and walls of the furnace was reached, was 33 kw. If, therefore, the furnace is working at 150 kw and a temperature of 1250°, the heat losses would be 22 per cent.

Another experiment was made which lasted 12 hours, and during this time the temperature of the bath was maintained at about 1450° C. Unfortunately, the temperature of the furnace as a whole was not constant at this time, for the temperature of the walls increased 100° during the experiment. The rate of generation of energy during this period was 42 kw, which, working at 150 kw, would give a loss of 28 per cent.

It must be noted that the heat insulation of this furnace was not by any means as perfect as it could be made. The thickness of the walls in the upper part of the reaction chamber was only 340 mm (13½ inches). That a much more efficient use of the heat generated could be made by more careful attention to heat insulation was demonstrated by later experiments.

The chief weakness shown by the furnace was in the magnesite bricks which formed the supports for the ends of the resistor carbons. The temperature of the resistor is very high, as was shown by making determinations of the density of the carbons forming the resistor after they had been in use. This shows that temperatures at least as high as 1900° C. were reached in the resistor itself.

In the experiments made with the Imbert process it was shown that the most satisfactory temperature for carrying on the reaction was between 1350° C. and 1400 C. in the bath, which would not require these excessive temperatures in the resistor. Neverthless, it was seen that the work had to be carried on too close to the limitations of the refractories, and this led to the development of a furnace of a different model.

In working the Imbert process the furnace is filled simply with vapor of metallic zinc, consequently there is no danger of burning the resistor carbons. The charging devices are so arranged that air is not introduced into the furnace, so that the zinc vapor does not burn to zinc oxide which would, of course, attack the carbon of the resistor. If, however, the furnace is used for other purposes it might be necessary to protect the resistor from combustion. For example, if a furnace is desired for copper melting it would be inconvenient and complicated to make arrangements absolutely to exclude air. It is better to enclose the resistor in a chamber, the bottom of which forms the roof of the melting chamber, and then make this resistor compartment air-tight.

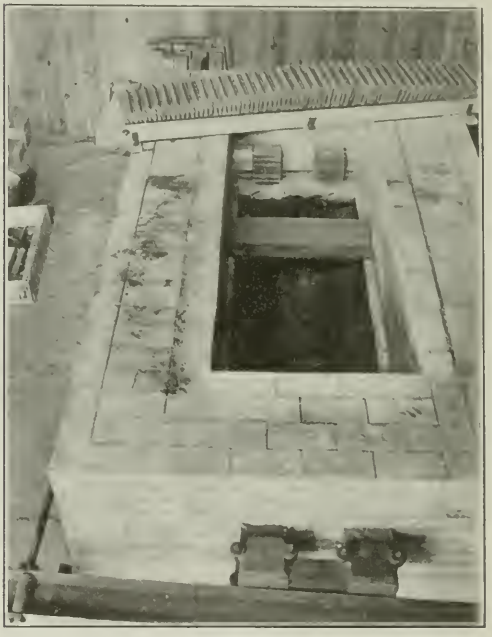


FIG. 6.—TOP OF FURNACE.

A great many experiments were made to determine the best material for forming this combined floor and roof, which requires above all two qualities: refractoriness and high heat conductivity. The result was the choice of recrystallized silicon carbide tiles. These are found to be satisfactory and to last well, even under severe usage. It would obviously be difficult to construct the resistor chamber so that it would be perfectly air-tight, so to prevent all possibility of oxidation it is desirable

to introduce an inert or reducing gas into the resistor chamber and to maintain this atmosphere slightly above external pressure.

A simple method, employed with success in experimental furnaces, is to put a long pipe in the cover of the furnace and mount on this a large oil cup with a sight feed. The oil cup is filled with kerosene, which is allowed to drop slowly into the resistor chamber so as to keep the latter continually full of oil gas. In one furnace constructed in this way after two months' continuous running the current was cut off, the furnace allowed to cool down and the resistor chamber opened. The resistor was found to be in excellent condition, though a few repairs to the terminals were necessary. The resistor chamber was then closed and the furnace is still running in a satisfactory manner, so that altogether the furnace has been working, with slight repairs, more than four months. This particular furnace had been used for a great number of experiments at temperatures varying from 1000° to 1500° C. (1830 to 2730° F.).

In a furnace constructed with the separate resistance chamber several experiments were made on melting aluminum, copper, monel metal, and iron, but the value of these was qualitative rather than quantitative. The furnace walls were only 230 mm (9 inches) thick, consequently the heat losses were so great that the furnace was not by any means efficient. The object of the experiments was to study the weak parts of the furnace. They confirmed what had been already observed, viz., the weakest part of the furnace was the support of the resistor.

The new model of the furnace is illustrated in Figs. 1 to 5. Fig. 1 is a longitudinal section of the furnace, the cover not being shown. Fig. 2 is a transverse section, and Fig. 3 a plan with the cover removed.

The walls and bottom of the furnace are cellular as well as the cover, so that it is almost completely surrounded with an air jacket as shown at I in the Figs. T,T, C,C, are carbon plugs which pass through stuffing boxes in the end walls of the furnace, T,T are the terminals of the furnace and are connected to the cables indicated by P and Q.

C,C are simply connected electrodes which form the other terminals of the sections of the resistor R,R and are connected by E. Bearing on the terminals T,T and the connector electrodes C,C are the channels B,B, which are connected to each other by the tension rods S,S and are insulated from the carbons. The lining of the furnace is shown at M and the taphole at H.

The resistor R,R is constructed of corrugated plates (Fig. 4), where the section is shown on the left and the form in which the plates are cut on the right. In Fig. 5 is a photograph of the plates set up to form the resistor. Referring to Fig. 4 it should be noted that the thickness of the corrugated plate is not the same at the top as at the bottom, but increases from the bottom up, so that when put in place they form an arch as shown in Fig. 5.

This arch form is not necessary, since the interlocking of the corrugated plates holds up the resistor perfectly well without any arch, but it seems to be desirable in the preliminary assemblage and also permits us to obtain a greater current density on the lower surface of the resistor.

By constructing the resistor in this manner all contact with refractories is avoided, the support being furnished by the carbon terminals. The form of the plates is not, however, solely designed to provide an interlocking device, but also to give a high contact resistance.

In one of these furnaces which was built for zinc smelting at Hohenlohehütte, O. S., there were 71 of these plates in each section of the resistor. The dimensions of the plates were as follows:

Length at top	405 mm	(16 inches)
Length at bottom	255 "	(10 ")
Width	165 "	(6.5 ")

When cold the resistor had a resistance of 0.200 ohm and when running at the full capacity of 150 kilowatts and with a temperature in the furnace of 1400° C. the resistance was 0.075 ohm. Now, if the resistance of the carbon itself is calculated it is found to be only 0.00064 ohm, consequently 98 per cent. of the total resistance is due to the contacts between plates.

In order to run a furnace of this kind it is necessary to have means by which a variable voltage can be applied to the terminals of the furnace. If the current is supplied at high voltage and has to be transformed there can be no doubt that the most satisfactory electrical apparatus is something in the nature of an induction regulator which permits of a continuous variation of voltage over a considerable range; but satisfactory results may also be obtained with a transformer from which several taps are brought out from the primary coil, so that the secondary voltage can be changed in a series of steps.

Both for the furnaces built at Hohenlohehütte and those in the FitzGerald and Bennie Laboratories transformers built by the Pittsburg Transformer Company are used with great success. These transformers have a capacity of 150 kilowatts and have 22 taps brought out from the primary coil. Leads are brought out from the two secondary coils and go to a switch so arranged that the coils may be connected either in series or parallel. When all the turns of the primary coils are in circuit the e.m.f. at the terminals of the secondary coils is 50 volts. Then as the turns are cut out of the primary coils the e.m.f. at the terminals of the secondaries is increased in steps of 2.5 volts until a maximum e.m.f. of 100 volts is obtained. By this arrangement there are obtained at the terminals of the furnace voltages of 50 to 100 in 2.5-volt steps and 100 to 200 in 5-volt steps.

When the furnace described above is to be used with an oxidizing atmosphere in the melting chamber the resistor is in a closed compartment, the floor of which is made of recrystallized carborundum plates or graphite plates coated with recrystallized carborundum to prevent them from burning. Fig. 6 is a photograph looking down into one of these furnaces and shows the two terminals and a few of the plates which form the bottom of the resistor chamber in place. The photograph also shows one section of the resistor mounted in clamps and ready to be put into the furnace.

Some anxiety was felt in regard to this design for a rather obvious criticism was that there would be a considerable loss of heat in passing from the resistor chamber to the melting chamber. In other words the thermal conductance of the side walls and top of the resistor chamber would approach that of the bottom of the resistor chamber in spite of the high conductivity of the latter.

Fortunately these fears proved to be groundless, for in the case of the furnace shown in Fig. 6 it was found that with a temperature of 1160° in the resistor chamber there was a temperature of 1090° in the melting chamber, and with a temperature of 1400° in the resistor chamber a temperature of 1290° in the melting chamber. In this particular furnace the heat insulation of the side walls and top of the furnace was not good, owing apparently to the high conductivity of the refractories employed, so the thermal conductance was higher than it should be.

Mr. FitzGerald believes that this furnace will find various useful applications. It apparently gives excellent results in the melting of copper, brass, aluminium, etc. It has not yet been tried for glass melting, but it is thought it will do this work well. Finally Mr. FitzGerald pointed out that the design of this furnace is such that it lends itself readily to the combination of fuel and electric heating. Preliminary experiments in this direction have been carried on and the results are highly satisfactory.

Mr. FitzGerald's paper was discussed by Dr. Whitney and Mr. Carrier. The latter asked as to the advantages of a resistor heater as a radiator over an arc. Mr. FitzGerald replied that with an arc the temperature is definitely fixed to a certain

extent and is higher than is necessary in many cases; further, there are electrode consumption and electrode troubles.

Replaceable Unit System of Electric Heating.

Mr. Milton M. Kohn, of the Multiple Unit Electric Company, of New York, presented a paper with demonstrations on a new replaceable unit system of electric heating. The essential features of this system have already been described on page 229 of our April issue. In this abstract, therefore, only such new points as were brought out in Mr. Kohn's paper are given.

Mr. Kohn pointed out that in comparison with the use of electricity for lighting and power purposes the amount of electric power employed for heating is practically negligible, although electric heating has the advantages of cleanliness, ease of handling, adjustability, and high efficiency. Mr. Kohn finds there are two reasons for this, first the comparatively high cost of electric current, and second the certainty of the electric heating device burning out sooner or later, no matter how carefully it may be used. This latter disadvantage is practically overcome in Mr. Kohn's replaceable unit system, as any heating unit which may burn out can be quickly replaced, as described in our April issue.

Some essential features of his system are summed up as follows: In the first place there is no special chamber used, nor is there one required, except for extremely fine and delicate work, the units themselves forming the chamber. In the second place the resistor is placed in units which have open slots towards the inside of the furnace, so that while the resistor is protected from mechanical and electrical injury which might result from the work which is placed in the furnace, it is still in such a position as to give off its maximum heat to the work, thereby giving greatly increased efficiency.

In certain industrial work, such as burning color on china, glass and similar material, there are pieces, such as very thin plates, plaques and other fragile shapes, where the heating should not be uniform because of differences in cross-section. In other words the same heat applied to the very thin almost knife edges of such work will almost certainly burn or melt the color off it, whereas that portion which is the thickest can safely stand a higher temperature. By the unit system of construction it is a very easy matter to so control the temperature on that portion of the furnace that the narrow or thin edges of the work can be heated very much slower or at a lower temperature, if desired, than the thick or body portion.

Again, in the laboratory work for the heating of various liquids, gases, etc., it is sometimes desirable to have the heating only on one side of the receptable, which may be a tube, flask or other differently shaped device, for the purpose of watching the work while it is being heated. To this end Mr. Kohn employs either a standard unit or a special one built for the particular work in question, which can furnish just the proper degree of heat and at the exact spot where it is wanted.

Another example is afforded by cases where it is desirable to have a "progressive heat." That is to say, that the heat should start very gradually at one end, and increase by successive stages up to the maximum. This can also very easily be done by using a unit of the required shape and length and so placing the convolutions of the heating medium that the heat will be progressive or gradual as required.

Again, by the unit system of construction the heat can be made absolutely uniform throughout the entire structure, or by various methods of connecting the units with and without resistance, which latter can be a part of the unit, a separate resistance in the furnace proper, or an external resistance, or a combination of all three of them together, the heat control can be made as gradual as may be required for the most delicate work.

As to the flexibility of construction, Mr. Kohn gave the instance of a muffle furnace in which the housing or body consists of a number of sections of refractory material; these are formed to make the desired shape, four of them generally being in a layer and possibly three layers being required to make a standard length of furnace. If it is desired to increase the length of the furnace all that is necessary is to increase the number of layers so that there is no limit to the length, except that caused by mechanical strength. The units which fit into this furnace are made in the same way; additional lengths can be added so as to make the required length.

In order to increase the cross-section of a furnace of this character all that is necessary is to insert between the corner pieces which make up the housing or support for the units a distance piece of the required dimensions by means of which the cross-section can be indefinitely increased, the limitation being mechanical. The units are increased in the same manner by inserting a special shaped distance piece, which has the same cross-section as the unit with which it is to be used.

The system is also well adapted to making special heating devices to conform to any specially shaped article that it is desired to heat. For laboratory work hot plates for sand baths, etc., can be made of any size or shape, and to give any desired temperature and any heat control by a simple combination of standard sizes of units and different combinations of circuits. For domestic use this type of construction has an unlimited field.

Another feature of value for domestic and laboratory use is that by means of units which are wound for the various standard voltages it is not necessary to have separate complete devices, but only an extra set of units, so that where a device has been purchased to operate on a 110-volt circuit the purchase of an extra set of units wound for a 220-volt circuit will enable the same device to be used on the higher voltage, everything else remaining the same as before.

Mr. Kohn exhibited some standard types of his furnace and purposely burnt out some units in one of the furnaces in operation in order to show how quickly the burnt-out heating units can be replaced and the furnace restarted.

In the discussion Dr. J. W. Richards criticised the black outside of the furnace and stated that if it was painted white with aluminium paint it would result in a saving of 10 per cent of the heat to keep up the same temperature. Dr. Patten described some experiments of his own with replaceable heating resistor units.

Electric Furnace for Heating Bars and Billets.

Mr. Thaddeus F. Bailey, of the Bailey Engineering Company, Alliance, Ohio, presented a paper on his new electric furnace for heating bars and billets. This method had suggested itself to him when noting the very wasteful and thermally inefficient oil-fired furnaces for heating forging stock used in a comparatively modern plant in northern Ohio.

A calculation of the thermal efficiency of the furnaces in question, which were said to be of the most modern construction, showed that less than 4 per cent of the heat value of the fuel was delivered into the steel.

Mr. Bailey's electric furnace, which is shown in Figs. 1, 2, 3, is of the resistance type, and consists essentially of two carbon electrodes, spaced from each other, and an intermediate resistance body of a carbonaceous composition in which the heat is generated. In the space directly above the resistance material and directly under the roof of the furnace is placed the metal to be heated, a ledge at the rear of the furnace and one at the opening supporting the bars or billets, as the case may be.

The electrodes enter the furnace through the rear wall, and are placed slightly convergent, so that the path of the electric current will be shorter from electrode to electrode at the front of the furnace than at the back. This arrangement compensates for the cooling effect, which is greater at the front of the furnace on account of the opening and the charging of the cold material.

The electrodes entering the furnace from the rear present

comparatively large contact surfaces to the resistance material, without the use of special electrode sections. The electrodes are also placed in a plane above the resistance material, which throws the shortest path for the electric current in the upper part of the resistance body and nearest the metal.

This is a feature that seems almost indispensable to the successful operation of a furnace of this character, since in the furnaces constructed without this feature the cooling of the upper

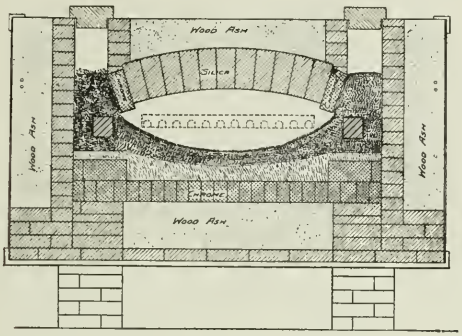


FIG. 1. VERTICAL, RIGHT-LEFT SECTION

part of the resistance body by the cold steel caused the electric current to take a lower path, in which portion of the body the temperature became very high and melted out the bottom linings and the bottom of the furnace itself, and at the same time did not maintain the heating chamber at a temperature high enough to heat the metal.

The resistance material is composed of coke or coal. Crushed foundry coke passing over a 0.25-in. (6-mm) ring and through a 0.37-in. (9-mm) ring giving the most satisfactory and uniform results. For a 40-kw furnace, heating 150 lb. (69 kg) of metal per hour, the distance between electrodes should be 36 in. (90 cm) and the cross-sectional area about 24 sq. in. (150 sq. cm). To maintain an electrical input of 40 kw a voltage of 200 would be required with a coke resistance body; if the body is composed of coal, the voltage should be about 150 volts.

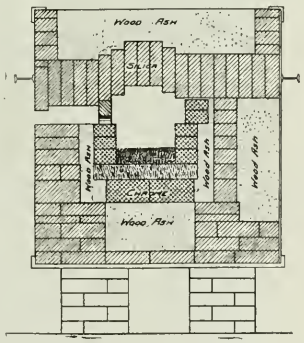


FIG. 2. VERTICAL FRONT-BACK; MIDDLE SECTION.

It is extremely difficult to determine the effective cross-sectional area of the material through which the electric current passes, as at working temperatures the lining materials become almost as good conductors of electricity as the resistance material itself.

The resistance body as it approaches the electrodes has a very greatly increased cross-section, so as to provide a better contact with the electrodes, prevent undue heating at the point of contact, and prevent, to as great an extent as possible, the dissipation of heat through resistance to the passage of the current except at that part of the resistance body directly under the metal.

The most serviceable lining material used by Mr. Bailey is made of chrome brick and chrome ore. The top of the furnace is composed of silica arch brick, and is 9 in. thick, and clamped from front to back. A row of chrome arch brick is placed at each side of the roof to prevent contact of the silica brick with the carbon in the hoppers above the electrodes.

The sides of the furnace are composed of silica brick and, in the larger furnaces, are incased in a sheet steel frame. Insulating material is placed between the silica brick and the casing.

The furnace base consists of a heavy cast-iron plate supported by cast-iron legs or brick piers.

The cable terminals are connected with the electrodes by means of iron sleeves, copper straps and iron wedges.

In making the connection the iron sleeve is placed over the end of the electrode and copper straps placed against the four sides of the electrode. The copper cable terminal is then placed between one of the copper strips and the iron sleeve, and wedges driven in on the opposite side between the strap and sleeve, and a firm contact thus secured. Wedges are then driven in on the other two sides between the copper strips and

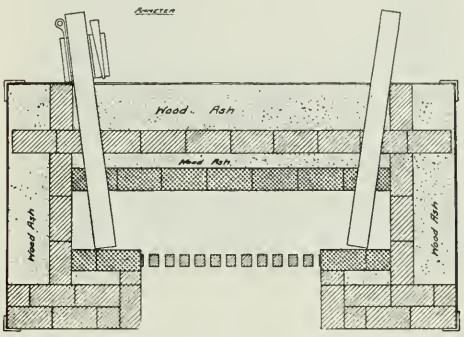


FIG. 3. HORIZONTAL SECTION.

the sleeve. In this way the current flow is equally distributed on all four sides of the electrode by means of the encircling iron sleeve.

The furnace is controlled by means of the regulating transformer and controller. The transformer gives full rated output at the lowest voltage rating. While hand regulation is usually provided the regulation may be made automatic at a slight additional cost. A voltage range of from 33 to 50 per cent is usually provided, depending on the operating and starting requirements.

The metal being heated is maintained in a reducing atmosphere, the high temperature carbon resistance material giving off a gas of a reducing nature. The electric furnace is the only furnace built in small units in which both a high temperature and a reducing atmosphere may be maintained. The saving in oxidation loss is very considerable in small furnaces of this character, as the loss from oxidation in the usual combustion furnace is the cause of the loss of a large number of forgings, due to the fact that a piece of stock, having lost a considerable amount of metal by the oxidizing flame of the combustion furnace, does not contain enough metal to fill the dies.

The thermal efficiency of the furnaces varies with the size, the percentage of operation at full capacity and the ruling temperature required. Efficiencies of from 33 to 65 per cent may be expected in furnaces with heating capacities of from 120 lb. (55 kg) per hour to 1000 lb. (455 kg) per hour.

In the concluding part of his paper Mr. Bailey gave details of a test of an experimental furnace at the plant of the Transue & Williams Company in Alliance, Ohio.

The principal advantages of this electric furnace are stated to be its high thermal efficiency, its non-oxidizing atmosphere in the furnace chamber at all temperatures, and its freedom from

soot, smoke and the hot contaminating gases of the products of combustion.

In the discussion of Mr. Bailey's paper Dr. Richards remarked that the efficiency of this furnace was high in view of the unfavorable conditions and that very effective use was made in this furnace as in Mr. FitzGerald's furnace (see abstract above) of the principle of getting the resistor hottest at the place of its radiating surface. Messrs. Hering, Sperry, Lidbury and Hitchcock also participated in the discussion.

As to the "efficiency" of an electric furnace, Dr. Richards proposed to define it as the ratio of the heat actually used for

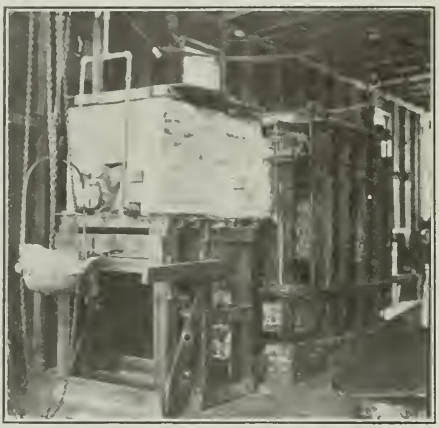


FIG. 1. ELECTRIC ZINC FURNACE.

useful purpose to the electric energy consumption. Mr. Lidbury did not like this definition, as according to it any furnace in which the temperature of the charge is not raised, for instance the Heroult steel furnace, would have zero efficiency.

In reply to a question of Mr. Hitchcock, Mr. Bailey stated that coke was usually added every five or six hours.

Electric Zinc Furnace.

Prof. Jos. W. Richards, of Lehigh University, then presented a paper on the Johnson electric zinc furnace. He stated that at the present time there is considerable interest among metallurgists in electric zinc smelting. This is an exceedingly gratifying circumstance to the American electrochemical world, as thirty years ago those electric furnace pioneers, the Cowles Brothers, tested out an electric furnace for smelting zinc ores from a New Mexican zinc-copper mine. As zinc smelters they were ahead of their time, but their interest in that zinc mine was one of the predisposing causes of the rise of the electrothermic industries during the last two decades of the last century.

If the zinc business once helped the electric furnace, certainly now the electric furnace can help the zinc business and repay its debt by giving it a continuous furnace of large capacity. For it is a commonplace remark in metallurgical circles that the process for making spelter has not been improved in principle since its birth, one hundred or more years ago. This is true in a measure and in a measure untrue. It is untrue in that hydraulic-pressed retorts and regenerative retort furnaces and mechanically operated calcining furnaces are advances. But it is true in that the same old retort averaging a charge of 50 lb. (23 kg) of roasted ore, with the costly and inefficient crude hand labor, with its limited life and its inability to stand corrosive ores, is still the operative unit. In the United States there are some 100,000 retorts making each 5000 lb. of metal per year.

Now, in this respect the electric zinc furnace which puts the heat right where it is needed and used, and so can be built in large sizes, is far ahead of the old retort potentially. Furthermore, a continuous character of operation is possible in electric smelting, and this is inherently better than the intermittent character of the present method of smelting of zinc ores.

It is quite within the bounds of probability to expect 500-kw zinc furnaces, each making 4000 tons of metal a year, the equivalent in production capacity of 1600 retorts.

The natural presumption is that when once developed to a practical and commercial basis the extraction of the main and by-product values will be greater in the twentieth century process than in the nineteenth century process. Likewise the labor charge per ton of ore will be far less, the metal purer and more uniform. The capacity for smelting a variety of ores will be greater, and the control of temperatures be better. The cost of electric power will approximately equal the cost of coal in the present practice.

Mr. W. McA. Johnson's first large-scale experimental work in the electrometallurgy of zinc was directed along lines of treating a charge that produced a dry fusible residue. Much actual progress was made in that line. Photographs of his earlier furnaces are shown in Figs. 1 and 2. He later concentrated his investigation on a proper practical condenser for large-scale continuous operation, which he considers to be the keystone of the "electric zinc" problem. For the last two years Mr. Johnson has been engaged in developing an electric fur-

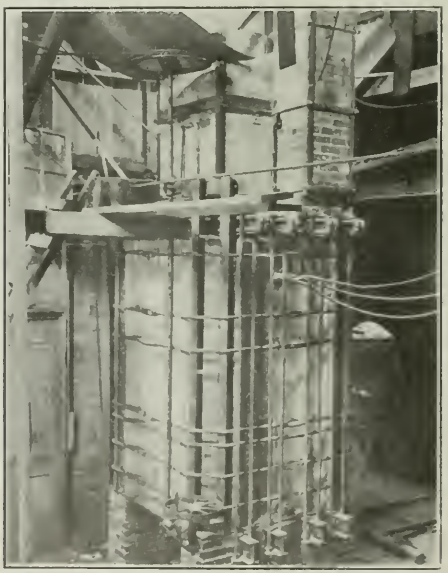


FIG. 2. ELECTRIC ZINC FURNACE.

nace to smelt zinc ores with the production of a base lead bullion, a copper-matte and a fusible slag. This is more difficult than the dry process, but far more attractive commercially. To use his own words "the smelting zone of my electric zinc furnace is practically a lead furnace, but operated by electrical heat and with slightly more intense reduction conditions, sufficient to reduce the zinc but not to reduce appreciable amounts of the iron oxide present in the slag."

The furnace (see also our Vol. VIII, p. 688, December, 1910) is of the Siemens arc type with certain important modifications.

In the smelting operations a certain percentage of carbon dioxide is evolved. This is fatal to efficient condensation. To reduce the percentage of blue-powder and dross to a minimum, or to ensure a high "condensation factor," i. e., a high percentage of metallic spelter made direct to total spelter made, Mr. Johnson passes his reduction gases through a filter of carbonaceous material. This reduces the percentage of carbon dioxide to extremely low limits, lower than those in the gases emanating from the ordinary retort. The condenser that Mr. Johnson uses in conjunction with this type of furnace is simple, yet designed to expose the vapor to the surface of metal, to abstract the heat and to regulate the temperature. Mr. Johnson's condensation factor averages usually somewhat higher than that in the ordinary retort plant. In usual practice from 70 per cent to 80 per cent of the spelter is made direct, and from 30 per cent to 20 per cent from the resmelting of "blue-powder" and furnace products.

Mr. Johnson's slag losses are remarkably low. Usually 99.5 per cent of the zinc content of the charge is reduced and volatilized, while 0.5 per cent of the zinc content passes off in the slag. The remarkable feature of the little 25-kw furnace which Dr. Richards had seen in operation is its great smelting capacity per unit of cubic content. The rate of reduction is about fifty times as fast as in the fireclay retort, and the rate of smelting about five times as fast as in the ordinary lead furnace. Furthermore, with proper manipulation the heat requirements of the charge are reduced to a minimum. Mr. Johnson is now engaged in working out his ideas on a much larger scale. If he is on the right track the ease of smelting and condensing will be greater in his commercially sized unit than in his 25-kw furnace. In the light of metallurgical experience an inventor is on the right track when on increasing the size of his operation his difficulties and annoyances decrease and conversely he is on the wrong track when on increasing the size of his operation his troubles increase.

Entirely irrespective of the merits and possibilities of Mr. Johnson's electric zinc smelting furnace, Dr. Richards believes that we shall see in the immediate future the start of the abolition of the fireclay retort, and that we now see the dawn of the new era of electric smelting in the zinc business.

Condensation of Zinc.

A paper by Mr. F. T. SNYDER, of the Metallurgic Engineering Company, of Chicago, dealt with the condensation of zinc vapor from electric furnaces. If the electric zinc furnace is successful, the design of large zinc condensers becomes of great importance. With electric zinc furnaces it is practical to deliver zinc vapor to a condenser at a rate 300 to 400 times the rate at which such gases reach the standard retort condenser. It is at once evident that the trade-skill basis of design is inadequate and that the opportunity exists for the application of engineering methods to the design of zinc condensers.

Mr. Snyder bases his discussion on the well-known data of the design of the successful small zinc retort condenser and shows how to derive therefrom data for the design of larger condensers. He analyzes in great detail the conditions of temperature and pressure and the phenomena of zinc condensation in the ordinary retort condenser. From these he reaches the following chief conclusions for use in the engineering design of larger condensers.

If we provide 1 sq. cm of clay cooling surface (or an equivalent amount of some other material) for each 0.40 gr. of zinc to be condensed per hour, and if we arrange the thickness of the material of this surface and the temperature conditions at the other or non-condensing side of the material so that 0.21 kg calories of heat will flow through each sq. cm of the surface per hour when the average temperature of the condensing surface is 725° and if we arrange the volume inside the condenser, so that the product of the average longitudinal velocity of the gases in cm per second and the average diffusion distance (corrected to the standard temperature of 864°) in cm, through which the zinc has to diffuse, is 9.5, then, the design of the large condenser will be on the same basis as the proportions of the standard retort condenser. It is to be kept in mind that only the cooling surface in the condenser that is between 850° and 450° counts in the production of liquid zinc. As this upper limit of 850° depends on the atmospheric pressure, the reduction of the atmospheric pressure with high altitude is to be taken into account for a condenser so located. It is interesting to note in the standard retort condenser, the proportions of which have been determined by many years of "cut and try," that these proportions now in general use give a diffusion factor that is substantially constant throughout the length of the condenser. This means that these proportions represent the maximum commercial economy in the use of the condenser material.

As to further details reference must be made to the full paper, which will be printed in full in the *Transactions*.

The author finally examines the design of one of the large condensers which have formerly been built—that in connection with the Lynen furnace—and shows in what respect this design was deficient.

In the discussion which followed this paper, Mr. Hansen stated that in a continuous run with the Weeks electric zinc furnace at the General Electric Company's plant in Schenectady 1600 lb. of zinc had been successfully and very satisfactorily condensed. Messrs. Bleecker and Quenean also participated in the discussion. In reply to a question of the latter, Dr. Richards stated that the specific yield of Johnson's furnace is 2 tons of zinc per horse-power-year; which is better than at the DeLaval plant in Trollhattan.

Reliability of Electric Furnaces.

Another paper by Mr. F. T. SNYDER dealt with the reliability of electric furnaces for commercial work. This paper had previously been presented before the Chicago section of the society. The conclusions are summed up as follows:

The electric furnace itself has passed from the field of experiment to that of engineering, but the fields of manufacture using the electric furnace products are still experimental.

The electric furnace is technically reliable and will operate continuously with the men and supplies that are available in this country. The details are simple and rugged, and the inherent regulating powers can be made such as to bring it well within the ability of usual plant labor.

The electric furnace is commercially reliable. When installed with the same business care and adaptation to conditions that should be used with other furnaces it will earn a profit on the investment, and a profit that is larger than the normal manufacturing profit in proportion as the field is newer and more open.

Incidentally Mr. Snyder mentioned that he recently examined an electric furnace plant, which started operation on a commercial scale in the summer of 1907 and which is now to be increased fivefold. Substantially all of the original equipment is still in use. The plant, which is in the West, has been operated since its installation by Chinese labor under a white superintendent and neither the superintendent nor the Chinamen know more of electricity than is involved in the opening and closing of circuit-breakers and the reading of ammeters and voltmeters. This particular plant, while not a steel plant, is of interest in showing on many of its operating records a heat development efficiency of over 90 per cent. That is, of the heat equivalent of the energy delivered to the plant by the high-potential electric wires, over 90 per cent is transferred to the material under treatment.

The paper was discussed by Dr. Amberg and Mr. Fitzgerald. The latter referred to the claim of a 90 per cent efficiency and said that as the loss of energy in the transformers, which presumably are used in connection with the high-potential wires, must amount to about 3 per cent, there is left only 7 per cent for the losses by radiation and the various other losses inevitably

associated with work of this kind. "It is not intended to deny the accuracy of Mr. Snyder's statement, but merely to point out that such an extraordinary observation should be at least substantiated with an account of the methods used in arriving at the result."

The Cyanamide Process for Gold.

The first paper called for discussion in the session of Friday morning was the paper of Mr. John Collins Clancy, of the Moore Filter Company, of New York City, on the electrolytic cyanamide process for treating gold ores which was presented before the New York section of the society some time ago and which was published in full in our January issue, page 21. It was discussed by Dr. N. S. Keith and Dr. A W. Smith.

Dr. Keith gave some principles of the ordinary cyanide process with special reference to Elsner's equation $4Au + 8 KCN + O_2 + 2H_2O = 4KAu(CN_2) + 4 KOH$, to show why and for what purpose the free oxygen is needed in the process, and described some simple experiments demonstrating some of the reactions in the cyanide process. He then criticized rather severely several points in Mr. Clancy's paper, especially the use of the term "protective alkalinity," further the proposed employment of cone-shaped tanks supplemented by blankets for separating sulphides from the gangue accompanying them in the pulp. He also thinks that the solvent of Mr. Clancy is rather complex and that the use of the electric current for regeneration of the solvent is too expensive.

Mr. Clancy's reply will be printed in the *Transactions* of the Society.

Electrolytic Amalgamation.

A paper by Mr. Elmer Ellsworth Carey dealt with "the electrolytic system of amalgamating gold ores." He pointed out the difficulties of ores where crystals of iron pyrites are found in quartz, and within the pyritic crystals are gold particles in a finely divided state. The values in such crystals cannot be recovered on the usual mill plate, as the particles of gold are coated with various substances, preventing amalgamation. The present practice is to separate the pyritic crystals from the crushed ore by various types of concentrating tables. This so-called concentrate is then sent to a smelter, or it is ground in tube mills (100-200 mesh) and delivered to the cyanide tank, where the extraction ranges from 85 to 96 per cent.

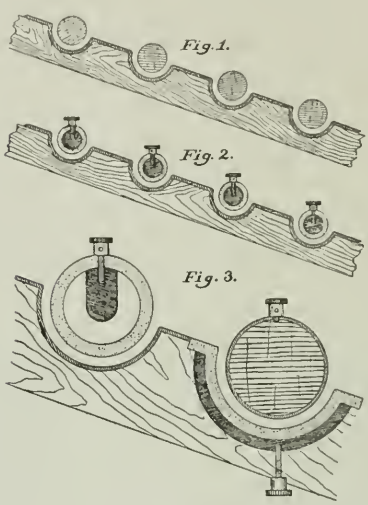
For these treatments Mr. Carey proposes to substitute fine grinding (100 mesh) and electro-amalgamation. An ideal plant for this process consists of some type of rotary crusher, with outside screening, possibly a secondary crushing device to finish the work of the first crusher, and from the crusher the pulp, ground sufficiently fine to release all economic values, is passed over electrolytic amalgamating devices. The released values are recovered in the form of amalgam.

In electrolytic amalgamation, the sole function of the electric current is to deposit hydrogen, sodium, potassium or ammonia in the mercury. The sole function of sodium or potassium is to deoxidize water, and the final work in the chain of reactions is the liberation of nascent hydrogen at the surface of the mercury. As the particles of gold sweep over the electrolytically excited mercurial surface, all substances usually preventing amalgamation are automatically and almost instantly destroyed or rendered inert. Grease is saponified by the caustic soda; oxide coating on gold particles is reduced by hydrogen; in a word, gold particles are cleaned and amalgamation quickly follows.

Hungarian mercury wells are described in Vol. II of Prof. R. H. Richards' Work on Ore Dressing; by making the baffle in such wells an anode, using suitable material, and employing an 8 to 12-volt current of high amperage, we have an electrolytic amalgamating device. The objections to the use of such mercury wells disappear when they are electrified, and their extraction efficiency is greatly increased. The interior of such wells must be lined with some very refractory substances.

A simple experiment showing the activity of hydrogen-sodium amalgam and the value of electrolytic amalgamation is to place a piece of plastic sodium amalgam of the size of a pea in a test tube, to add an ounce of water heavily saturated with ammonium chloride, and to test the amalgamating powers of the very curious resultant amalgam by copper and iron wire.

The amalgamator for gold ores consists of a series of electrolytic sodium-mercury cells, followed by another electrolytic amalgamating device in which the values are recovered on silver-plated copper plates of suitable construction. With the mercury wells high amperage may be employed, liberating large volumes of hydrogen, which removes all coatings (oxides, sulphides, grease, talc, silicious coatings, etc.) from the micro-



FIGS. 1, 2, 3. ELECTROLYTIC AMALGAMATION APPARATUS.

scopic gold particles; in the secondary amalgamator the plates offer a large cathode area so that all portions of the pulp are forced into contact with a highly active mercury surface. The supplementary amalgamating device above mentioned consists of a silvered copper plate of suitable width; this plate contains parallel, transverse, semi-cylindrical depressions (grooves, pockets or riffles) the full width of the plate; into these grooves project cylindrical terra cotta cylinders, leaving a ¼-in. clearance, from 3 to 4 in. in diameter; these cylinders contain a graphite core, 1 in. in diameter, connected as an anode.

The pulp passes under the cylinders, and sweeps gently over the curved (cathode) amalgam plate, while gravity, the force of the water and centrifugal force tend to drag each gold particle into contact with the highly excited mercury surface.

Fig. 1 shows a section of a new type of mill plate which Mr. Carey has devised. The sides are not represented. The silver-plated copper plate (being of the usual length of amalgamating plates) contains transverse parallel semi-cylindrical grooves into which fit solid cylindrical wooden baffles; these baffles may also be made of standard piping or casing, from 3 to 4 in. in diameter, with the ends closed. The clearance between the plates and the baffles is from 3/16 to ¼ in. One piece of copper plate may be used, with the proper depressions pressed therein, or, preferably, a number of overlapping plates may be used, each plate containing one or two semi-cylindrical depressions. The plates are held in position by their own weight, fitting closely to the sides, and can be quickly removed from the device for cleaning up. The device may be given any desired grade, and the plates are dressed and operated as the usual mill plate. A mill plate arranged as described will make a

better extraction on the average ore than the usual type of amalgam plate, there being no loss of amalgam.

In Fig. 2 is illustrated an electrolytic mill plate, constructed similarly to the one shown in Fig. 1, which will not only recover all values saved by the standard types, but in addition all free values in silicious pulp and slimes, all values in placer material, beach-sand and all black-sand values are also recovered. The cylindrical baffles are made of terra cotta, and each contains a graphite core connected to the positive lead of a low-voltage generator; the amalgam plates are connected to the negative lead of the generator.

In the first groove or riffle shown in Fig. 3 the water and pulp pass over the amalgam plates as in Fig. 2; in the second riffle of Fig. 3, the amalgam plate connected to the negative lead forms a casing for the baffle, and the pulp stream passes under the mercurial surface, thus bringing the surface of the water into intimate contact with the electrically excited mercurial surface; or a copper cylinder, silver plated, may be used as a baffle, and at the same time act as an amalgamating surface.

A gold-saving device may consist of a series of such riffles as are shown in Fig. 3, arranged alternately; such an arrangement is particularly useful in treating pulp containing gold in a finely divided form, or for recovering values in slimes or in solutions. By screening placer material to 10 or 12 mesh, and passing the undersize of electrolytic amalgamating sluiceway of suitable length, all fine, rusty, float, coated, and greasy gold is recovered.

In certain classes of base ores it may be necessary as a preliminary measure to treat the pulp for 30 minutes by electrolytic pan amalgamation before passing the pulp over the electric sluice; and for ores containing gold in chemical combinations (sulphotellurides, etc.) a preliminary roasting may be required.

In the discussion of Mr. Carey's paper, Dr. Keith said that, while the idea of electrolytic amalgamation was old, Mr. Carey had improved on the apparatus.

Electrolytic Corrosion of Electroplated Objects.

A paper by Prof. Walter S. Landis, of Lehigh University, referred to the observation that gold-plated steel pens corroded far more rapidly than plain steel pens. This is due to the fact that the friction of the pen point on the paper rapidly wears off the plating on the point, thus exposing the steel; an electrolytic couple consisting of steel, ink, and gold is thus set up and corrosion results.

The e.m.f. of the couple was measured between a fresh gold-plated pen and a like pen, stripped of its gold plating, in different kinds of ink.

"While the corrosion of the plated pens is not a matter of great commercial importance, yet there is to be gained from the experience a certain wholesome lesson on the application of electroplating to the protection of less noble metals: Do not plate with a nobler metal a material that is likely to be subjected to wear on one spot and which spot is to be moistened with liquids. For, as soon as the plating is worn through on that spot, accelerated corrosion will take place, and in the end the resulting corrosion will be far worse than without the plating."

The paper was discussed at some length by Messrs. Murphy, Walker, Hering, Richards, Hogaboom, Kohn, and Willard. The latter pointed out that nickel-plated iron or steel will invariably rust in a moist atmosphere. Not only the weight of the nickel deposit, but also its continuity are of primary importance, and it seems impossible to insure absolutely the latter in commercial practice.

Electrolytic Iron Refining.

A paper of Prof. Charles F. Burgess, of the University of Wisconsin, which had already been presented before the Chicago section of the society, dealt with electrolytic refining as a step in the production of steel. The question is: Can iron, refined electrolytically by his process (our Vol. II, page 183), be made an industrial product? Iron can be refined electrolytically almost as easily as can copper. Whether there is a field of usefulness for such iron depends upon whether electrolytic iron has superior qualities due to purity or physical condition, and whether it can be procured at low cost.

From the standpoint of purity it must compete with some notable metallurgical developments which have resulted in the product such as is being made by the American Rolling Mill Company in a basic open-hearth process. The electric furnace is another factor which is contributing largely to the control of the composition and purity of iron alloys.

The chief source of commercially pure iron has been, and perhaps now is, the high grade Swedish and Norway iron, used largely as a base material for high-grade crucible steel. The analyses of this material usually indicate a high purity, though frequently and erroneously calculated by difference after determining carbon, sulphur, silicon, phosphorus and manganese. It is not uncommon to find oxides and slag to the extent of 2 per cent in this material, and this impurity undoubtedly may have an influence on the resulting alloys made from it.

Electrolytic iron can be produced with a purity as high as 99.97 per cent and perhaps even better, using extraordinary precautions. This record has been made by using the best commercial grades of pure iron as anodes. A few hundredths of a per cent of purity must be sacrificed in using anodes of mild steel or other less pure materials. Further the uniformity of this electrolytic iron should be of importance for its uses.

Using an electrolyte containing 40 grams of iron per liter in the form of ferrous sulphate together with 40 grams of ammonium chloride, it has been found possible to conduct a continuous refining operation for many months at a current density of 6 amp to 10 amp per square foot of cathode surface, and at a potential difference of about 1 volt. The current efficiency is very close to 100 per cent, as it is in copper refining.

This leads to the calculation that 1 kw-hour will refine 2 lb. of iron; or a cost for power of ½ cent per pound is attainable. The costs for labor, solution maintenance and fixed charges are estimated to be not greater than the power costs, making a cost of refining of about $20 per ton. Assuming the anode material to be mild steel costing $35 per ton, the cost of electrolytic iron would be in the neighborhood of $55. These approximate calculations indicate that this material might well compete in price with high-grade Swedish iron.

Among the properties of electrolytic iron which may give it some added usefulness is its content of hydrogen and the brittleness which results from this occluded gas. This hydrogen may be of some service in reducing oxides in a melt. The brittlenes of the electrolytic iron before the hydrogen is driven off makes it easy to break it up into pieces suitable for introduction into the steel crucibles.

Unsolved Problems in Electroplating.

A paper entitled "Some Unsolved Problems in Electroplating" was presented by Mr. George V. Hogaboom, secretary of the National Electroplaters' Association. He pointed out that electroplating has been looked upon more as a trade than a science, and that its development has been left to the practical man, guided only by "rule of thumb." An electrochemist in the plating room of a factory is so rare that it probably can be said without fear of contradiction that they can be counted on the fingers of one's hands. The need to-day is mutual co-operation between scientific electrochemists and practical platers in solving their problems and developing new ideas.

In the discussion of electroplating problems, it must be borne in mind that a mere deposit of a metal is not all, but that the deposit must be soft and smooth and lend itself to a decorative process; the anodes should be capable of being reduced easily; the electrolyte must offer little resistance to the electric current,

and, last, but not least to the plater, who hears it so often that it becomes a part of him, the cost must be nominal.

The automobile industry has brought about, more than anything else, the need of a heavy deposit of brass. At present this is done in a solution of cyanide of copper. The deposit is not only slow, but unsatisfactory, because of what is known as "spotting out"—a discoloration in spots which appears on the work after it has been polished and lacquered. Deposits on cast metal give the most trouble It is probably caused by the acids or alkaline solutions being absorbed in the pores of the metal, or in the small blow-holes, and the deposit covering these holes partially, leaving minute holes through which the solution oozes out. Several remedies have been suggested and tried, such as boiling out in some neutralizing chemical solution, placing in a drying oven for several days, but a satisfactory remedy has not been found.

An acid brass solution would be a great advantage. There is an acid copper and an acid zinc solution, but no acid brass electrolyte. The difference between the deposit from a cyanide and a sulphate of copper solution well illustrates the advantage of having an acid brass solution.

Tin.—The formula for a tin solution published by Roseleur is the most generally used to-day, as little, if any, improvement has been made upon it, although a good solution which would give a heavy deposit is much desired. In Roseleur's solution the electrolyte is not replenished by the anode, but by the constant addition of a concentrated solution. This should be overcome, and would be appreciated by manufacturers of tinware.

Aluminium.—While several solutions have appeared from time to time for plating upon this metal, none of them are in general use, and a good electrolyte that would deposit gold, silver, brass or copper so that it would stand burnishing and not peel off in time could be used.

Nickel.—The successful removal of a deposit of nickel from another metal without affecting the latter has not been accomplished.

To give all the unsolved problems in detail would make a lenghthy paper, and a simple statement of those most desired will be given:

An electrolyte that will remove the fire-scale from brass: also one that will produce a bright or a matte surface in place of using the present acid dips.

An electric cleaner that will saponify the grease and take it into solution instead of driving it to the top, where it has to be constantly removed to prevent it adhering again to the work as the latter is removed from the solution.

A heavy deposit of lead on the inside of iron pipes, to prevent rapid corrosion.

A method to coat electrogalvanized iron or steel with decorative metals without destroying the rust-resisting properties of the zinc.

An alkaline nickel-silver solution that can be worked with a low voltage.

A method of etching steel without destroying a resistance film of gelatine.

Some alkaline substance that would replace cyanide of potassium. This would be universally welcomed.

For the above suggestions the writer is indebted to 100 different platers, who were kind enough to answer a request for unsolved problems. It may be interesting to note that 80 per cent of them requested an acid brass solution.

In the discussion Mr. Lidbury said that the list of unsolved problems was quite interesting but that it would seem that at least half the problems were easy of solution, if electroplating shops would employ competent electrochemists. Dr. Walker said that in their jealous secretiveness many modern electroplaters (and some other manufacturers) compared with the old alchemists, but that progress depends on the free interchange of ideas.

Polarization.

A paper by Dr. Oliver P. Watts, of the University of Wisconsin dealt with polarization in the relation to the decomposition pressure of electrolytes. Dr. Watts endeavored to show that the principal factor in causing the phenomenon of the decomposition voltage as observed in electrolysis is not the resistance of a gas or other film upon either or both electrodes, but that it is due to a counter e.m.f. which is produced at the electrodes as a result of the action of the impressed e.m.f. This counter e.m.f. is referred to as polarization.

By the introduction of a third electrode and the measurement of the differences of potential between this and the anode and cathode, it is possible to ascertain what proportion of the total polarization is contributed by the anode and what by the cathode. The most satisfactory reference electrode to use for this purpose is the normal calomel electrode. The results of various tests are given in which the total polarization as well as the anodic and cathodic polarizations separately were determined.

The source of the counter e.m.f. is of the same nature as the e.m.f. of any primary or storage cell, viz., it is the result of an unsymmetrical electrochemical system, consisting of two electrodes in one or more electrolytes. Before electrolysis the electrolytic cell was symmetrical consisting of like electrodes in an electrolyte of uniform composition, concentration and temperature, and therefore the e.m.f. of the cell was zero. With copper sulphate and platinum electrodes the application of an e.m.f. causes the system to become unsymmetrical, by the deposition of copper at the cathode and oxygen at the anode. The e.m.f. of this little copper-oxygen storage cell, which, as in all storage cells, is opposite in direction to the charging e.m.f., continues to increase in magnitude, as more and more copper and oxygen are deposited, until its e.m.f. equals the impressed e.m.f., when the flow of current ceases. All this may occupy one or two seconds. With each increase of impressed e.m.f. this process is repeated, until the value of the impressed e.m.f exceeds that attainable by the little storage cell; then a permanent current flows.

This refers to insoluble electrodes. With soluble electrodes the concentrations of the electrolyte are changed right at the electrodes and this change of concentrations produces a counter e.m.f.

The final conclusions of Dr. Watts' paper are as follows:

Polarization is responsible for the phenomenon of decomposition pressure in electrolytes.

The total polarization may be resolved into two components, the one at the anode, the other at the cathode.

Since it is polarization which prevents the flow of current in accordance with Ohm's law, the residual current is the equivalent in current of the rate of depolarization of the electrodes by the electrolyte

Moreover, if electrolysis causes a change in the material of an electrode, or of the electrolyte in immediate contact with it, the polarization at the electrode is large. This is the case with an insoluble anode, and usually so for a cathode whose position in the electrochemical series is far from that of the element which is deposited upon it by the current.

If the chemical composition of the electrode and of the electrolyte in contact with it are not changed, the polarization will be small. This occurs with a soluble anode and with a cathode of the same material as that deposited by the current.

The paper was briefly discussed by Mr. Hering and Dr. Richards.

Electrical Heating as a Means of Prevention of Superheating.

A paper by Dr. J. Howard Mathews, of the University of Wisconsin, described a modification of the method proposed a number of years ago by Bigelow for the determination of molecular weights in solution by elevation of the boiling point.

The modification consists primarily in the arrangement of the heating spiral, in the use of a heavier heating current and in the more accurate control of the heat thus supplied. With the apparatus as described, and with the precautions enumerated, it was found possible to make molecular weight determinations with a degree of accuracy heretofore unattainable. The use of glass bead in the boiling chamber has not only been shown to be useless as a means of prevention of superheating, but is a positive detriment.

A long series of determinations were then made to ascertain whether the slight superheating effect experienced would be the same when wires of different materials were used for the heating elements. It was shown that German silver wire produces a much greater degree of superheating than any of the other wires used. Wires of the resistance materials known as nichrome, advance, manganin, and monel showed but little tendency to superheat. The tendency to superheat the liquid was not found to be dependent in any way upon the specific resistance of the heating element. German silver wire when rolled out in the form of a ribbon and well roughened with a file showed about as little tendency to superheat the liquid as did wires of platinum or of the other materials above enumerated. A covering of oxide over the surface of a smooth German silver wire seemed to produce no effect upon the tendency to superheat.

Bifunctional Accumulator.

Mr. A. O. Tate, of Toronto, Ont., Canada, then presented a paper on his new "bifunctional accumulator." Nothing is changed in the chemistry of the ordinary lead accumulator. Mr. Tate's work has been chiefly along mechanical lines. The positive and the negative elements are contained in the same plate on both sides of it, hence the name "bifunctional." The chief mechanical feature is the use of short narrow conducting strips, separated by perforated celluloid strips. The slight depth of the active material permits very quick charging. Other claims for the battery are very uniform current density, insurance of good contact between active material and support, and avoidance of sulphating.

In reply to a question by Dr. Brown, Mr. Tate stated that the battery gives 11 to 12 watt hours per pound of complete battery in service.

Production of Aluminium in Laboratory.

Dr. H. K. Richardson, of the Pennsylvania State College, presented a paper on "Some Observations on the Laboratory Production of Aluminium." The paper gave an account of runs made by students during the regular scheduled laboratory periods in electrolytic deposition of aluminium. The author follows in many respects the method of Neumann and Olsen (this journal, Vol. VIII, p. 185).

The instructions given to the students are as follows:

Make 4 kg. of a mixture:

Alumina .....20 to 15%

Natural cryolite .....80 to 85% old charge

Melt up 3 kg. of charge at once. Add the remainder as the bath becomes diminished.

Remove the heating carbon as soon as the bath is thoroughly fluid.

Add alumina to bath every 30 minutes. Quantity to be calculated on assumption of 50 per cent current efficiency.

Use anodic current density of 3 amperes per square centimeter.

If anode effect causes trouble, *i. e.*, current suddenly drops

(1) Open and close circuit quickly

(2) Add a charge of alumina

(3) Tap electrode sharply

(4) If none above effective, try adding cryolite.

If voltage gradually rises at constant current, add, in order named, alumina, mixture, or cryolite, whichever brings the voltage down.

Keep crust broken on at least one side of electrode. Crushed coke around the electrode will tend to keep crust soft.

Take voltage and current readings every five minutes.

The results of the tests given in the paper show that a satisfactory laboratory run can be made upon aluminium, following instructions as given above. The best results are to be expected under the following conditions:

Mix: 85 per cent natural cryolite; 15 per cent alumina.

Current Density: Approximately 3 amperes per sq. cm. of anode surface.

The anode effect appears to consist of two kinds:

(a) Gaseous envelope which opens the circuit. This can be remedied by opening and closing the circuit or tapping the electrode.

(b) Non-wetting of the electrode, thereby causing numerous small arcs to form from electrode to electrolyte. This causes the voltage necessary to be increased, and appears to be remedied by addition of alumina or cryolite, although it is present in some degree during the entire run.

In the hands of students, using the apparatus of the author, and following the instructions as closely as students usually do, an ampere efficiency of about 60 to 70 per cent may be expected.

Dr. Richardson's paper brought out quite an extended discussion. Dr. Jos. W. Richards stated that the value of 2.8 volts, given in the paper as the decomposition voltage, refers to the decomposition of the alumina, but the cell is to be credited with the voltage due to the oxidation of the carbon. Thus the theoretical voltage comes out as 2.25 volts. This agrees with practical experience. The energy efficiency of Dr. Richardson's tests is then less than 26 per cent. This agrees with the energy efficiency in practice, where it is 25 per cent. 75 per cent of the energy is consumed in keeping the bath at the right temperature. This fact emphasizes the importance of reducing heat conduction and radiation losses.

Dr. Richardson further pointed out that a high current efficiency of over 90 per cent is possible only with low temperatures. In practice a current efficiency between 75 and 80 per cent is attained, so that there is a loss of 25 to 20 per cent due to redissolving of aluminium in the fused bath.

Mr. A. H. Cowles doubted whether there is a redissolving of aluminium, except when there is a voltatic couple formed by the carbon walls and the aluminium.

Dr. Richardson replied that the dissolving of the aluminium was not due to a voltatic couple, but was a case of simple solution. Whenever a fused salt is electrolyzed there is a tendency for the metal set free to dissolve back into the fused bath, and this tendency is the greater the higher the temperature. If it was possible to work with the aluminium bath near the freezing point of aluminium, nearly 100 per cent efficiency could be obtained. But there is a practical reason, due to specific gravities, which prevents this. If the salt is cooled near to the melting point the aluminium floats on the bath.

There was some further discussion, especially on the analogous case of sodium, in which Mr. Cowles, Dr. Walker and Mr. Lidbury participated.

Electrolytic Meter.

A paper by Prof. Frank C. Mathers and Albert F. O. Germann, of the University of Indiana, dealt with a mercurous perchlorate electrolytic meter. Silver and copper are the metals generally deposited in coulometers or voltameters, but mercury has two great advantages for this purpose: it is liquid and has a very high electrochemical equivalent. The difficulty seems to have been in the past with the electrolyte to be employed.

The authors recommend very strongly mercurous perchlorate and give an account of their experiments with this electrolyte. They describe the preparation of the materials, conductivity experiments, and coulometer experiments, and sum up their results as follows:

The advantages of mercurous perchlorate solution as an electrolyte for meters and coulometers are:

The ClO_4 radical is exceptionally stable and is absolutely unchanged by electrolysis.

Mercurous perchlorate is very soluble in water, and the solutions thus formed are entirely free from any decomposition.

Mercurous perchlorate solutions containing free perchloric acid have very high conductivity, such that a very high current density may be used in electrolysis.

The amount of mercury deposited may be very rapidly determined by measurement rather than by the much slower method of weighing.

The deposited metal is in such form that it may be transferred to the anode for further use without any treatment whatever.

Equivalent quantities of electricity deposit from two to six times greater weight of mercury(ous) than of the other metals. This gives greater accuracy.

The paper was discussed by Mr. Schluederberg, who spoke of the reasons which have restricted the use of electrolytic meters in commercial practice. In this country they are not used at all, but in Europe to a certain extent in small direct-current installations, their chief advantage for small installations being their cheapness. Mr. Hering and Dr. Sharp also participated in the discussion.

Electric Discharge Through Gases.

Dr. Louis A. Parsons, of Johns Hopkins University, presented a paper giving an interesting and concise summary of the experimental work which has been done during the last few years by many experimenters on electric discharges in gases at reduced pressure.

He described the phenomena which have been experimentally established, and summarized the conclusions which can be drawn from the experimental evidence on the nature of electricity and matter and especially on the fundamental units of negative and positive electricity.

The paper was discussed by Dr. Whitney and Dr. Walker, who emphasized how much could be done by research work in universities and colleges along scientific lines.

Electrolytic Transfer.

A paper by Prof. Jos. W. Richards, of Lehigh University, discussed the mechanism of electrolytic transfer with special reference to the phenomena in fused salts. The author emphasized the frequent deviations of a chemical compound from the simple or fixed atomic proportions. For instance, if iron pyrites FeS_2 is melted, sulphur leaves it and at a low temperature one has Fe_4S_5, Fe_5S_6, Fe_7S_8, or various approximations to FeS.

By an increase of temperature one passes by insensible gradations all the way to Fe_2S, and at electric furnace temperatures perhaps beyond this.

A chemical compound in absolutely atomic weight proportions is practically unknown in the whole class of melted mattes. In general a rather extensive study of fused salts and compounds leads him to regard fixed atomic weight prooprtions as the exception rather than the rule, in practical work.

If a tube full of smelted PbS is electrolyzed, lead is set free at one end and sulphur at the other. Now, Dr. Richards argues, the deficit of lead at the cathode is quickly distributed by diffusion throughout the whole electrolyte, while the deficit of sulphur at the anode is similarly rapidly distributed throughout the electrolyte. This distribution of the two deficits at the two electrodes is made possible by the known solubility of such intermediate products in the normal salt, and amounts to a virtual transfer of the two constituents from the whole mass of the electrolyte to the electrodes.

According to this view, the transfer in electrolysis is essentially solubility and diffusion of these slightly abnormal compounds formed by electrochemical action at the two electrodes. Dr. Richards therefore claims that the electrolytic transfer occurs as the result of electrolysis and is not the preexisting mechanism of electrolysis.

In the discussion which followed Dr. Parsons suggested that to investigate the nature and mechanism of electricity one should begin with the simplest case, the transfer of electricity through gases, where the phenomena are already complicated enough, although much less in solutions.

Presidential Address.

In the evening of Friday a joint session with the New York section of the American Chemical Society was held, at which the retiring president, Dr. William H. Walker, of the Massachusetts Institute of Technology, presented his address on "Chemical Research and Industrial Progress."

Within the last few years research has become a word to conjure with. The new field of human endeavor epitomized by the word research represents a change in the method of finding truth rather than the intensity of desire to know the truth. It is understood now that a search for truth can only reach its goal through intimate contact with the things themselves, and an accurate determination of the facts concerning them.

In former ages the dictum of Aristotle that "industrial work

W. H. WALKER, RETIRING PRESIDENT AMERICAN ELECTROCHEMICAL SOCIETY.

tends to lower the standard of thought" prevailed, and it is to this want of sympathy that we must ascribe the fact that the old historians failed to note the discovery of even the most important early chemical processes, while they gave detailed accounts of those men who advanced mere speculations and taught untenable opinions on the constitution of the universe.

To find the philosopher's stone was the goal of the alchemists; their labors were not a search for the truth, but for gold. It may be a long step from the work of the alchemists to modern industry. But there may be something in common between the work of the ancient investigators and their influence on civilization, and chemical research of to-day and industrial progress.

The alchemists did not make material progress in their methods in centuries. The explanation of this is to be found in the secrecy which they at all times maintained. There was no accumulation of knowledge or experience and each succeeding investigator continued to grope around in darkness.

The change came in the thirteenth century with Roger Bacon who, in his treatise, showed that phenomena which had been attributed to supernatural agencies were in fact due to common and natural causes. He is to be regarded as the intellectual originator of experimental research. Just as knowledge was sought after for its own sake, and in proportion as there was free and honest intercourse among the investigators of the time, just so rapidly was real progress made.

But for many years after the broad generalizations upon which modern chemistry is founded were well established industry did not profit much by scientific work. One hundred years ago the men who melted the iron and copper, the lead and zinc knew little of the principles underlying their practice. Leather was tanned, woolens, cottons and silks were dyed, porcelain and glass were made without the aid of those who alone knew the chemistry involved.

It is not that scientific men took no interest in the manufacturing industries, for we can recall the great work of Liebig for agriculture, and the immense amount of analytical chemistry which is the foundation of industrial chemical practice, but these were times when the advance in chemical knowledge was far ahead of the industries on the success of which our material comforts depend.

A real attempt to apply chemical knowledge and methods commenced about 1850. It was in 1856 that Perkins made the first synthesis of a coal-tar color, and founded the industry which has become the most remarkable example of applied chemistry that we have. In 1855 Bessemer introduced his revolutionary process for making steel, made possible by the clear understanding of the nature of steel through improved analytical processes.

The last fifty years may be said to be characterized by the production of men who combined the ability to appreciate and enjoy work in science for itself alone, but also possessed the ability and inclination to apply their chemical knowledge and training, and to make results of past generations of pure scientists of ever increasing service to humanity.

But within the last ten or fifteen years we have seen a third kind of chemical activity develop, namely, a class of men who while possessed of the ability and love of science which characterized the leaders in pure science of old, yet are not handicapped by the doctrine of Aristotle that contact with industry contaminates thought. This movement is seen in the tendency of great industrial organizations to establish research laboratories within themselves, and in the willingness of educational institutions to maintain research work in these fields of chemistry which are immediately applicable for industrial practice.

For the purpose of further studying the relationship existing between chemical research and industrial progress, Dr. Walker divided this kind of chemical activity into three classes. First, there is that which for want of a better term we will call original work in pure chemistry; second, there is the work of the so-called industrial chemist, the man who primarily applies existing chemical knowledge to the accomplishment of specific ends, and third, there is research work in what again for want of a better name he calls applied chemistry.

The environment of pure science has in the past been academic; the great investigators were teachers. This is so in America as in Europe. But there is a very different attitude shown toward men of science here and in Germany, for example.

In Germany everybody trusts science, even the Government. The Germans are accustomed to consult a scientific expert before going into a new business. That is the reason why; as Oswald says, Germany has managed to put more brains into her goods and has assumed its present position in the world.

Our American industries have flourished very well, it is true, but rather in spite of a lack of scientific aid, than on account of such aid. We have grown vastly satisfied with our achievement, but we might pause now in our self-congratulation long enough to inquire whether the things we are doing cannot be better done.

There is a mutual obligation existing between our educational institutions and our industries. The former must continue to increase their facilities for the research, which has made the German universities the avenues through which German civilization and industry have been brought to the point that we find them to-day.

On the other hand, the industries should not fail to recognize that progress based alone on industrial prosperity is but apparent progress, and that a sound civilization depends not only on conditions which make for material comfort, but on the culture which comes from an education in the broad sense that scientific research implies.

The great tendency of our times is toward service. This is seen in every sphere of human activity.

There is with scientific men a general awakening to the fact that the highest destiny of science is not to accumulate the truths of nature in a form that no one else but the select few can utilize, but that the search for truth can be combined with a judicious attempt to make the truth serve the public good. Thus the distinction which has existed between the terms pure science and applied science is rapidly falling away. An attempt to define these two kinds of science reveals the fact that this distinction is a general impression rather than a clear statement.

According to a fundamental law of psychology thought tends to pass over into action. Applied science is nothing more than the realization of this, and is thought in action. Force does work only when in motion, so are ideas of value only when carried into effect.

But the carrying of an idea into practice is not always an easy matter. It is frequently much easier to make a discovery or to develop a new fact than it is to make of such a discovery a serviceable reality. For example, the reactions underlying the ammonia-soda process were well known as scientific facts for many years, but this knowledge did not benefit the world until the genius of Solvay made through it purer and cheaper soda available. Cavendish long ago discovered that an electric spark produced nitric acid in the air, the world waited until but a few years ago in order to profit by his knowledge. It was then that the researches of Birkeland and Eyde made of the idea an industrial process.

Men who can interpret the scientific results already available have been of incalculable value in the growth of our industries, and there will ever be a field for this type of chemical activity. In fact it is just here that we find the faith of the Germans in science bearing fruit. In America there is a national lag in the application of new scientific data to every-day problems. We as a people are so wonderfully keen in developing mechanical ideas when once they are presented, that the marked lag in the acceptance and application of chemical principles is remarkable.

The industries themselves are frequently to blame for disappointing results which sometimes are met in an attempt to introduce scientific methods into their works. They employ a so-called chemist, without inquiry as to whether he has had the kind of training that could be expected to fit him for the work he is expected to do. They furnish him with a meagre equipment and then expect revolutionary results. When these are not forthcoming they exclaim in disgust: "There is no money in chemical control, or chemical research, we have tried it."

But there are signs of a change. The most distinct one is the impatience shown by some of the more enterprising manufacturing concerns, to wait for scientific facts to be discovered by others, and their willingness to establish research laboratories within their own organizations; to actively enter the field of research in applied chemistry.

How does research in "applied chemistry" differ from research in "pure chemistry"? Dr. Walker made the distinction clear by an analogy. The development of research in a science may be compared to the exploration of a new country. New roads are to be laid out, tunnels bored and bridges built, in other words, new problems solved. This may be done in two ways. First, constructive work may be undertaken wherever an interesting problem presents itself, without regard as to whether there is a demand for such structure or not. As a rule, the builder is under no great pressure to get the structure com-

pleted. This may represent the method of pure chemistry, and the great advance in chemical knowledge of the past was made almost entirely by boring just such tunnels and building just such bridges. The industries have used these structures when they could, or when some second builder could adapt them to use.

Research in applied chemistry differs from that just described only in this or it needs differ only in this, that when a problem is to be solved, a bridge to be built, the work is undertaken at a point where there is a demand for its use; where people are waiting to cross over so soon as it is finished. The method of building is no different, the difficulties no less. The fact that the bridge is to be used makes the work of the building no less dignified, nor is it possessed of less pleasure.

Why is there not the respect for this kind of work as when a bridge is built with the knowledge that it may not be used for an indefinite period? Why does an eminent writer a few months ago lament the fact that there is not more research "uncontaminated with the worship of usefulness"? Why does usefulness contaminate? Dr. Walker thinks it lies in this. The investigator of pure science works in the broad daylight, throws his product open for inspection, and invites all to come and use it when they can. In applied chemical research the spirit of the alchemist tends to creep in. The builder keeps his materials of construction and his designs a secret, and so boards up his bridge that those who cross over it cannot see how it is built, nor profit by his experience. The moment a thing becomes useful we become jealous of its possession; we become narrow in our horizon; we sell our scientific birthright for a mess of pottage; we become alchemists.

There is a heavy moral obligation on the part of large industrial organizations having fully equipped research laboratories to contribute their share to the advance of the world's knowledge. An obligation to see to it that they do not become saturated with the spirit of alchemy. They have well-stocked libraries and are provided with all the current periodicals; they profit by all the scientific work which has been done and is being done. This is as it should be, and such firms are to be commended for their progressiveness. But is there not a reason why such laboratories should do their part in adding to the sum of available knowledge? There is in every laboratory much work which could be published while the interests of the corporation are conserved.

Germany is very justly held up as a shining example of marvelous industrial progress and prosperity. A very great deal of the credit for her present position is due to her splendid educational system. But no small factor in her national progress is the helpful attitude which her industrial organizations take toward the publicity of scientific data. The individual does not suffer, while Germany both from a scientific and an industrial standpoint is rapidly advanced.

But too often with us the president and his board of directors are alchemists; they fail to see why if they pay the salaries of the research men, they should give the public or their competitors any part of their results. They exclaim "what has posterity done for us"? They would have their laboratories remain the secret chambers of the alchemists, and continue to improve their methods of changing baser materials into gold, without regard to the obligations which they owe their fellows.

It is to the men who form the working force in our industrial laboratories that we must in a great measure look for making our national scientific societies the power for industrial progress which they ought to be. But it is this general disinclination on the part of industrial concerns to allow their chemists to disclose in any measure the results of their work by contributing papers for the meetings or in entering heartily into the discussions that makes this realization difficult.

This is to be deeply lamented and we believe it is the fundamental mistake; a short-sighted policy which can but react upon industrial progress as a whole. We cannot operate a scientific laboratory as we would do a factory. The conditions of maximum productiveness are not found in an atmosphere of selfish rivalry.

It requires no extensive mathematical calculation to prove that the manufacturers themselves would be the ones to profit by more liberal treatment of the results of scientific work.

Of one hundred manufacturing concerns each one would give but 1 per cent of the whole contribution, while he would receive the remaining 99 per cent. He could not in the long run be the loser. But of vastly more importance he would feel and know that his organization was taking part in a world movement toward that increase of human knowledge upon which all real progress depends. The greater sense of satisfaction, the greater success even of an industrial organization lies in a fuller, freer, more generous publicity of the scientific results of their laboratories.

Would that we might benefit by the experience of Solomon, King of Israel, when asked "What shall I give unto thee?" replied "Give me knowledge and wisdom," and he answered, "Wisdom and knowledge are granted unto thee, and I will give thee riches and wealth and honor."

Surface Action and Electrochemistry.

Dr. Harrison E. Patten, of the Bureau of Soils, U. S. Department of Agriculture, Washington, D. C., then lectured on the subject of the relation of surface action to electrochemistry. In a brief introduction he outlined the general form of treatment as follows: First, a definition of surface tension as the first derivative of surface energy with respect to surface change; second, a consideration of the general equations of surface changes following the treatment given by J. Willard Gibbs; third, an extension of these equations with the assistance of a cyclic process to surfaces having an electric charge; fourth, the application of the general equation developed in the third section to electrocapillary phenomena, to migration of suspended particles, and to electric osmose; fifth, the relation of absorption phenomena to electrocapillary phenomena, to settling of suspensions, to formation of deposits, and to the behavior of disperse systems in general.

The lecture was illustrated with slides, so that a concise survey of the numerous mathematical and physical factors entering in was given in a short space of time. The lecture will be printed in full in the Transactions.

Electric Iron and Steel Furnaces.

At the Saturday morning session two papers on the electrometallurgy of iron and steel, previously presented before the Chicago section of the society, were called up for discussion. The first was a paper by Mr. James Lyman on "The Electric Furnace for the Manufacture of Iron and Steel," describing the Frick electric reduction arc furnace for smelting iron ore, the Frick induction type furnace for refining iron and steel, the Heroult furnace, and induction electric furnaces.

Electric Steel Refining at South Chicago.

The next paper by Mr. C. G. Osborne, of the Illinois Steel Company, of South Chicago, Ill., gave an account of a few experiences with the 15-ton Heroult electric furnace at South Chicago at the Illinois Steel Company's works of the U. S. Steel Corporation.

The paper first gives a description of the furnace. This particular furnace was described and illustrated in some detail in our Vol. VIII, page 179. Further details are here given from Mr. Osborne's paper.

The furnace shell is of plate steel, 1 in. (2.5 cm.) in thickness, riveted together. The outside horizontal cross-section plan is approximately that of a complete circle of 13½ ft. (4 meters) in diameter, with two flattened portions situated at the front and back respectively.

On the bottom of the furnace, within the 1-in. plate, and next to it, one row of magnesite brick, laid the 4½ in. (11.5 cm.) way, is placed across the flat portion.

The side walls of the furnace are vertical and consist of two rows of magnesite brick, laid the 9-in. (22.5 cm.) way, giving

a thickness of 18 in. (45 cm.) of magnesite brick. These solid magnesite brick walls extend up to the furnace roof.

The bottom "proper" of the furnace consists of dead-burned Spaeter magnesite to a depth of 12 in. (30 cm.) at its thinnest point, which is, of course, at the extreme center. From this thinnest point the bottom slopes gradually upward so as to form a portion of a sphere 7 ft. 2 in. (2.15 meters) in radius.

This bottom was put in in the following manner: Dead-burned and carefully ground Spaeter magnesite was mixed with basic open-hearth slag in the proportion of four of magnesite to one of open-hearth slag. To this mixture, sufficient tar was added to make the mass sufficiently plastic to enable it to be tamped into the furnace in the usual manner. The entire depth of the bottom was tamped in this way. Next, the furnace was filled with wood, dried out for about 48 hours, and then filled with coke, and the electrodes lowered and the current turned on. In this way the bottom was fluxed into place.

The furnace roof is of silica brick 12 in. (30 cm.) in thickness.

The roof is made up on a movable ring. This ring is fitted with a top and bottom angle iron to take a skew-back brick, and then from this the arch is spanned across the 10-ft. (3 meters) interior of the furnace with an 8-in. (20 cm.) rise. The bricks are set in circles parallel to the steel ring, the usual wooden wedges being placed here and there to take care of the subsequent expansion of the brick. Holes for the electrodes are left in the roof by means of templates placed in position. The bricks are held in position around these holes by their lateral pressure.

There are five doors, two on each side of the furnace, and one in front over the pouring spout. These doors are of cast iron, lined with clay brick, laid the 4½-in. (11.5 cm.) way. They work in the usual groove arrangement, and are operated by steam pressure of about 150 pounds (11 kg. per sq. cm.). The front door over the pouring spout is an exception to this, being operated by hand with a counter balance.

The three electrodes are let down through the roof in the form of an equilateral triangle, each side of which is 5 ft. 2 in. (1.55 meters) in length, the apex of this triangle pointing directly east, that is, toward the back of the furnace. The center of this triangle coincides with the center of the furnace roof.

There are, of course, three separate holders, one for each electrode. Each holder is constructed of a solid copper casting bolted directly to the bus-bar. In front, these holders are split and joined with a right and left screw, which enables the holder to be opened or closed at will. The holders are designed to carry a 24-in. (60 cm.) electrode, but by means of contact blocks to fill, any smaller-sized electrode can be employed.

The weight of the electrode is supported by the chains. These chains run back over pulleys to the drums at the back of the furnace. These electrodes are kept in alignment by vertical guides.

The electrodes are regulated by individual motors placed at the back of the furnace. These motors are attached by gears to the drums. The regulation is by hand, by controllers, or by an automatic device.

The operating platfrom of the furnace is about 9 ft. (2.7 m.) from the level ground. Around the furnace on this platform, at convenient points, bins are placed for the miscellaneous materials used in furnace operation.

The front part of the furnace platfrom opens up to allow a ladle to be hung in position when the furnace is tapped and for any miscellaneous work in the pit.

The power for the furnace is generated by dynamos having as prime movers reciprocating gas engines, reciprocating steam engines, high-pressure and low-pressure turbines. It is three-phase, 2200 volts and 25 cycle.

At the electric furnace it is stepped down by means of three 750-kw transformers to the voltage of the furnace. These transformers are so arranged with switches that the primary turns may be altered to give a secondary voltage of 80, 90, 100 110 volts as desired. Ordinarily 90 volts is used.

The entire building is spanned by a 50-ton crane.

The pouring platfrom is situated to the south of the furnace. It is 30 ft. (9 meters) long, and enables eight molds to be placed in position for pouring.

The normal operation of the furnace is as follows:

Ordinary Bessemer pig iron is full blown in a 15-ton Bessemer converter, in from eight to twelve minutes. It is then poured directly from the Bessemer vessel to an electric furnace transfer ladle, and drawn by locomotive to the electric furnace building, a distance of about ¼ mile (400 meters), in about five minutes. As a precaution against the possible formation of a skull in the ladle, the Bessemer charge is blown about 1500 lb. (680 kg) of scrap "hotter" than ordinary Bessemer practice.

Immediately the ladle is received at the electric furnace it is picked up by the crane, slightly tilted, and the silicious slag is completely cleaned off by hand-rabbling. The metal is now ready for charging. To do this, the ladle is merely turned over on its trunions and the metal poured into a spout, through which it rushes to the furnace. This operation of cleaning off the slag and charging occupies from five to ten minutes.

The electrodes are lowered and the current turned on.

As the metal is being poured into the furnace, the helpers shovel iron oxide and lime into the furnace through the working doors.

In this way a basic oxidizing slag is produced which serves to remove the phosphorus. In about thirty minutes this slag has served its purpose, and the furnace is tilted slightly forward and the slag removed in from five to ten minutes by hand-rabbling.

The recarburizer is added at this point. On the bare surface of the oxidized metal, lime is quickly added, with sufficient fluor-spar to keep the mass fluid. In about fifteen minutes this lime is melted, and finely divided coke dust is now thrown onto the top of the slag beneath each of the three electrodes.

Under the influence of the arc, calcium carbide is produced in gradually increasing quantities.

As soon as this state of affairs is reached, a neutral if not actually reducing atmosphere has been obtained. From here to the finish there is practically a dead-melt in a reducing atmosphere. The slag at this stage of the process is fluid and highly basic. If a sample should be taken and water added to it, the resultant acetylene gas, from the well-known calcium carbide and water reaction, is of sufficient quantity to light and burn for half a minute.

Tests are now taken to show the condition of the steel. A small cylindrical test piece is poured and forged to a round pancake-shaped object under a steam hammer located at the furnace. If this forged sample shows by its appearance a satisfactory condition of the metal, the bath is tapped. If not, further refining is necessary.

To tap the furnace the electrodes are raised from the bath and the ladle swung by a crane under the pouring spout and the tilting level pulled forward.

The pouring is done through a 1½-in. (4-cm) nozzle to molds of varying sizes.

It takes an hour and a half to two hours to a heat, according to the grade of steel produced.

What is actually done in the electric furnace at the South works is to take oxidized blown metal of an approximate analysis of

C	S	P	Mn	Si
0.05-0.10	0.035-0.070	0.095	0.05-0.10	0.005-0.015

and produce deoxidized steel low in sulphur and phosphorus and, within reasonable limits, of practically any analysis required by the consumer.

The electric furnace at South Chicago has operated on a greater variety of product than any furnace in the world, and with bewildering intermingling of high-grade alloy steels, high-grade carbon steels and ordinary carbon steel. They have made

ordinary carbon steel rails of a dozen different sections, billets of all sizes and grades, plates of all sizes and grades, structural shapes, castings, small and large, high carbon and low, and forgings of all sizes; of alloy steels they have made nickel, nickel-chrome, chrome, manganese and silicon steels.

They have made a number of heats from cold materials. In several of these they took the worst scrap they could pick up—stove-plate scrap. With this stock an axle heat was made which showed excellent physical tests. Seventy-seven blows from a 1640-lb. (750-kg) hammer falling 43 ft. (13 m) broke the first axle, and 68 similar blows the second.

The main chemical or physical characteristics in electric steel seem to be these:

A comparative freedom from oxidation.

A comparative freedom from segregation.

A higher tensile strength and slightly higher ductility for the same chemical analysis up to about 0.40 carbon, where the difference becomes less apparent.

A steel of greater density than other commercial steels, with the possible exception of crucible.

In conclusion, Mr. Osborne said that the three electric furnaces of the Steel Corporation are the largest in the world that have as yet been put in operation. They have presented many interesting and complex problems, and the manufacturer has encountered occasional troubles. "Although we feel that we are as yet but on the threshold of this method of manufacture, we feel that electric steel is a success and its future assured."

Mr. Osborne's paper brought out considerable discusion. Mr. Osborne stated that the South Chicago furnace was the oldest of the Héroult electric furnaces of the Steel Corporation and that in the newer one in Worcester several improvements have been made, especially in the arrangement of the electrodes. While it takes five minutes in Worcester to change electrodes, it takes half an hour in South Chicago. Other disadvantages at the South Chicago plant are the relatively small space available and the distance of the electric furnace from the Bessemer converter house, which makes it necessary to blow the steel 200° or 300° hotter than ordinary Bessemer metal. This accounts for the fact that the metal comes colder out of the electric furnace than it goes in. They have used all kinds of electrodes; at present they are using 14-in. amorphous carbon electrodes of the National Carbon Company.

In reply to a question by Mr. Hansen, Mr. Osborne stated that the increased tensile strength of the electric steel is due to greater density. The greater density is due to absence of gases. Electric steel, properly made, has segregation practically nil, except right below the pipe.

Mr. Saunders considered it the function of the electric furnace to do only the most difficult work, to supply only the peak of the temperature, and referred to a furnace of the National Carbon Company, which is chiefly heated by gas and only the last 200° or 300° are supplied by electricity.

Mr. Hansen disagreed with respect to the use of electric furnace for peak work only; he considered this as uneconomical, since the overhead charges of the electric plant go on all the time.

Dr. Richards referred to the use of the electric furnace as a dead melter without any refining. Simply by holding the finished steel in an electric furnace, say, for 30 minutes, to let it rest and let the oxides and gases rise and get out, it is possible to improve the quality of the steel considerably.

As in the South Chicago practice the steel goes hotter into the electric furnace than it comes out of it for the local reasons stated above it is evident that the advantage of the electric furnace in this case is not thermal efficiency. This brought up again a discussion on the efficiency of electric furnaces.

Dr. Richards argued that two kinds of efficiency must be distinguished. First, there is the case where the furnace is used as a melting furnace; electricity is then used to raise the temperature. Second, there is the case where the metal is simply kept molten at a materially constant temperature; in this case the efficiency must be defined differently from the first case. Both cases must be clearly distinguished.

Mr. Langmuir thought that in the second case it was hardly possible to speak of efficiency because the use of this term presupposes that a 100 per cent efficiency is imaginable.

Dr. Lidbury expressed the opinion that there was nothing to be gained at all by speaking of the "thermal efficiency" of an electric furnace, because in order to express it in percents it is necessary to make so many arbitrary assumptions and hypotheses on account of a great many factors which cannot be measured. Instead of speaking of efficiency he preferred a straight statement that so and so many kw-hours are required per ton of metal produced, etc., as such specific statements would give all the information needed.

Dr. Richards thought that the use of the term "efficiency" was proper even in the case where it is simply a question of keeping a furnace charge continually at the same temperature, because one furnace does that with 50 kw per ton of charge and another with 200 kw.

Dr. Lidbury thought that such comparative specific statements as the one just mentioned gave all the needed information without bringing in "efficiency" at all.

Mr. Osborne stated that it was a matter of experience that with 14 practically identical open-hearth furnaces, as at South Chicago, the pounds of coal per ton of output might vary considerably, and that the figure depended essentially on the management. Hence the "thermal efficiency" would be in practice largely a question of management, of the hustling of the attendants, etc.

Electrolytic Furnace Process for Production of Pure Metals.

Mr. John Woods Beckman, of Niagara Falls South, Canada, presented a paper in which he expressed the opinion that the fused electrolyte would be the electrolyte of the future. He referred to the production of aluminium from alumina and asked whether it should not be possible to produce all metals in a similar manner from their oxides; but what electrolyte is to be used?

A comparatively little known group of salts are those occasionally occuring in nature, composed of equivalent amounts of calcium oxide and the oxide of any metal less positive than calcium. These salts occur occasionally in nature comparatively pure, as is the case with Scheelite, and can very readily be produced by heating the oxides together in any suitable manner.

There is a great number of these salts possible, and the characteristic feature of them is that they all fuse at about and over 1000° C., the temperature varying according to the metal oxide which calcium oxide is combined with. The boiling point of these salts is very high.

The valuable feature of these salts is that when in a molten condition they carry the current very readily, and are very suitable as molten electrolytes for the production of a great number of metals.

Mr. Beckman stated that he had produced in this way manganese, iron, chromium, and a 50 per cent alloy of vanadium.

To describe the principle of the process Mr. Beckman described the production of iron. Equivalent amounts of Fe_2O_3 and CaO are mixed together and heated in any suitable manner. The salt $CaO.Fe_2O_3$ is formed and fused in a receptacle fitted for electrolytic purposes. Oxide of iron is introduced into this fused bath, and this molten solution is electrolyzed, iron being deposited on the cathode and a lively oxygen development taking place on the anode.

If the electrolyte itself is submitted to the direct current, no noticeable reaction takes place, but the moment iron oxide is added the electrolysis starts, and by adding from time to time more oxide it can be carried on continuously.

The practical features of this method are: First, cost of electrolyte practically negligible, as it is composed of calcium oxide and the metal oxide of the desired metal; second, a great quantity of metal can be produced out of the same electrolyte;

third, in case the electrolyte becomes fouled the loss in discarding the electrolyte is practically negligible; fourth, the flexibility of the process making it possible to produce a great number of metals and their alloys, such as iron, manganese, chromium, zinc, tin, titanium, vanadium, tungsten, tantalum, molybdenum, nickel, cobalt, lead and a number of others more or less rare.

Mr. Beckman finally gave the following list of the theoretical amount of meal deposited per ampere hour, considering only ideal conditions:

	Oxide	Gram per amp. hr.
Aluminium	Al_2O_3	0.3377
Lead	PbO	3.868
Chromium	Cr_2O_3	0.655
Iron	Fe_2O_3	0.696
Manganese	MnO	1.027
Nickel	Ni_2O_3	0.732
Zinc	ZnO	1.220
Titanium	TiO_2	0.445
Vanadium	V_2O_3	0.64

There was considerable discussion of the paper, in which Messrs. Hansen, Bleecker, Tucker, Walker, Richards and Saunders participated. Mr. Hansen and Professor Tucker expressed the opinion that there was no experimental evidence that the metals produced by Mr. Beckman were the result of electrolysis, since on the small scale employed there might have been simple high-temperature reduction by carbon from the graphite crucible.

An Electric Converter Process.

Mr. ALBERT E. GREENE, of the American Electric Smelting & Engineering Company of Chicago, presented a paper on "Electric Steel Processes as Competitors of the Bessemer and the Open Hearth." This is a new development, as heretofore electric steel processes have been used largely for the production of high-grade steel, practically of crucible-steel quality. Mr. Greene urges, however, that there is a distinct possibility

FIG. 1.—ELECTRIC CONVERTER.

for an electric-furnace process for ordinary grades of steel in competition with converter and open-hearth on account of the unavoidable great losses of metal in the two latter processes.

In the conversion of pig iron into steel by the Bessemer process there is a loss, which approximates 4 or 4.5 per cent of the total weight of the charge in iron and practically all of the manganese, which latter approximates ½ per cent, and making a total unavoidable loss of about 5 per cent of the weight of the charge in these two metals. For the open-hearth process some authorities give the amount of iron lost in open-hearth slag as 2.5 per cent of the weight of steel made, and the loss of manganese, 0.5 per cent.

It is an interesting fact that much of our iron ore contains the elements manganese and silicon in sufficient quality to meet the specifications of ordinary steel, and moreover, these elements are reduced in the blast furnace and are almost invariably present in the pig iron in sufficient quantity to meet

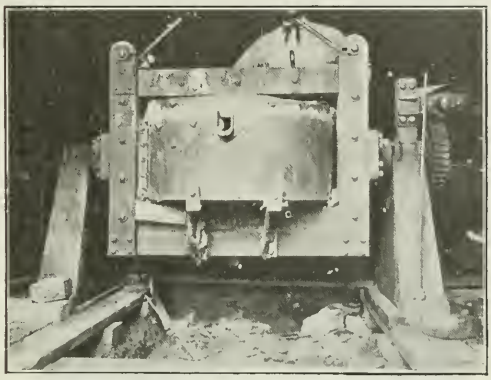


FIG. 2.—ELECTRIC CONVERTER.

requirements without any additions, were it possible to save them in the converter; but during the converter blow the silicon and manganese are practically all eliminated and then these alloying metals are put back again by charging into the steel some expensive alloy additions. It costs to reduce these elements in the blast furnace, and it costs to replace them again after they are oxidized. The value of the manganese so lost is quite important.

The object of Mr. Greene's paper is to point out how the electric furnace in combination with a control of chemical reactions by regulating the gaseous agents, provides a means of producing metal of higher grade and of doing it more efficiently and with far less loss than is possible in Bessemer or in open-hearth furnaces, and to describe a new process which takes advantage of a combined control of temperature and gaseous reagents to accomplish such a saving. The chief advantage of the electric furnace does not consist simply in its ability to produce extremely high temperatures; but rather in the fact that it provides a means of maintaining a charge at any given temperature, thereby enabling one to control the composition of the atmosphere at will.

In this connection Mr. Greene discussed the meaning of the term "neutral," as commonly applied to the atmosphere within furnaces in general, including electric furnaces. "Neutral" means inactive. But the atmosphere, if of proper composition, may react in such a way as to greatly aid a metallurgical process; for instance, the atmosphere in a steel-producing furnace might be reducing toward iron oxide and prevent the steel from oxidizing; or it might be oxidizing toward carbon or phosphorus, and thereby eliminate them. Now, it is a most interesting fact that we can have an atmosphere which is reducing with respect to iron oxide and nevertheless oxidizing toward carbon or toward phosphorus. For this reason it seems that the term "neutral" and also the term "non-oxidizing" is very indefinite, where there are different elements present, unless it is stated with respect to what the atmosphere is neutral or non-oxidizing.

In view of the fact that it is possible to have an atmosphere which simultaneously is reducing toward iron oxide and oxidizing toward carbon, the author sketches the following theoretical process for oxidizing carbon out of iron without oxidizing the metal itself. If molten pig iron be placed in a closed crucible,

made of magnesite for instance, and kept at a uniform high temperature, and the space above the iron in the closed crucible filled with carbon dioxide gas, the gas and iron will react and finally come to a state of equilibrium. The reaction might be represented as follows: $CO_2 + Fe = FeO + CO$. Under the equilibrium conditions which would be reached the ratio of the amounts of the two gases present would have a given value for the particular temperature.

If we then replace the mixture of these two gases by another mixture containing relatively more carbon monoxide and less carbon dioxide than the equilibrium mixture, and allow this new mixture to react with the iron and the iron oxide, some of the iron oxide would be reduced and the CO_2 component of the gas increased. This replacement could be made continuous by successive introductions of the new mixture. In this way reduction of iron oxide could be carried practically to completion, and moreover, this could be done by means of a gas containing an oxidizing agent for carbon.

This was the theory on which the American Electric Smelting & Engineering Company started in its experiments several years ago to develop a practical process for converting pig iron into

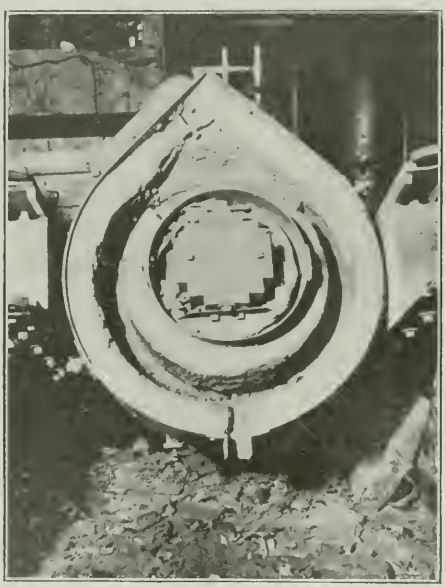


FIG. 3.—ELECTRIC INDUCTION-HEATED CONVERTER WITH COVER REMOVED.

steel and refining it without oxidizing the iron and other valuable constituent metals.

In the preliminary experiments it was found that there was a wide range of composition of mixtures of CO_2 and CO gases suitable to prevent oxidation.

It was found that producer gas might easily be made to serve the purpose, and that a producer gas containing as little as 4 per cent of carbon monoxide and as much as 15 per cent of carbon dioxide would, when blown into molten pig iron at a temperature of about 1450° C., eliminate the carbon with practically no oxidation of iron, manganese or silicon.

After the preliminary experiments which were made in small electrically heated crucibles, the process was carried out in an induction-heated vessel of 300 pounds' capacity.

This furnace had an enlarged portion at the back forming a pot, and tuyeres were put into this part in much the same way as they are put in a Tropenas converter.

The process consists of providing a bath of molten low phosphorus pig iron containing, for example, the usual proportion of manganese, silicon and carbon. The charge preferably contains sufficient manganese and silicon to slightly more than meet the specification for the particular steel to be made. The temperature of the bath is raised to something over 1425° C., and, while it is maintained by the electric heat, a gaseous mixture containing carbon monoxide and carbon dioxide is blown into the metal in much the same way as air is blown into a Tropenas converter.

This mixture may contain 12 to 18 per cent of CO_2 and 5 per cent or more of CO.

The rate of elimination is a little slower than with the Bessemer process for the same rate of blowing. A well-operated, side-blown converter requires from 25 to 35 minutes to make a blow, using about 1500 cu. ft. of gas per minute per ton of metal. In this small induction furnace, using about 50 cu. ft. of gas per minute per 200 lbs., or about one-third the rate of blowing in the side-blown Bessemer vessel just referred to, the carbon can be eliminated in something more than an hour and one-half, depending on how much carbon there is at the start. Thus, with only one-third as much gas blown per minute per unit weight of metal, the time is a little more than three times as long as for a similarly blown vessel using air. By increasing the rate of blowing the time can be very greatly cut down.

In this small furnace pig iron can be converted into steel with as small a loss of iron and manganese and silicon as 2.5 per cent, including losses of metal in charging and spills in handling the steel.

The most convincing evidence in Mr. Greene's opinion, that oxidation does not take place in this process may be found in connection with the loss of manganese. In converting a 13.4 per cent spiegel (containing 13.4 per cent manganese) into manganese steel they have done this without any additions of any kind and have produced a steel containing over 12.5 per cent manganese, and this was without the use of a slag to prevent vaporization. The carbon was reduced from over 4 per cent to 1.20 per cent.

In later tests, using a lime slag, the vaporization loss has been further diminished.

It is well known that the high percentage of carbon present in commercial ferro-manganese makes it practically impossible to obtain as much as 12 to 14 per cent of manganese in manganese steel without raising the carbon content above 0.9 or 1 per cent. No great difficulty is met with in producing very low-carbon manganese steel by the electric converter process, and the properties of low-carbon manganese steel are very interesting.

Mr. Greene finally considered the application of the process to the removal of phosphorus and sulphur and to the production of a high-grade steel from low-grade pig iron in one continuous process.

Mr. Greene states that in the electric converter he can oxidize phosphorus out of iron and hold it out by means of lime and do this with practically no oxidation of iron or manganese. He has done this at pig iron temperatures below 1350° C. without oxidizing carbon, and he has done it at high temperatures, above 1500° and up to 1900° after carbon has been oxidized. And beside these facts he found that the sulphur can be taken up and held in this same slag that holds the phosphorus, which action he attributes to the absence of oxide of iron in the slag.

He has converted a low-grade pig iron into a high-quality steel in one continuous operation in this way, taking out the carbon without oxidation of iron and manganese and then, with a lime slag on top of the metal, by continuing the blow of the gaseous mixture, the phosphorus has been oxidized and eliminated. The phosphorus was found combined with the lime as calcium-phosphate and practically all the sulphur was found in the slag as calcium-sulphide.

He has had such results as follows:

Analysis of charge.....	Phosphorus, .76%	Sulphur, .113%
Analysis after blow.....	" .026	" .040

and also

Analysis of charge.....	Phosphorus, .094	Sulphur, .040
Analysis after blow.....	" .008	" .017

He has carried out such a conversion of molten pig iron into refined molten steel as just outlined in less than thirty minutes, which indicates that the reaction takes place quite rapidly and that the problem of installing the process in large vessels of ten or more tons capacity is a mechanical one of getting the gas into the metal.

Mr. Greene then compared the electric converter process with the Bessemer converter, the open-hearth and the Bessemer converter electric furnace combination processes, and finally said a few words on the type of electric furnace used for his converter process. The induction furnace has shown important advantages. In the operation of this process in large induction-heated furnaces he believes that less than 30 kilowatt-hours per ton of steel will be required.

The substitution of a gas containing carbon monoxide and carbon dioxide for air is believed not to involve as great a change in present methods as at first it may appear to do. Such gas as is needed to carry out this process is available from cupolas or blast furnaces or may be made in a very simple gas producer. Blast furnace gas is preferably modified by mixture with stack gas, since the percentages of carbon monoxide in blast furnace gas is usually much higher than required. The process itself, at least in so far as the elimination of carbon is concerned, is a gas producer process, the off-coming gas being richer in CO and lower in CO_2 than when blown into the metal; and this fact opens the possibility of using the gas over and over by burning a part of it to maintain the desired composition.

The paper was briefly discussed by Dr. Richards and Mr. Hibbert.

The Pinch Phenomenon Employed for Electric Furnace Design.

A paper by Mr. Carl Hering, of Philadelphia, Pa., describes "a new type of electric furnace" in which use is made of the 'pinch phenomenon."

Let the circle, Fig. 1, represent the cross-section of a cylindrical conductor; the electromagnetic forces which constitute the pinch phenomenon tend to act radially as shown by the arrows. If now the conductor be made to be a vertical column of liquid as, for instance, the part marked "resistor" in Fig. 2, consisting of a hole in a non-conducting material closed at the bottom by the electrode and filled with the liquid, then these forces will act horizontally and perpendicularly to the axis along the whole length. They create a suction near the periphery and a pressure at the center.

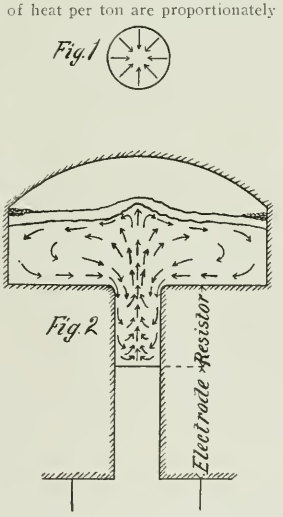
Moreover, by hydraulic action these radial forces in turn produce corresponding longitudinal forces at the center and at the periphery, and as these axial or vertical forces cannot expend themselves at the bottom they do so at the top, thereby producing a veritable fountain, as shown in Fig. 2, the liquid flowing up in the center and down at the periphery about as shown by the arrows; moreover, the direction of this hydraulic flow is independent of the direction of the electric current, hence is the same for direct or alternating currents.

This new phenomenon, which was colloquially named the "squirt phenomenon," is made by Mr. Hering the basis of a liquid resistor furnace. If such a column is made the resistor in which the heat is generated by passing the current through it by means of an electrode at the bottom, the heated liquid will be rapidly expelled and forced to the top against the blanket of slag, while the cooler liquid at the bottom will be sucked down into the hole near the circumference to be in turn heated and immediately expelled with considerable force. With energetic circulation through the resistors the fatal objections to the use of liquid resistors could be overcome.

In practice one would naturally use two holes, one for each electrode, as that reduces the depth of the hole to one-half. The proportions of the hole, that is, the ratio of the section to the length, are, of course, determined by the resistivity of the material, the amount of heat to be generated and the relation of the volts to the amperes; the actual dimensions are therefore still a matter of choice. The ejecting force, or "squirting force," increases with the square of the current and diminishes as the cross-section increases. It will be seen therefore that there is very fortunately some choice left as to the dimensions of these "flowing resistors."

In most materials there is apparently a very rapid rise of resistivity with the temperature, which is very favorable to the proportions of such resistors, as is also the apparently very rapid increase of the specific heat of iron at the higher temperatures.

The advantages of a furnace based on this principle are stated by Mr. Hering as follows: It is ideally simple and cheap, consisting merely of a crucible or hearth with two suitable holes in the bottom plugged with the electrodes. The heat is generated in the material itself, which is just where it is wanted; moreover, it is generated in the most rational place, namely, at the bottom and not on the top surface; it is conveyed and distributed mechanically, so to speak, throughout the melted mass, and this conveyance of heat is immensely more rapid than it would be by mere conduction or by the slow flow due to differences of density; hence a charge is treated and finished rapidly, whereby the output of the furnace per day is increased and the losses of heat per ton are proportionately reduced. As



FIGS. 1 AND 2.—PINCH AND SQUIRT PHENOMENA.

far as is known now there is no temperature limit except that of the failure of the refractory lining, as the pinch effect can no longer limit the temperature.

This very active agitation and circulation of the liquid mix the product rapidly with any of the refining ingredients which may be added and tend to make the product very homogeneous. Every particle of the liquid is in turn, and repeatedly, brought to the surface, where mechanically suspended impurities and gas bubbles are expelled and forced into the slag and where the metal is exposed to the chemical action of the slag. The temperature at the surface of contact of metal and slag, on which temperature the slag action depends, is maintained better when the heat flows up naturally from the hot metal to the cooler slag than when it is forced to flow down unnaturally from the hotter slag to the cooler metal.

Other advantages besides this lie in the fact that the electrodes are very cheap, are not consumed, and as they may be made of

metal they are far more economical in energy loss than carbon or graphite would be. No moving and adjusting of the electrodes are required. The interior of the furnace may be made as small as the quantity of material to be held requires, hence the thermal efficiency of the walls of such a furnace could be made to reach the practical maximum of all types of furnaces. Moreover, as the heat is rapidly distributed mechanically throughout the mass, the maximum temperature need be no higher than that required for the desired metallurgical or chemical action, hence the heat losses, which increase rapidly with the temperature, are correspondingly reduced; in this there is a great advantage over arc furnaces, in which the temperature is necessarily that of the vapor of carbon, hence immensely higher than what is needed.

The regulation is simple, rapid and in fact ideal. When the electrode is made of the same diameter and material as the resistor the furnace tends to become self-regulating as to temperature. When, for instance, the temperature becomes higher the electrode automatically becomes longer and the resistor therefore shorter, whereby less heat will be generated and the reverse when the temperature becomes lower. The place where the electrode ends and the resistor begins is, of course, that section across which no heat flows; the heat generated below this section flows out to the terminal and that generated above it flows into the furnace; it is this plane that automatically moves up or down.

The hottest liquid is that in the center of the resistors and is therefore not in direct contact with the refractory lining.

The friction of liquid against liquid, in the interior of the conductor, is far less than might be supposed, provided the liquid is in a very fluid state. The rapid flow is not along the walls of the hole, but in the center where it causes no wear.

The counter e.m.f. which must necessarily be generated by the movement produced is an advantage as it shortens the resistor.

By using a bath of some suitable metal in the bottom of the furnaces as the prime receiver of the heat, the furnace is adapted to be used to melt non-conductors like glass, or to heat granular ores, reduce and volatilize such metals as zinc and arsenic from their ores, etc. By dissolving carbon in the iron in one part of a bath and the oxide in another part, iron can be reduced from its ores and the CO gas produced can be burned to preheat the ore, whereby the total energy of the carbon may be utilized.

The power factor can, of course, be made very high, probably practical unity, and, unlike in the induction furnace, the frequency need not be abnormally low.

When badly proportioned the columns are apt to pinch off, causing rapid "makes and breaks"; this so greatly reduces the amount of heat which can be generated that there is danger of freezing. Such repeated breaking of the circuit should by all means be avoided. Several times a charge was left to freeze in the furnace over night and yet was readily started the next day from the cold state, thus showing that the resistors do not part in shrinking.

The larger the furnace the better and the less will be the difficulties, because the resistors then become shorter and thicker. There are, of course, many modifications which at once suggest themselves which will be tried in due course. It would be quite simple, for instance, to add gaseous fuel heat, or the heat of an arc, in the upper part; or to add the electrical parts to existing open-hearth or other furnaces and use the electric heat only for the final treatment.

The transformer is best attached directly to the bottom or side of the furnace, as the furnace currents are of course of low voltage, hence large in amperes. The flexible connections to a tilting furnace then consist merely of the two thin primary wires. The furnace is readily adapted to three-phase currents; the only change is that there will be three holes and electrodes instead of two. This is an advantage because for the same current in each electrode it shortens the resistors by one-third. A two-phase current may be readily transformed into one of three phases by means of a Scott transformer.

Under all conditions there should always be reached a stable state of temperature in the resistors, at which the heat generated in them is equal to that carried out of them mechanically; the best conditions must be determined by practice; fortunately there is a considerable range of latitude in the proportions which could be chosen.

Mr. Hering's paper is concluded by a list of some of the objectionable, troublesome or undesirable features of arc, induction and simple resistance furnaces with liquid resistors, and the claim is made that they would all be avoided entirely or at least lessened in his present type of furnace.

Mr. Hering's paper was discussed by Messrs. Hibbert, Clamer, Saunders, Richards, Hitchcock, Kemery, and Queneau. Mr. Hering stated that the tubes in which the squirt phenomenon is produced are made of alundum and Mr. Clamer stated that the wear and tear of the tubes is practically nil.

Iowa and Maine for Electrochemical Industries.

A paper by Mr. Frank P. H. Knight discussed the advantages of Keokuk, Ia., for electrochemical industries, due to the water-power development now going on. The initial installation will be 120,000 hp, which it is expected will be completed in two and one-half years. This will be increased later to 200,000 hp. The author also gave data on commercial advantages, natural resources, transportation facilities by rail and water, and factory sites.

A paper by Mr. C. Vey Holman, State geologist of Maine, discussed the advantages of Maine for electrochemical industries. He pointed out the geographical advantages and the possibilities of water-power developments; he estimated that 2,500,000 hp would be available. The author concluded with reference to the iron ores and other mineral resources of Maine.

After a vote of thanks to all who had made the convention successful, the meeting adjourned.

The next convention will be held in autumn at Toronto, Ontario, Canada.

In the following we give a complete alphabetical list of the names of members and guests in the last printed registry list. Where no city is stated, New York City is meant.

P. O. Ahbe, C. E. Acker, Ossining, N. Y.; A. Allan, Jr.; Mr. and Mrs. G. Alleman, Swarthmore, Pa.; R. Amberg, Pittsburgh, Pa.; F. Andrews, Perth Amboy, N. J.; J. C. Andrews, Ithaca, N. Y.; W. C. Arsem, Schenectady, N. Y.; D. K. Bachofner, Irvington, N. J.; H. R. Badger, Detroit, Mich.; Dr. and Mrs. L. H. Backeland, Yonkers, N. Y.; T. F. Bailey, Alliance, Ohio; J. T. Baker, Phillipsburg, N. J.; W. D. Bancroft, Ithaca, N. Y.; M. Barnet; A. W. Barron, Cleveland, Ohio; Chas. Baskerville; E. A. Beck; F. Becket, Niagara Falls, N. Y.; E. H. Bedell, Newark, N. J.; J. W. Belkman, Niagara Falls, Ont.; P. McN. Bennie, Niagara Falls, N. Y.; S. D. Benoliel, Philadelphia, Pa.; D. D. Berolzheimer; Mr. and Mrs. E. R. Berry, Walden, Mass.; Mr. and Mrs. W. F. Bleecker, Denver, Colo.; P. A. Boeck, Worcester, Mass.; G. Boericke, Primos, Pa.; M. T. Bogart; W. Bowman; C. S. Bradley; A. A. Breman; Miss C. P. Briggs, Brooklyn, N. Y.; C. P. Brown, Smithsburg, Md.; J. W. Brown, Cleveland, Ohio; C. A. Campbell, Boston, Mass.; C. R. Carey, Philadelphia, Pa.; C. F. Carrier, Jr., Sewaren, N. J.; H. R. Carveth, Niagara Falls, N. Y.; H. Casselberry, Altoona, Pa.; J. Casselberry, Altoona, Pa.; C. C. Cito, Irvington, N. J.; G. Clamer, Philadelphia, Pa.; J. C. Clancy; Mr. and Mrs. F. O. Clements, Dayton, Ohio; Mr. and Mrs. W. R. Clymer, Cleveland, Ohio; Mr. and Mrs. J. Coates, Coatesville, Pa.; F. P. Coffin, Schenectady, N. Y.; B. Cohen, C. C., N. Y.; S. Cohn; H. B. Cohn; E. A. Colby, Newark, N. J.; A. M. Comey, Chester, Pa.; E. A. Congdon; E. B. Conklin, Ossining, N. Y.; H. R. Connell, Pittsburg, Pa.; J. C. Cothran, Harrison, N. J.; Mr. and Mrs. W. A. Cowan, Brooklyn, N. Y.; Mr. and Mrs. A. H. Cowles, Sewaren, N. J.; F. Crabtree, Pittsburg, Pa.; F. N. Crawford, Middletown, Conn.; B. O. Crites, Cleveland, Ohio; M. A. Daniels; P. Darkicos, E. Orange, N. J.; T. Darlington; Mr. and Mrs. E. A. Deeds, Dayton, Ohio; P. K. Devers, Jr.; W. Dickinson; C. Dill, Perth Amboy, N. J.; C. A. C. Dreyfus; W. Dreyfus; G. Drobegg; E. Durant; H. W. Durham; A. S. Dwight; C. H. Eldridge, Newark, N. J.; J. W. Elms, Haskell, N. J.; C. E. Evans, Philadelphia, Pa.; J. W. Evans; W. E. Fairhurst, S. Bethlehem, Pa.; R. W. Faust, Bethlehem, Pa.; K. Fenning; B. G. Fernberg; H. M. Fernberger, Newark, N. J.; A. Finck, Cleveland, Ohio; Dr. and Mrs. C. Fink, Harrison, N. J.; C. Fitz Gerald; F. A. J. Fitz Gerald, Niagara Falls, N. Y.; K. G. Frank; Milton W. Franklin, Schenectady, N. Y.; T. Fukin; Richard H. Gaines; F. W. Garen; Miss Gibbs, Wyandotte, Mich.; H. E. Gibbs, Wyandotte, Mich.; J. Goldbaum, Philadelphia, Pa.; F. X. Govers; S. H. Graham, Newark, N. J.; A. E. Greene, Chicago, Ill.; J. Griffin, Bethlehem, Pa.; H. B. Haigh; W. A. Hamor; J. H. Hall, High Bridge, N. J.; A. M. Hamblet, Rumford, Me.; C. Hambuechen, Madison, Wis.; W. J. Hammer; N. V. Hansell; C. A. Hansen, Schenectady, N. Y.; L. O. Hart, Hoboken, N. J.; D. S. Hartshorn; W. H. Hendricks, Palmerton, Pa.; W. A. Hendryx; C. L. B. Hennig, Haskell, N. J.; C. Hering, Philadelphia, Pa.; P. L. T. Heroult; H. Hess, Philadelphia, Pa.; B. C. Hesse; R. Hewitt, Ardsley-on-Hudson; H. D. Hibbard, Plainfield, N. J.; E. F. Hicks, Philadelphia, Pa.; A. Higgins, Worcester, Mass.; A. E. Hikelman, Tremont, Pa.; Mr. and Mrs. H. H. Hitchcock, Tarentum, Pa.; G. B. Hogaboona, Newark, N. J.; J. F. D. Hoge; W. E. Holland, E. Orange, N. J.; F. Holmgren, Sweden; A. H. Hooker, Niagara Falls, N. Y.; H. Howard,

Boston, Mass.; Mr. and Mrs. H. A. Horner, Philadelphia, Pa.; G. A. Hulett, Princeton, N. J.; M. A. Hunter, Troy, N. Y.; W. R. Ingalls; Mr. and Mrs. A. von Isakovics, Monticello, N. Y.; E. B. Jewett, Clarksburg, W. Va.; Mrs. R. A. A. Johnston, Ottawa, Ont.; W. M. Johnson, Hartford, Conn.; Mrs. A. F. Joyce, Boston, Mass.; Mr. and Mrs. C. M. Joyce, Arlington, N. J.; R. Kann; C. P. Karr, Plainfield, N. J.; N. S. Keith, Philadelphia, Pa.; J. T. Kelly, Philadelphia, Pa.; P. Kemery, Pittsburg, Pa.; E. F. Kern; F. L. Koethen, Bloomfield, N. J.; M. M. Kohn; L. Kristeller; L. J. Krom; Miss N. Laib, Philadelphia, Pa.; Dr. and Mrs. G. F. Kunz; I. Langmuir, Schenectady, N. Y.; J. Langton; E. Leavitt, Brooklyn, N. Y.; C. A. Le Boutillier, High Bridge, N. J.; W. S. Lenk, Toledo, Ohio; F. A. Lidbury, Niagara Falls, N. Y.; C. M. Liddell, Elizabeth, N. J.; C. Lindsay, Schenectady, N. Y.; M. Loeb; M. L. Loeb; E. G. Love; A. F. Lucas, Washington, D. C.; H. J. Lucke; P. Luhr, Newark, N. J.; T. L. B. Lyster, Niagara Falls, N. Y.; Wm. Main; C. W. Marsh; G. H. Marshall; E. J. K. Mason; J. H. Mathews, Madison, Wis.; Irving C. Matthews, Harrison, N. J.; P. H. Mayer, Mt. Vernon, N. Y.; G. W. Maynard; F. J. Maywald; A. S. McAllister; J. H. McCreary; C. F. McKenna; C. Mendeleff, Mauver, N. J.; F. C. Meucke, Philadelphia, Pa.; L. B. Miller, Newark, N. J.; C. P. Monte, Cleveland, Ohio; H. R. Moody; S. Morey; E. P. Morgan, Mt. Vernon, N. Y.; J. F. Morgan, Bronxville, N. Y.; Mr. and Mrs. F. D. Morgans, Mt. Vernon, N. Y.; G. D. Morgans, Bronxville, N. Y.; C. H. Moritz, Niagara Falls, N. Y.; C. W. Moulton, Poughkeepsie, N. Y.; J. M. Muir; A. Neisch; H. B. North, New Brunswick, N. J.; C. H. Ohlwiler, Altoona, Pa.; R. S. Orr, Pittsburgh, Pa.; C. G. Osborne, Chicago, Ill.; J. A. Oswald, Dayton, Ohio; G. S. Page, Pittsburg, Pa.; J. H. Parker, Reading, Pa.; L. A. Parsons, Gettysburg, Pa.; H. W. Pattcn, Washington, D. C.; F. A. Patterson; T. R. Patton, Philadelphia, Pa.; A. M. Peck, Port Chester, N. Y.; J. C. Pennie; G. A. Perley; P. A. Pfeiffer, Hoboken, N. J.; W. J. Plews, Montreal, Can.; H. A. Plusch, Perth Amboy, N. J.; B. Popper; W. S. Potter; W. B. Pritz, Cleveland, Ohio; J. Probst, Paterson, N. J.; A. L. Queneau, Philadelphia, Pa.; T. S. Quinn, Brooklyn, N. Y.; A. W. Ramage, Buffalo, N. Y.; R. W. Rehenklau; Dr. and Mrs. J. W. Richards, South Bethlehem, Pa.; H. W. Richardson, Bethlehem, Pa.; H. S. Richardson, State College, Pa.; H. S. Riederer; G. Rigg, Palmerton, Pa.; M. M. Riglander; C. D. Robb; A. G. Rodgers, Niagara Falls, N. Y.; E. F. Roeber; A. Rogers, Brooklyn, N. Y.; F. W. Roller; E. Romanelli, State College, Pa.; L. G. Rowland, Summit, N. J.; D. B. Rushmore, Schenectady, N. Y.; Wm. T. Rynard; A. V. Salmon, C. C., N. Y.; P. G. Salom, Philadelphia, Pa.; G. W. Sargent, Pittsburg, Pa.; L. E. Saunders, Niagara Falls, N. Y.; M. Schaak, Brooklyn, N. Y.; L. B. Schleicher; L. Schlichting; C. A. Schluederberg, Pittsburg, Pa.; W. S. Schneider; F. F. Schuetz; R. C. Schwarz, Rochester, N. Y.; C. F. Scott, Pittsburg, Pa.; W. K. Scott, State College, Pa.; Mr. and Mrs. J. A. Seede, Schenectady, N. Y.; Clayton H. Sharp; H. F. Sill, Pittsburg, Pa.; S. Skowronski, Perth Amboy, N. J.; W. M. Slater, Washington, D. C.; R. Sliter, Jersey City, N. J.; A. Smith, Niagara Falls, N. Y.; A. S. Smith, E. Orange, N. J.; W. Smith, Cleveland, Ohio; W. Smith, Montclair, N. J.; E. S. Smith, Niagara Falls, N. Y.; H. H. Smith, Beville, N. J.; J. G. Smull, Palmerton, Pa.; R. C. Snowdon, Ossining, N. Y.; J. L. K. Snyder, Cleveland, Ohio; W. L. Spalding, Buffalo, N. Y.; Mr. and Mrs. E. A. Sperry, Brooklyn, N. Y.; E. G. Spillsbury; N. Statham, Yonkers, N. Y.; S. R. Steinhardt; Mr. and Mrs. K. R. Sternberg, Springfield, Mass.; F. E. Stoller, Cleveland, Ohio; G. C. Stone; J. S. Stone, Boston, Mass.; J. A. Stremel, Brooklyn, N. Y.; A. H. Strong; F. L. Summers, Altoona, Pa.; J. Sunlver, Newark, N. J.; M. A. Swenson; Y. Taiima, Hoboken, N. J.; J. Takamine; A. O. Tate, Toronto, Can.; Miss C. Taylor, Penn Yan, N. Y.; E. C. Taylor; Mr. and Mrs. E. R. Taylor, Penn Yan, N. Y.; F. D. Taylor, State College, Pa.; J. E. Teeple; C. J. Thatcher; V. J. Thrane, Niagara Falls, N. Y.; H. Thurston, Arlington, N. J.; C. B. Thwing, Philadelphia, Pa.; Mr. and Mrs. M. Toch; S. A. Tucker; R. H. Vail; W. Valentine, Waterbury, Conn.; Mrs. E. W. Vivian; C. H. Vom Baur; L. A. Voorhees, New Brunswick, N. J.; J. D. Vorce, Detroit, Mich.; M. Waddell; Leonard Waldo; D. K. Wright, Newark, N. J.; M. A. Yunck, S. Orange, N. J.; Fr. Zimmermann, Newark, N. J.

SYNOPSIS.

Gold and Silver.

Amalgamation Following Tube Milling.—It has been the generally accepted practice in South Africa that amalgamating plates in a tube mill plant should shake, as it was considered that the pulp was too concentrated to allow the gold particles to come in contact with the amalgamated surfaces of the plates. In the *Journal* of the Chemical, Metallurgical and Mining Society of South Africa, for January, 1911, Mr. W. R. DOWLING gives some results obtained by stationary plates, showing that they are quite as efficient as the shaking plates.

The experimental work was carried on at the joint plant of the Knights Deep and Simmer East mines. Fifteen plates, each 10 ft. 5 in. long by 4 ft. 7 in. wide, of a three-tube mill plant were stopped shaking, and the fall of the plates increased from 10 to 18 per cent. Under these conditions amalgamation was most satisfactory. For the two months preceding the use of stationary plates the extraction by tube mills was 21.81 per cent of the screen assay value and 34.04 per cent of the tailing value. For the two months succeeding the extraction was 22.94 per cent of the screen assay and 34.02 per cent of the tailing assay.

With a view to reducing the number of plates if possible, it was decided to try a concentrating scheme. Two small cones and two stationary tables were erected. It was found that the gold released from the sand by tube milling could be concentrated in the underflow of the cones, and that the underflow with some diluting overflow could be handled by two plates. To test this scheme five and two plates took the pulp from one tube mill on alternate days. The result was a difference of 1.2 oz. of amalgam per day in favor of the two plates. After these results the entire plant was put on a basis of two plates per tube mill, and has been operating thus for some time with good results.

In adopting two instead of five plates it is important that the water ratio should be kept low, and it has been found that a moisture of 55 to 57 per cent, or a ratio of solid to liquid of about 1 to 1.27, is the irreducible minimum. Reduced to its simplest form amalgamation simply means bringing the amalgamable gold into contact with the amalgamated surface, and it is apparently accomplished by this arrangement.

Fine Grinding.—The all-sliming or one-product method of cyaniding gold and silver ores has been the subject of more or less varied criticism for some time. The latest and one of the most ardent champions of the system is Mr. H. S. DENNY, who contributes his views to the *Mining Magazine* (London) for March, 1911. Mr. Denny's wide experience in South Africa and Mexico will earn for him a respectful hearing.

Granting that in certain unusual cases the additional outlay for grinding machinery and operating cost will offset any advantages gained by fine grinding, Mr. Denny states most emphatically that as a generalization this belief is wrong. The most pertinent questions on the fine grinding process are its limiting factors and its efficiency. The former are involved in a comparison between the added cost of each extra unit of fine grinding and the added extraction, and are not easily determined. The author finds hope, however, in the recent work by Stadler on the efficiency of grinding machines, and believes that we may yet find a method of computing with some degree of accuracy the all-important figure of cost. As regards the added extraction, there is no doubt that if care is taken this point can be determined.

Discussing the economic considerations, the author assumes the treatment of a silicious ore containing 5 per cent pyritic material. Assume that this be ground to 60-mesh, and that 90 per cent of the pyritic content can be concentrated, carrying 70 per cent of the gold. If the ore assays 0.4 oz. per ton we have after concentration: 4.5 tons of concentrate carrying 28 oz., or 6.22 oz. per ton; 95.5 tons of pulp carrying 12 oz., or 0.125 oz. per ton.

If the pulp were then classified we probably would have, say, 65.5 tons of sand, assaying 0.15 oz. per ton, and 30 tons of slime, carrying 0.0725 oz. per ton.

Now, assuming that an extraction of 80 per cent could be made on each of the above products (concentrate and pulp) by agitating and pressing and without further grinding, the question naturally would arise as to application of finer grinding. It is quite clear that the concentrate is the portion demanding attention. An 80 per cent extraction of the 6.22 oz. would leave a residue of 1.24 oz., whereas an 80 per cent extraction on the original ore (80 per cent of 0.4 oz.) would leave a residue of only 0.08 oz., and it would at once seem possible to materially reduce that by a proper handling of the concentrated portion.

Fine grinding would be suggested as the remedy and might be accomplished as follows:

Regrinding all the pulp from the crushing plant that is coarser than 200 mesh.

Regrinding a hydraulically classified portion of the pulp.

Regrinding a carefully concentrated portion of the pulp.

Assuming that one of these methods was followed and that the extraction would be increased from 80 to 94 per cent, we would find that our concentrate is the portion yielding the greatest increase extraction, and is, therefore, the part to be

considered most carefully. The question would arise whether it might not be best to get a very clean concentrate and regrind that portion only. The main points in connection with this are:

What is the cost per ton of regrinding from 60 to 200 mesh?

If the sand is not reground would it be necessary to have a separate treatment plant for it, or could it be mixed with the concentrate?

If 50 per cent of the valuable content of the ore could be extracted by amalgamation, would that step be advisable, even though the ultimate recovery would not be increased?

As the concentrate contains the bulk of the value, would it not be well to make some provision for regrinding this product so as to insure reduction to absolute impalpability. Could this be done, and how?

Answering these questions seriatim, the author believes that:

The cost of regrinding from 60 to 200 mesh will not be far from the average figure of 75 cents per ton, although varying conditions may change this.

It would not be wise to try to treat the unground sand and the reground concentrates together. In filter pressing the mixture of coarse and fine would cause trouble. While it would be necessary to have separate treatment plants for sand and concentrate, the sand need not be ground so fine as the concentrate. A plant designed to handle and treat one product only would be much more simple in design, less costly in capital outlay, and cheaper to work than a plant consisting of three departments for as many products.

There is nothing to be gained by a recovery of 50 per cent by amalgamation, because the operation costs something and involves some loss, and because it is too easy to steal gold in amalgam form. The extra consumption of cyanide and zinc would be almost inappreciable if all the gold were recovered in the cyanide solution.

If a clean concentrate can be made there is little doubt that it would be advisable to recover it and grind it to impalpable pulp, which is possible in several ways.

The author quotes from a paper of A. Grothe presented before the Mexican Institute of Mining and Metallurgy, in which Mr. Grothe shows a flow sheet embodying the idea of regrinding concentrate and coarse sand in tube mills, keeping the concentrate in the circuit of tables and tube mill until it is ground fine enough to pass off from the tables with the slime.

The author concludes as follows: The day of the percolation method of treatment for sand and the decantation system for the treatment of slime has gone, except in special cases, and it is only due to the persistence of some of the metallurgical fraternity who have been wedded to these ideas that they are to-day included so freely in the treatment of simple gold ores. There can be no logical argument opposed to the principles of the one-product treatment now becoming almost universal, and the only points remaining for settlement are the methods and devices to be adopted in fine grinding, and the degree to which this principle is to be pushed in order to gain the best results.

Modern practice has so emphatically taken up the circulation of cyanide solution throughout the entire plant, followed by fine grinding and filter pressing, and the results have so abundantly demonstrated the value of this practice that there can no longer be any discussion as to its merits. The author predicts that the tendency in the future will be to crush in stages, classify, grind in stages, agitate and filter press on the one-product principle. He believes that the upshot will be cheaper plants, lower working costs and improved extractions.

Combination Treatment of Silver Ores.—As an example of the combination concentration-cyanide treatment of a Mexican silver ore, V. B. Sherrod contributes an experience to the *Mexican Mining Journal* for April, 1911. Before the present system of treatment was adopted a great deal of unsuccessful experimental work was done in an effort to dispense entirely with concentration. Extraction was fair but consumption was prohibitive. A thorough concentration was possible and successful metallurgically, but the grade of the concentrate was so low that marketing costs would have been excessive.

Finally a series of experiments conducted for another purpose showed how a combination could be effected, whereby the grade of the concentrate would be high, and the consumption of chemicals in cyanidation would not be much higher than when concentration was thorough. The following screen analysis of the lip sample from a battery crushing through 4-mesh screen illustrates the possibilities of concentrating a selected portion of the pulp, selection being made by screening, followed by hydraulic classification or not.

Screen Size	Per cent of Total Pulp	Kilos Silver	Per cent of Total Value
— 20	43.9	0.647	35.3
— 40	13.6	0.733	12.4
— 80	11.2	0.876	12.4
—150	6.6	0.850	7.0
—200	2.7	0.968	3.2
—200	22.0	1.086	29.1

A trial of the concentrates from the above samples showed that every particle of mineral liberated would easily pass a 60-mesh screen, but on account of the trouble and expense connected with fine wet screening it would be ore practicable to use coarser screens and handle a somewhat larger tonnage for the concentration. A Callow screen of 30-mesh slot-woven wire cloth, traveling at 60 feet per minute, gave the following results:

	Screen Size	Per cent of Total Pulp	Kilos Silver	Per cent of Total Value
Oversize.....	30	49.5	0.653	40.5
Undersize.....	30	50.5	0.945	59.5

The undersize from the above operation was classified into two grades of sand and one of slime, the latter being sent directly to the slime plant. By this scheme a very high grade of concentrate was obtained, and six tables were sufficient for a 300-ton plant. Conditions in the cyanide treatment were more normal, and the problem was solved successfully.

ASSAYING.

Notes on the Matte Assay.—Most assayers will be surprised to hear a method of fire assaying recommended which intentionally gives rise to the formation of a matte. Such, however, is the recommendation of Mr. L. J. Wilmoth, who communicates his method in the December *Journal* of the Chem. Met. and Mining Soc. of South Africa. He suggests its use for the assay of battery chips of iron and steel, and for copper drillings. He has even extended it to the assay of high-grade copper-bearing material with good results. The principal advantage derived from the formation of a matte is that it will take care of a large amount of base material.

The following charge was used with metallic copper: Drillings, 10 g; litharge, 1 AT.; sodium carbonate, 1.5 AT.; borax, 1 AT.; silica, 0.5 AT.; charcoal, 1.25 g; sulphur, 8 g; and a heavy iron nail. The average result from three assays was 16.417 per cent gold. The same sample assayed by the scorification method gave 10.31 per cent fine gold, and by the combination method, 10.49 per cent.

For copper matte carrying 25 per cent to 30 per cent copper and some lead the charge consisted of: Matte, 0.5 AT.; litharge, 1 AT.; sodium carbonate, 1 AT.; borax, 1 AT.; silica, 1 AT.; sulphur, 0.1 AT.; charcoal, 1.5 g; and a heavy iron nail. On pouring the charge the matte was easily detached from the lead button. On reassaying the matte it was found to contain no gold and only a little silver.

The author concludes that the application of the matte assay for copper-gold-bearing ores and products presents no difficulty and is of great advantage when the percentage of copper is high and the gold value low. The advantage is illustrated by the assay of 20 per cent copper matte. In this case a scorification gave 48.5 dwt. and a matte assay 53 dwt. per ton.

Recent Metallurgical Patents.

Iron and Steel.

Improved Construction.—With the object of providing a cheap, simple and efficient form of bosh, in which burning out is reduced to a minimum, Mr. FRANCIS J. ZIPPLER, of Avalon, Pa., has patented a form of construction shown in the accompanying cuts, Figs. 1 and 2. Fig. 1 shows the lower portion of a furnace embodying his form of invention, in which 1 represents

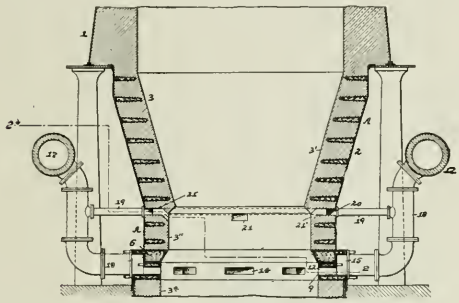


FIG. 1.—BOSH CONSTRUCTION.

the upper portion and 2 the bosh of a furnace. Fig. 2 represents a section along the line 2-2 of Fig. 1. The usual bustle pipe and tuyere supply pipes are designated by 17 and 18 respectively. Water jackets 6 and 9 form an annular air chamber 15, into which air from the pipes enters, being deflected into pockets 13 formed by the walls 12, so that it will pass around the same and into the tuyeres 14 of the furnace. At the same time a portion of the air blast will pass through the pipes 19, and thence into an annular space 20 in the wall portion 3", and enter the tuyeres 21. Owing to the tapering construction near the tuyeres, the lower or main blast is directed upward, and that from the auxiliary blast downward, through the charge. By the use of the annular air chambers, a large volume of air

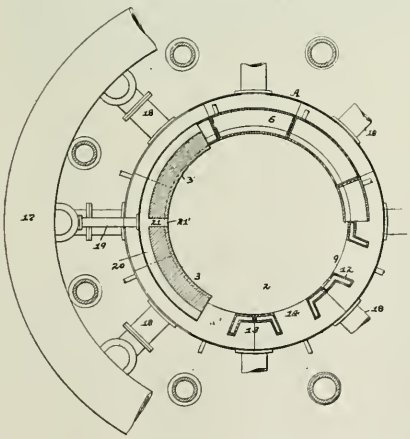


FIG. 2.—BOSH CONSTRUCTION.

is admitted and equalized before it passes through the tuyeres. By the use of the upper tuyeres in connection with the lower, the furnace is prevented from scaffolding, and will economize in fuel. The construction is further claimed to prevent the burning out of the bosh walls and the contained tuyeres. (986,792, Mar. 14, 1911.)

Conversion of Iron to Steel.—The type of apparatus in which iron is converted into steel by means of a surface blow is the subject of a patent granted to Mr. SELDEN S. DEEMER, of New Castle, Del. In this process as heretofore practised there has been inefficiency by reason of the sluggish movement of the molten bath, and the difficulty of introducing means of desulphurizing, dephosphorizing, decarbureting or recarbureting the metal. The inventor's proposed converter and process are designed to overcome these objections.

Fig. 3 is a side elevation of the converter, and Fig. 4 a section on line 3-3 of Fig. 3. The converter has the usual outer shell and lining, with the bottom of the hearth rounded to present as little resistance as possible to the circulation of the metal during the blow. Air for the surface blow is provided by a suitable engine, and conducted to the tuyere *B* by way of the connection d^1, c and wind box *C*. The trunnion *D* is hollow and provided with a slotted opening *d*, through which the air may flow when the converter is in proper operating position. The tuyere may be a series of small openings or a single long one in the form of a narrow slot.

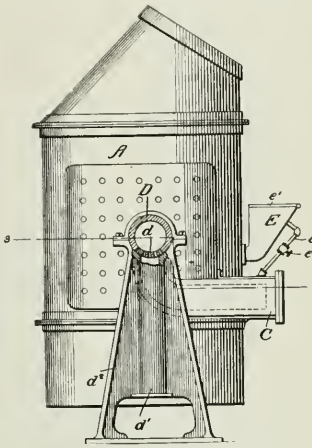


FIG. 3.—CONVERTER.

In order to provide for the suitable introduction of reagents at the proper time, a charging box *E*, divided by partitions and served by air from the main air chamber through pipes e^2, controlled by valves e^3, is placed just above the wind box as shown in the cut. The opening from the charging box into the converter is arranged to deliver the reagents onto the molten bath at a greater oblique angle than that of the surface blow.

In operation the converter is charged with molten low-grade metal, such as pig iron, to the proper depth; the charging box is filled with the proper reagents; the air blast is turned on and the surface blow continues, with rapid circulation of the metal, until the time arrives for the charging of the reagents. This is accomplished by opening the proper valve e^3. When the blow is completed the converter is tipped and discharged in the usual manner. (987,704, March 28, 1911.)

Agglomerating Fine Ore.—The preparation of fine and friable iron ores for smelting is the subject of an invention of Mr. HUGO DICKE, of Frankfort-on-the-Main, Germany, which he has assigned to Jacob E. Goldschmid, of the same place. The inventor claims that in previous devices for agglomerating fine ore the principal fault has been the method of firing. He states that by firing a revolving furnace with water gas entirely new results are obtained in the treatment of iron ore. Not only can all the good results heretofore accomplished be obtained in the water-gas flame, but, owing to its peculiar formation, oxidation or reduction and fusion can take place at will. The essential advantage of the water-gas flame is the possibility of controlling it and adapting it to the needs of the material under treatment.

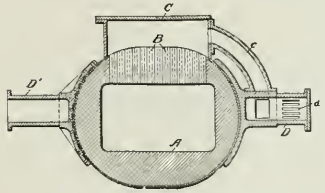


FIG. 4.—SECTION OF CONVERTER.

In the practice of the invention it has been ascertained that what is called a neutral water gas flame, viz., one in which the proportions of gas and air are in chemical equivalents, iron ores of the formula Fe_2O_3 can be reduced to Fe_3O_4, and that this will continue, even though the flame be slightly oxidizing.

The working of the furnace can be so regulated as to produce three zones of action; one in which drying and preliminary heating occur; another where oxidation and fusion are in progress, and a third where volatilization of noxious elements such as sulphur and arsenic may be removed. The inventor states that the furnace and water-gas flame may be used with equal advantage on ores of iron, copper, etc., the removal of small quantities of zinc, sulphur, arsenic, etc., being greatly facilitated in case of the latter. (986,271, March 7, 1911.)

Still another method of preparing fine materials for furnacing is that recently patented by Mr. George W. Coggeshall, of Dedham, Mass. The process does not relate especially to iron ores, but is applicable to the preparation of mixtures of fine material preliminary to some subsequent operation in a furnace. The principal idea involved is to add a suitable binding agent, such as water or a solution of sugar, glue or the like, to an excess of the dry mixture, thus getting the full benefit of the binding agent without using an excess of it: then screening the unbound material from the nodules formed by the binder, and treating the nodules in any way desired. The operation can be carried on in an ordinary rotary drier, the upper part of which is screened to permit the discharge of the fine unbound material. The nodules pass over the screen to the lower part of the tube, which may be heated to bring about the desired reaction between the constituents of the nodules.

Specifically, the inventor applies his operation to the formation of potassium titanate, which is formed from dry, finely ground titanium oxide and anhydrous potassium carbonate. The mixture is subjected to the action of drops of water which agglomerate it into nodules, which are later heated to perfect the reaction. This takes place at a dull red heat and is almost quantitative in as short a time as 17 minutes. (987,554, March 21, 1911.)

A Blazing Tuyere has been devised by Mr. Charles A. Case, of New York, for the purpose of firing a furnace with liquid or gaseous fuel admitted through the tuyere with an air blast, and igniting the fuel electrically as it passes through the tuyere. The air jet passing through the tuyere has a tubular core of low-pressure air, surrounded by a zone of high-pressure air. The oil or gas used as fuel enters with the low-pressure air and passes an arc which ignites it. Combustion is supported by the low-pressure air core, while the high-pressure zone protects the shell of the tuyere from burning and supplies oxygen for the reduction of the ores in the furnace. (985,394, Feb. 28, 1911.)

Gold and Silver.

Treatment of Telluride Ores.—In view of the speculation which has been rife regarding the process used by the Portland Gold Mining Company at its mill at Victor, Col., the following patent specifications will be interesting. They form the basis of a patent granted to Mr. Thomas B. Crowe, superintendent of the Victor mill and one of the staff engaged in experimental work prior to the adoption of the process now used. It is not to be inferred that the following is a statement of the process now being used at Victor, although it may contain some of the essential details.

The two main points in the patent are that an alkaline hypochlorite is a solvent for tellurium, and that, in order to apply this reaction profitably to telluride ores, it is necessary to remove from the ore all possible reducing agents, such as sulphides of the base metals. Another essential is that the natural acidity of the ore be *just* neutralized, instead of having an excess of lime, for example, as is customary in ordinary cyanidation.

The inventor mentions the use of an alkaline persulphate with potassium cyanide for the similar purpose of dissolving tellurium and gold in telluride ores, but states that he has found the required quantities so large as to be economically prohibitive. He further states, however, that he has discovered that cyanogen iodide will serve as a suitable and efficient solvent if the natural acidity of the ores is just neutralized as stated above. The advantage in the use of cyanogen iodide is that it is at once a solvent for both gold and tellurium thus being a simpler reagent than a combination of, say, potassium cyanide and an alkaline persulphate. The claims granted in the patent, however, refer only to the use of an alkaline hypochlorite and alkaline cyanide, and not to the use of cyanogen iodide. The removal of sulphides by concentration and the neutral state of the ore are referred to as necessary conditions. (987,964, March 28, 1911.)

Vacuum Filters.—A continuous vacuum filter has been devised by Mr. George Johnston, of Chihuahua, Mexico. As to its type, it consists of a number of filtering frames of

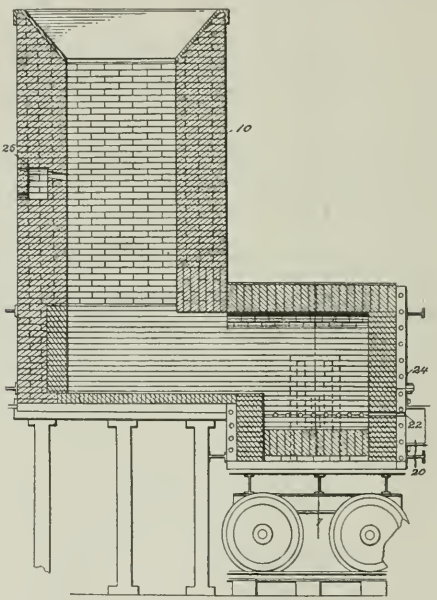


FIG. 5—VACUUM FILTER.

the usual construction arranged singly or in groups around an axis. The axis and frame are made to revolve so that the filter frames are successively immersed in the pulp to be filtered and raised out of the pulp for the purpose of washing and discharging the cake. The filter is in reality a modification of the drum filter idea, in which the filter surface is made of a number of frames, instead of the periphery of a drum. Or, in other words, it is a modification of the ordinary leaf filter idea, in which the leaves are subjected to continuous instead of intermittent operation.

The apparatus is arranged in such a manner that when the filter leaves are immersed in solution they are also in connection, through suitable valves, with a vacuum pump, and that as they revolve out of the solution they are brought to a position convenient for washing and dumping the cake. If used for the filtration of gold cyanide solutions it would appear that the filter is not complete in itself for there is no provision for displacing the solution held in the pulp, and this operation would have to be performed in a second filter treating the discharged cake from the first. (988,107, March 28.)

Copper.

Continuous Matte Smelter.—A novel idea in copper smelting furnaces has been patented by Messrs. REDICK R. MOORE, of Garfield, and JOHN H. KEES, of Salt Lake City, Utah. The furnace has for its object the continuous production of a matte of lead, copper, nickel, cobalt and the precious metals, without the need of previous roasting and without the addition of carbonaceous fuel.

As shown in Fig. 5, the furnace consists of a stack 10 and a converting chamber 14 connected by a passage 13. The converting chamber is suitably mounted on wheels to facilitate its removal for repairs, etc.

The underlying object is to utilize the heat escaping from the converting chamber to smelt the ores in the stack, causing them to melt and run into the converting chamber, where they are converted into a matte by a suitable blast applied through tuyeres 22. The air blast is supplied through a flue 20 surrounding the converter. The latter is water jacketed in the usual manner, shown conventionally at 24.

In operation the stack 10 is charged with the ores to be reduced, and a quantity of molten matte from any source is placed in the converter. Vapor of S and SO_2 are both present with nitrogen in the upper part of the furnaces treating pyritic ores and the sulphur will condense on the cool charge if it be high enough above the source of heat. As the charge continually travels downward the condensed sulphur will be again vaporized, resulting in a zone where the charge is permeated with sulphur vapor. This may be removed by a condensing arrangement, 25, which is a pocket opening in the stack wall.

When the air blast is turned on heated air will be forced through the matte, oxidizing iron, sulphur, etc., forming ferrous sulphide which will unite with the silica and other constituents of the charge in the stack to form a molten slag and produce sulphurous gas which with the nitrogen of the air will give up its heat to the cooler charge in the stack (thus melting down more sulphide to be converted into matte in the lower chamber, and supply fuel for further smelting. (985,105, Feb. 28.)

Treatment of Low-Grade Ores.—The recovery of copper from the so-called porphyry copper ores of the West is usually accomplished by concentration and smelting. The ores contain an average of 2 per cent copper and they are treated on a huge scale. Mr. GEORG SCHNEIDER, of Lauenstein, Germany, proposes a chemical method of treatment which he believes would result in greater percentage recovery of copper.

His experiments have shown him that such ores are amenable to treatment consisting of a roast with 10 per cent salt at a low red heat, and subsequent lixiviation with 5 per cent HCl solution for eight hours. His tests showed that ore containing 2.34 per cent copper would yield 2.22 per cent metal. He finds that these results are due to the presence of silica in the ore, which acts in a way to decompose the alkaline chloride used, liberating chlorine for the chlorination of the copper sulphides. It is claimed that this process would be cheaper and more efficient than the present method of concentrating and smelting, and that ores of as low a grade as 1 per cent copper could be treated at a profit. Any silver in the ore would be extracted as chloride in the presence of iron and alkali chlorides and gold would have to be extracted by the ordinary chlorination process. (986,508, March 14, 1911.)

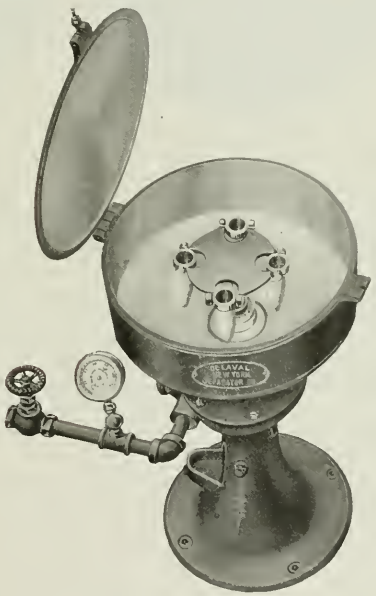
Nickel.

Sulphide ores of nickel, cobalt, iron, copper, arsenic, zinc and molybdenum, but particularly nickel, may be treated according to a process recently patented by Mr. JAMES A. MCLARTY, of Toronto, Canada. His object is to oxidize the sulphide ore without resorting to the usual roasting process. Preferably he first concentrates the ore in the usual manner, after which it is treated in a magnetic concentrator to remove iron and nickel sulphides. These are then oxidized by heating in a closed vessel with steam at 125 lb. pressure. The reaction which follows forms oxides of the metals and H_2S gas. The nickel oxide thus formed is not magnetic and may be removed from the iron oxide which is magnetic by again passing the pulp through a magnetic separator. (987,156, March 21, 1911.)

Centrifuge Tester.

In our March issue, page 147, a machine for the separation of liquids and solids by centrifugal force was described, which has recently been developed by the De Laval Separator Company, 165 Broadway, New York City. In this development work, which extended over ten years, the engineers of this company had to carry out a large number of tests with mixtures of materials of all kinds to see what kind of a separation could be made with them by centrifugal force.

For these tests they developed a small centrifugal tester, which has proven exceedingly convenient and handy for such work. It is operated by direct steam, the steam acting on a turbine wheel mounted on a flexible shaft, and the driving mechanism being very similar to that of the well-known De Laval cream separator. On the basis of many years' practical experience with the latter it has been possible to give to the



CENTRIFUGE TESTER.

small centrifuge tester a very rigid and compact construction.

When the tester revolves the test tube pockets take a horizontal position, as is indicated by a dotted line for the right-hand pocket in the adjoining illustration. There is, of course, nothing new about this. But the feature of the De Laval tester is the exceedingly high speed which can be obtained without difficulty and the wide speed limits within which it can be operated. This tester can be driven up to a speed of 4000 revolutions per minute, which is believed to be much higher than the maximum speed obtained in former machines of this kind. In this way a pressure of 168 pounds per square inch can be obtained for water, the corresponding pressures for other materials being easily determinable from this figure with the aid of their specific gravities.

As the speed can be lowered down to about 100 revolutions per minute, it is evident that any suitable pressure within wide limits can be obtained and that with this tester it is easy to determine quickly the possibilities of separation by centrifugal force in any given case.

A New Type of Synchroscope.

In the alternating-current power plants, which are now the rule rather than the exception in our large metallurgical and chemical works, the parallel connection of alternators is still at times a troublesome problem. At present there are only two types of synchronizing device in general use, namely, the synchronizing lamp or voltmeter and the rotating synchroscope. The first method is crude and consumes much time. It is entirely unsuited to use on large systems. The second device is unreliable and is generally used with synchronizing lamps to guard against a fatal error. At best the rotating synchroscope, even when combined with a lamp, is an extremely unsatisfactory device, since it requires the operator to divide his attention between the lamps and the instrument.

FIG. 1.—FRONT VIEW OF SYNCHROSCOPE.

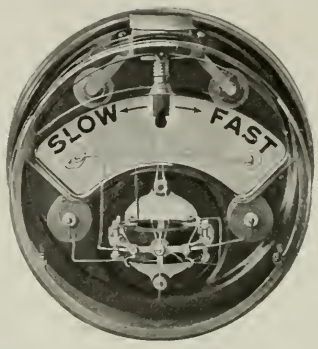


FIG. 2.—PHANTOM VIEW SHOWING LAMP AND POINTER.

There has recently been placed on the market an entirely new type of synchroscope, which has all the characteristics of an ideal rotating synchroscope and at the same time its indications are infallible and accurate.

In construction the instrument is quite similar to the new Weston type switchboard single-phase wattmeter; that is, it is a Weston electrodynamometer instrument. The pointer is mounted behind a translucent glass and stands normally in the middle of the scale. Behind the pointer is installed a small synchronizing lamp, which casts the shadow of the pointer on the scale. The pointer is only visible by the light from this lamp.

The connections of the instrument are shown in Fig. 3. The movable coil is connected in series with a condenser across the incoming machine, while the fixed coil is connected in series with a slightly inductive resistor across the line. The lamp is connected in the secondary circuit of a special three-legged transformer with two primaries, one across each source. The lamp transformer, condenser and resistor are mounted on one base and installed in an auxiliary box.

The mode of operation may be easily comprehended by thinking of the instrument first as a correctly designed wattmeter. It is well known that a good wattmeter will show no deflection when the current and e.m.f. are in exact time quadrature; therefore, in the present instance, if the constants of the circuits are properly chosen the currents in the coils will be in quadrature when the e.m.fs. of the two sources are in phase coincidence or in phase opposition. That is, the pointer stands at rest in the middle of the scale under these conditions, and since the lamp is dark when the e.m.fs. are in phase opposition, it will be seen in this position only when the machines to be synchronized are in exact phase coincidence with each other.

For every value of phase displacement between the e.m.fs. a corresponding torque will be developed and the pointer will be deflected to one side or the other, depending upon whether one of the waves is lagging or leading with respect to the other. That is, a good wattmeter connected in this way will indicate the phase displacement between two e.m.fs.

When the frequencies of the two e.m.fs. differ, the phase displacement between the two will vary continuously through complete cycles of 360 time degrees and the currents in the coils of the instrument will alternately lag and lead with respect to each other, and the pointer will swing back and forth over the scale.

Such an instrument would indicate a difference in frequency, but would give no information as to which one of the machines is fast or slow. Here is where the lamp comes in. One complete swing corresponds to a phase displacement from quadrature plus or minus to quadrature minus or plus, and this period also coincides with the period of incandescence or darkness in the lamp; therefore, the pointer will be seen only during every other swing, and will appear to rotate smoothly and silently in one direction. The direction of apparent rotation indicates whether the incoming machine is fast or slow and the speed of rotation is a measure of the amount by which the frequencies differ.

Just what takes place inside the instrument is best demonstrated by a diagram such as Fig. 4. At the top of the diagram are shown two waves, which differ in frequency. These two waves may be chosen to represent the currents in the primaries of the lamp transformer, then the resultant (shown dotted) may be considered as the current active in the lamp. Assuming that when the resultant current remains inside the shaded band the lamp filament will not be incandescent, the period of light and darkness is roughly determined by considering the points where the current crosses the band as transition points between light and darkness, which may be found from diagram.

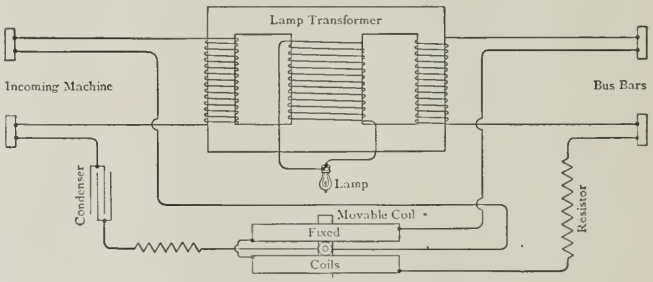


FIG. 3.—CIRCUIT DIAGRAM.

Assuming the fast wave to belong to the incoming machine, the currents in the coils of the instrument may be represented by plotting the slow wave exactly as it is shown at the top and plotting the fast wave 90 time degrees ahead of its original position to allow for the effect of the condenser. This has been done in the middle set of waves. The product of these current waves gives the torque active in the instrument, which is shown by the curve bounding the shaded areas. The sloping straight lines indicate the general trend of the average torque. They show that when the incoming machine is fast the pointer travels from left to right during the time the lamp is lighted.

The bottom set of waves is constructed in the same way as the middle set and shows that when the incoming machine is slow the pointer reverses the direction of apparent rotation.

The accuracy of this instrument is quite remarkable, es-

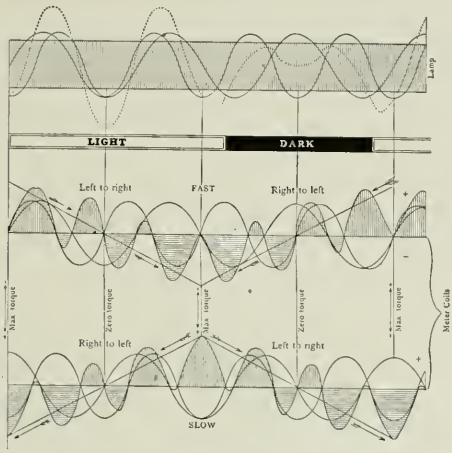


FIG. 4.—PERFORMANCE DIAGRAM.

pecially as regards its freedom from errors due to fluctuations in voltage or frequency. For example a 60-cycle, 110-volt synchroscope, when subjected to frequencies varying from 40 to 80 cycles per second and e.m.fs. varying from 95 to 130 volts, indicated true synchronism to within 1 deg. of phase coinci-

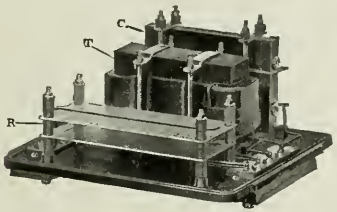


FIG. 5.—AUXILIARY DEVICE.

dence. This new type of synchroscope has been placed on the market by the Weston Electrical Instrument Company, of Newark, N. J.

The application of these principles to the operation of a synchromizing device is the invention of Mr. B. P. Romaine.

Personal

Mr. Lucien I. Blake, consulting engineer of the Submarine Signal Co., Boston, and inventor of the Blake machine for electrostatic separation of ores, intends to spend the coming year in Europe, traveling in the interests of the Submarine Signal Company and to inspect various ore treatment plants. Mr. Blake has made his home in Denver for the past few years, where certain problems in ore dressing have attracted his attention.

Mr. Warren F. Bleecker, of Denver, is spending some weeks in New York preparing for the erection of a mill in the San Juan district of Colorado, for the treatment of vanadium ores.

Mr. D. W. Brunton has returned to Denver after spending a few weeks at Hot Springs, Ark., where he was obliged to go for his health.

Prof. Bucherer, of Germany, during his recent brief stay in this country, attended the smoker at the New York meeting of the American Electrochemical Society. While he is now professor of theoretical physics at the University of Bonn and, by fundamental experiments, has largely contributed to the modern theory of relativity, Professor Bucherer was years ago among the pioneers of industrial electrochemistry, being connected for a year with Mr. Alfred H. Cowles in his early electrochemical development work.

Mr. J. J. Fitzgerald, of Florence, Colo., is acting as mill superintendent for the Frontenac Consolidated Mines, Ltd., at Black Hawk, Colo.

Mr. Mark R. Lamb, who recently returned from a South American trip, has been appointed manager of the South American branch of the Allis-Chalmers Company, with offices in Santiago, Chili.

Mr. Victor G. Hills, of Denver, has been appointed consulting engineer of the Scheelite Mines, Ltd., in Nova Scotia.

Mr. Frank Klepetko has been in Peru making an examination of the Ferrobamba copper property.

Dr. Karl Georg Frank, the American representative of the Siemens & Halske Company, has sailed for Germany and is expected back in about two months.

Chief Engineer Maleyka, Mr. Werner von Siemens, and **Dr. Puppe,** all of the Siemens & Halske Company, of Germany, are in this country to study American practice in the use of electric power in large metallurgical plants, especially electric driving of rolling mills.

Prof. Robert H. Richards will spend some time in the West this spring, visiting Montana and other Western States.

Mr. Jesse Scobey has acquired an interest in the ore-testing plant of Henry E. Wood, Denver, and will be in charge of the works in the future. Mr. Scobey's superintendence of this work will insure fair and accurate tests on ore sent to the plant.

Mr. George F. Waddell, mill superintendent of the Nevada Consolidated Copper Company, McGill, Nev., is making a professional trip among the Western copper concentrators to inspect the methods of slime treatment, with a view to making some improvements at McGill.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

317,246, May 5, 1885, to Edward P. Thompson, of Elizabeth, N. J., and E. P. Roberts and G. H. Pierce, of Cheyenne, Wyom., assignees.

Gold. Electro-deposits from a solution of its chloride The cell is a circular vessel, containing concentric annular cathodes of corrugated sheet copper. Between them is an anode consisting of depending electric-light carbons arranged in a circle. Steam is admitted to the electrolyte, heating and agitating it. The gold-plated cathodes are then removed to another cell containing a solution of sulphuric acid or copper sulphate and a current is passed from them to an intermediate cathode of copper. The copper of the former cathodes, now anodes, is thus dissolved and redeposited, leaving the gold.

332,705, Dec. 22, 1885, to Henry H. Eames, of Philadelphia, Pa.

Gold, silver, etc. Electrolytically chlorinates prior to amalgamation or lixiviation. The cell is a cylindrical iron vessel

lined with wood and having a steam-tight iron cover with manhole, safety valve and gage. The electrodes are opposed copper plates, secured to the inside of the tank and extending from top to bottom. A shaft passes up through the bottom of the cell, carrying a water-sealed depending cone with vertical stirrers. The electrolyte is a solution of sodium chloride, sulphuric acid, or iron or copper sulphate, into which the pulverized ore is charged. Steam is used at a pressure of from 20 lb. to 25 lb.

333,815, Jan. 5, 1886, to Michel Body, of Liege, Belgium.

Gold, silver, copper, nickel cobalt. The ore is subjected to preliminary treatment by exposing a heap of it wetted, to the atmosphere. Spent electrolytic liquor is preferably used for wetting. Or a sodium chloride solution, or calcium or magnesium chloride lyes, may be used. Or the pulverized ore can be roasted. The ore is then electrolytically dissolved at the anode of a cell containing a solution of a ferric salt and sodium chloride. The ores are thus dissolved by the action of the ferric salts, simultaneously converted into ferrous salts; the dissolved metal is deposited at the cathode, and the evolved chlorine re-oxidizes the ferrous salts, any excess assisting in dissolving the ores. The electrolytic cell is a rectangular vessel of Portland cement, painted inside and out with an impermeable coating. The inner portion of this floor is elevated and supports a carbon-plate anode. The four sides of the vessel are lined with carbon. A central compartment is provided by vertical inner walls, between the lower ends of which and the elevated portion of the floor are diaphragms of felt. The ore is placed in this central compartment, which contains a revolving stirrer, resting on the carbon anode, and the inner and outer compartments are filled with a solution preferably consisting of water, 100 parts; sodium chloride, 20; iron persulphate, 5; and surphuric acid, 5. The solution is introduced into the central compartment and flows upward and over its walls into the four outer compartments, escaping from their bottoms. The metal is precipitated on copper cathodes, depending into the outer compartments.

BOOK REVIEWS.

Engineering Chemistry.—A manual of Quantitative Chemical Analysis for the Use of Students, Chemists and Engineers. By **Thomas B. Stillman**, M.Sc.,Ph.D. Late Professor of Engineering Chemistry in the Stevens Institute of Technology. Fourth edition; 8vo, 744 pages, 174 illustrations, price $5.00. Easton, Pa.: Chemical Publishing Company.

The excellences of this work are well known to all students of chemistry. This edition has been brought up to date in all departments. The chapters on coal, alloys, lubricating oils, and paints have been amplified by the addition of specifications for the purchase of these materials. The subject of pyrometry has been revised by Messrs. E. A. Uehling and W. H. Bristol. A chapter on "Foundry Chemistry" has been contributed by H. E. Fields, one on "Official Methods of Sampling Iron Ore" by Chief Chemist J. M. Camp, of the U. S. Steel Corporation, and one on "Acetylene" by R. E. Brückner. The new edition is, therefore, a considerable improvement on the valuable and much appreciated earlier edition, and will prove an acceptable addition to chemical engineering literature.

Iron and Steel: A pocket encyclopedia, including allied industries and sciences. By **Hugh P. Tiemann**, B.S., A.M. 354 pages, 43 illus. Price, $3 net. New York: McGraw-Hill Book Company.

This little book is a combination of a dictionary, a cyclopedia and a handbook. It is alphabetically arranged, giving under each heading a concise definition and description (sometimes illustrated) of the apparatus or process in question. Cross references are numerous. With respect to the subject matter included the book is absolutely up to date.

That there should be a large demand for such a book is admirably brought out in an introduction written by Prof. Henry M. Howe, of Columbia University, from which we quote the following:

"The jargon of the millman, like that of the philosopher, is a deplorable necessity. It is a collection of invaluable special tools for special men doing certain work. When the dentist and the obstetrician regard the unfathomable mysteries of each other's 'kits,' each has the consolation that he need not attempt the fathoming. But, alas, you and I cannot thus escape each other's jargon, for the metallographist must needs learn from the millman and the millman from the metallographist, and each has become the slave of his own tools, his own jargon. He talks and perhaps thinks in terms of it, if indeed we think in any language, which I doubt. At least, if he thinks in any language it is in his jargon. Life is too short, patience too flimsy, to permit our forcing our thoughts into others' minds by means of any tools other than our own jargon. The author gives me an admirable case in point.

"Foreman: 'How does this steel work?'

"Heater: 'If you don't wash it, it won't clean.'

"This is 'shorthand' for: 'The iron oxide or scale which forms on the surface of this steel adheres so firmly that, unless it is heated so highly that it melts, some of it will cleave to the metal during the operation of rolling and hence will deface the finished plates into which the steel is to be rolled.' The heater's words are not English; they are jargon, and it is proper that they should be. If he persisted in translating them into English as I have done, and in talking English in general, he would simply justify the dismissal which would surely come. The guild has evolved its jargon for its own use. To replace it with the king's English would be as unwise as to replace it with French or modern Greek, or to replace 'shorthand' with 'longhand' in the reporting of debate.

"When the metallographist and millman meet it is as the meeting of French and Greek. The millman scolds the metallographist; yes, and I have had to endure with my scanty patience many such scoldings for deliberately inventing jargon, useful for the metallographic guild, but a stumbling block to the millman. Of course, it is not I, but the nature of things that ought to be scolded; but then it is pleasanter and less transparently foolish to scold me, especially if I have previously earned your gratitude by disentangling some of your fallacies and sophisms.

"Now, here comes the gallant author to the aid of fumbling metallographist and irritated millman. With infinite pains, ingenuity and skill he blesses both where they had banned each other, and enables the brothers to dwell together in harmony, unfurling to each the jargon of the other by means of a tri-lingual dictionary, translating the jargons of both into the common language, English."

The book has even a wider scope. While it forms, as Professor Howe points out, an admirable conecting link between millman and metallographist in the steel industry, it should appeal with equal force to the large number of chemists, electrochemists, mechanical engineers and others who, while not primarily active in the iron and steel field, are necessarily interested in its development and are often puzzled by the jargons of the steel millman or metallographist, however familiar may be the conceptions expressed under other names. In all such cases this little book will prove an easy and effective key.

If a suggestion may be made for future editions—which should surely be numerous—it is the addition of references to existing literature; not a complete bibliography, but in each case a reference to one or two of the principal publications on the subject. The increase in volume would hardly be prohibitive, while such references would often be very handy, even for the experienced steel man. As it is, this little volume is certainly excellent.

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The Foundry Convention at Pittsburgh.

The sixteenth annual convention of the American Foundrymen's Association, held late last month in Pittsburgh jointly with the newer bodies, the American Brass Founders' Association and the Associated Foundry Foremen, illustrated the enormous progress which has been made in the art of iron and steel founding in the past 15 years. According to the viewpoint one may attribute the remarkable growth of the American Foundrymen's Association to the progress in the art, or may attribute a large part of the progress in the art to the association movement. It is really a case of action and reaction. They have grown together, and in observing such results it is immaterial what have been the precise causes, for whatever they may be they are at work and are producing better and better results every year. Certain it is that the free exchange of information which the association movement has brought about has been of inestimable benefit to all connected with the foundry industry.

Not the least important, and perhaps the most important, feature of the foundry convention at Pittsburgh was the colossal exhibit of machinery and materials. Papers can be published and read at any time; they pass through the mails and can be distributed readily. Not so with the machinery which visiting foundrymen were able to see in actual operation.

Two Causes of Mill Failures.

In the domain of ore treatment the history of large companies is seldom marred by those financial and technical failures that characterize lesser concerns. Expressed in common phraseology, the principal reason for this is that the large companies "go at it right." In other words, they provide adequate capital and employ the best technical skill available. And these two points form the basis of a fundamental lesson.

On the financial side there is a critical period in the early history of a mill, which many small companies do not anticipate and which they are not able to weather—a sort of peak load of financial stress which bears them down to disaster. And the strange feature of it to them is that this period comes when they think they have provided ample funds and should commence to make money. Given a new mill, free from debt and ready to run, it is difficult for them to realize that their troubles have just begun and that very substantial sums frequently must be spent in making alterations and tests before real operations can begin. Thus, while many small companies are able to build and pay for their plants, they have not sufficient money to tide over the critical period of alteration and adaptation which ensues. It is almost safe to say that a small company should have a reserve equal to one-third or one-half the cost of its mill for use in making the plant a real success. Large companies have these experiences to a certain degree, but they are not heard of as failures, because money is available to make them ultimate successes.

In the second instance we may lay down certain axiomatic facts for the guidance of the small concern. First, it is poor economy to "save" the professional fee of a competent metallurgist or engineer when the sum thus involved bears only a small ratio to the total expenditure. Second, excellent business men in other lines do not necessarily make good managers of metallurgical plants, and it is likewise a mistake to put in charge of operations a man whose chief qualifications are that he is a nephew of one of the directors and unemployed. Those who have had experience in such matters know that it pays to be guided by specialists and that many mistakes and perhaps ultimate failure are avoided by employing an engineer who can make a complete study of the situation before a stroke of construction work is done. The whole matter sums up to this: that if adequate financial reserve cannot be supplied and competent professional advice afforded, the venture had better be abandoned before it ends in almost certain failure.

Experiment and Practice.

A principle in testing metallurgical processes on a small or laboratory scale is that the conditions of the experiment shall comply, as far as possible, with those which will be encountered in actual practice. In every point that the conditions of the test depart far from those of practice there is a possibility of failure, or at least of disappointment, under commercial operation.

In the cyanide process, extraction usually is relative to the degree of fineness to which the ore is ground; the finer the subdivision the better the extraction. But in commercial operation we strike an equilibrium between the cost of grinding and the value of the gold extracted, beyond which it is not profitable to go. At that point finer grinding ceases, and we arrive at the best possible commercial conditions. We have reason to believe that in laboratory experimental work this condition in grinding is neglected to a degree, or rather that the laboratory conditions are not made comparable to commercial conditions, with the result that commercial operation does not come up to the expectations indicated by the tests.

One reason for this is inherent in the different means used for grinding in laboratory and mill. Laboratory grinders usually will produce a greater proportion of undersize of any given mesh than will the commercial machines used in the mill. Hence, if the laboratory tests indicate a suitable extraction when all of the ore is put through a 30-mesh screen, for obvious reasons that standard of excellence scarcely will be attained in practice. By the time a laboratory sample is ground on a bucking board or in a disk grinder so that the whole sample will pass a 30-mesh screen, it probably will contain a greater proportion of, say, the minus 150-mesh size than will be discharged by a Chilean mill fitted with 30-mesh screen. Further, if the ore is of such a nature that the best value is contained in the fine pulp or slime the laboratory test will show a better cyanide extraction than can be obtained in practice, owing to the prohibitive cost of regrinding to equal the laboratory conditions. We believe that a closer control of this tendency toward excessively fine grinding in the laboratory will result in a closer comparison between experimental and practical results.

Reduction Works as Public Utility Corporations.

Scarcely a biennial session of Western legislatures passes without a spasmodic effort being made to "regulate" the companies engaged in treating custom ore. Some of the proposed legislation has been well meant, but futile, while more of it has been inspired by political motives and without regard to the welfare of metallurgy. Hitherto smelting companies have maintained an active opposition to this sort of activity, at the expense of time and money, and have anticipated the necessity of combating adverse legislation with each recurring biennial period.

The latest attempt to control ore reduction is the passage of a bill by the Colorado Legislature submitting to the people a constitutional amendment which, if adopted, declares all smelters, mills, reduction plants, and ore-treatment works of all kinds to be public-utility corporations, and, as such, amenable to legislative regulation and control as regards the tariff charged for their work. Aside from being poorly drawn as to exact wording and meaning, and leaving out of consideration the fact that the people of Colorado will scarcely adopt the amendment, the bill possesses certain potential features which are undesirable.

If the amendment should be adopted, future Legislatures would be empowered to regulate ore-treatment rates. In the smelting business particularly this is a technical matter, owing to the necessity of having the proper variety and quantity of ores to make suitable smelting mixtures. In chlorination and cyanidation it is frequently necessary to discriminate in favor of clean ores and against those containing an undesirable element or substance. In all ore treatment the gross value of an ore affects treatment rates and recoveries.

Considering these things it will be apparent how difficult it would be to regulate rates for ore treatment, save through the medium of a board of experts who would have to be as competent technically and as well-informed commercially as the custom companies themselves. In other words, in order to act competently such a board would have to be in possession of all the professional and business knowledge of the companies, and this is obviously impossible. Further it would be incumbent on such a board to reconcile the differences of mining companies in different districts, producing ores of diverse metallurgical character and justly subject them to discriminating rates of treatment. If it is not clear now that the entire procedure would result in a hopeless tangle, the fact will be appreciated when the attempt is made to put the law into effect.

The mere adoption of the amendment, however, would be a menace to metallurgical progress even though future Legislatures did not use their powers and try to regulate rates. New companies contemplating entering Colorado would hesitate about engaging in an industry over which hung a possibility of disturbance and interference.

Perhaps those against whom the bill was directed realized the futility of the whole scheme, for no obstacle was placed in the way of the bill's passage. Should its provisions finally come into full effect, it is difficult to say what would be the action of the principal smelting companies. Instances have been known where companies have seized on a suitable pretext to quit business altogether and abandon plants, and as the smelting industry in Colorado is in none too good condition it might be fur-

ther hampered by the abandonment of plants. Granting that metallurgical companies need regulation in Colorado, the cause will hardly be improved by the present effort.

Efficiencies in Electrochemistry

We believe it was Mr. James Swinburne who once remarked that efficiency is a byword of politicians and that when engineers use the term they should speak of efficiencies rather than efficiency. It may not be out of place to give here a concise sketch of the almost bewildering variety of efficiencies with which we have to deal even in such a restricted field as electrochemistry. The subject is of timely interest in view of the discussion, or, rather, repeated discussions, at the recent convention of the American Electrochemical Society, where besides various shades of intermediate opinions two extreme views found expression—one to the effect that a manager of a works does not know what he is doing except he can state the efficiency of operation in per cents, the other to the effect that in most cases of practice efficiency is so vaguely defined that it means nothing and that the term should not be used at all.

In any case, when we speak of an efficiency we compare a figure obtained in practice with a theoretical figure corresponding to ideal conditions. In electrolytic processes we may compare the quantity of material reduced at the cathode or oxidized at the anode in practice with the quantity which should be reduced or oxidized according to Faraday's law under ideal conditions. Faraday's law is always exactly fulfilled. "Under ideal conditions" means, therefore, such conditions that only the one desired reaction and nothing else occurs at the cathode or anode. For instance, according to Faraday's law, every 1000 ampere hours deposit from the solution of a bivalent copper salt 1.2 kilogram of copper if this is the only cathodic reaction. If in practice we get only 0.9 kilogram of copper, we say the cathodic amperehour efficiency is 75 per cent. This means that the balance of 25 per cent. of ampere hours are consumed to produce some other reaction that is not wanted and are therefore wasted. The figure of amperehour efficiency has therefore a distinct well-defined practical meaning; it states to what degree the one reaction which we desire to obtain really goes on. But even here we should distinguish between two efficiencies, the anodic and cathodic amperehour efficiencies, since there is no reason why both should have the same value.

However, of more direct practical bearing is the watthour efficiency, since not the amperehours, but the watthours, are paid for. In this case we, therefore, should give the energy given out in practice by a galvanic cell or the energy consumed in practice by an electrolytic cell in per cents of the energy given out or required according to theory under ideal conditions. A galvanic cell works "under ideal conditions" when the whole available energy is changed into electrical energy (that is, if there is no local action, if the Joulean heat developed is negligible, etc.). But what is the "whole available energy"? In practice it is customary to use simply the reaction heat. This is absolutely wrong in principle. The available energy is the "free energy of reaction," not the reaction heat, to be correct in principle. This subject was discussed at some length in an editorial on page 2 of our Vol. II in connection with a paper by Prof. Lorenz (Vol. II, p. 15); hence there is no necessity to discuss it here further than by pointing out one example. This is the rather rare case that an endothermic process may be used for the generation of electrical energy in a battery. In one example referred to in Prof. Lorenz's paper a cell, according to Thomson's rule, should require the expenditure of 0.14 volt to be impressed from the outside, while in reality it gives out spontaneously an e.m.f. of 0.32 volt; when the reaction goes on at constant temperature heat is supplied from the surroundings to the system, and this heat is partly used up in supplying the heat of the chemical reaction and partly changed into electrical energy. It would clearly be foolish to define watthour efficiency in such a case on the basis of taking the heat of the reaction as available energy. This whole question is intimately connected with the question of Thomson's rule versus the Gibbs-Helmholtz equation. The latter is the correct one in principle. Nevertheless the former is mostly used and is, in the majority of cases, a good enough approximation for practical purposes. In an analogous way we may continue to use the old watthour efficiency, employing the reaction heat as basis of calculation, if we always remember that it is only an approximation and that in some cases it may lead into unsurmountable difficulties.

However, industrial electrochemistry is a much broader field than the restricted one which we have discussed so far. It includes all electric-furnace processes, whether they involve any chemical reaction or not. Further, industrial processes are in general not constant-temperature processes, as has been silently assumed in the preceding paragraphs. It is, therefore, important, as Dr. J. W. Richards pointed out at the meeting of the American Electrochemical Society, to distinguish two things: First, the process involving a temperature change, like the melting of a cold charge; and, second, the process of keeping a molten charge at a constant temperature. In the first place an exact definition of efficiency is possible; to calculate it numerically, it is necessary to know or to assume specific heats, latent heats of fusion and perhaps vaporization, and also to take into account any chemical reactions which occur.

On the other hand, in the second case—where a molten charge is kept at a constant temperature—it seems impossible to give in an analogous way a useful definition of efficiency, simply because the ideal condition in this case is that there is no energy required (assuming that the charge is allowed to "rest" and that no chemical reaction takes place). Hence, on the basis of comparison with an ideal furnace, all real furnaces are alike in having zero efficiency in this respect. What is possible, however, is to compare real furnaces among themselves by stating, for instance, that different furnaces, otherwise alike, require so and so many kw-hours per ton of charge to maintain their temperature constant. As Mr. Lidbury pointed out at the meeting, such comparative figures are all that is needed and there is no need of speaking of efficiency at all. What such a figure represents is strictly specific power consumption, not efficiency. But if people want to call it an efficiency we do not see how they can be prevented. The jargon of the engineer cares very little for the rules of the philologist and logician. This much, however, must be insisted upon: That when an efficiency figure is given it should be accompanied by an exact definition of what is meant and by a statement of the assumptions made and of the figures used in the calculation.

American Institute of Chemical Engineers.

The summer meeting of the American Institute of Chemical Engineers will be held at Chicago from June 21 to 24. All the sessions will be held at the Congress Hotel.

Dr. Frerichs will present his presidential address on methods of attacking chemical industrial problems. Dr. S. P. Sadtler will present the report of the committee on chemical engineering education.

The program of papers is as follows:

Wednesday, June 21, 11 a. m.:

Report of committee on chemical engineering education. By Dr. Sam. P. Sadtler, chairman.

The four-year chemical engineering course. By. Prof. J. H. James.

Wednesday, 8 p. m.:

Presidential address by Dr. Frerichs.

Friday, 9:30 a. m.:

Industrial chemical calculations. By. Prof. J. W. Richards.

Rapid determination of tin in food products. By Dr. Edw. Gudeman.

Distillation on the continuous system. By. Chas. L. Campbell.

A new system of lead and silver lining for chemical apparatus. By Chas. L. Campbell.

The Institute and the United States patent system. By Dr. Wm. M. Grosvenor.

Friday, 7:30 p. m.

Manufacture of gelatine. By Ludwig A. Thiele.

The practical value of calorific tests on anthracite coal. By S. F. Peckham.

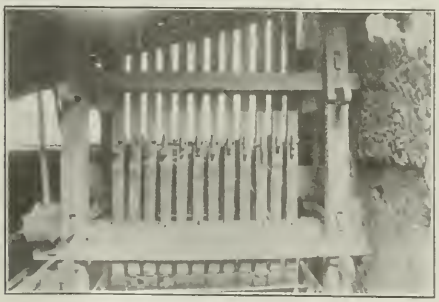
Two methods of testing asphalt. By. S. F. Peckham.

A large number of interesting excursions has been arranged for the afternoons of Wednesday and Friday, and for the whole days of Thursday and Saturday. These include the plant of Swift & Company at Union Stock Yards, the Corn Products Refining Company at Argo, Ill., the laboratories of Dearborn Drug & Chemical Works, the By-Products Coke Corporation at South Deering, the Carter White Lead Company at West Pullman, the Indiana Steel Company at Gary, Ind., the Universal Portland Cement Company at Buffington, Ind., the Illinois Steel Company at South Chicago, and the Standard Oil Refinery at Whitney, Ind.

Dr. J. C. Olsen, Polytechnic Institute, Brooklyn, is the secretary of the Institute.

Old Mexican Stamp Mill.

The accompanying illustration is interesting as showing the primitive practice of the Mexicans in stamp milling gold ore.



OLD STAMP MILL AT TEZICAPAN, GUERRERO, MEXICO.

The photograph was taken at Tezicapan, Guerrero, Mexico, by Mr. George J. Hough, and shows twelve stamps of wooden stems and iron heads. The stems are about five inches square, and the heads of cast iron are about the same dimensions in cross section and twelve inches long. One of the stamps in the illustration is shown detached from the stem, into which it was the practice to insert it with wedges. The cam shaft was a large, round timber with spikes or stems driven into it. As the shaft revolved these stems engaged the keys on the stamp stems and raised the stamps a short distance. Perforated cowhide was used as a screen, and portions of hide tanned with the hair on were to be found in a good state of preservation near the mill.

The Western Metallurgical Field.

Metallurgical Education.

Diversity of opinion exists as to the best methods of teaching metallurgical processes to students in our schools of mining and metallurgy, particularly with reference to the equipment and apparatus used. On the one hand we find a tendency to favor work on a large scale with standard size apparatus and machinery; and on the other a preference for small operations with well-designed models of the standard machines. The first idea is finding expression at home and abroad in the Imperial College of Science and Technology, South Kensington, London, and at the Colorado School of Mines, Golden. At each of these schools there is being installed a metallurgical laboratory and testing plant which compares in size and equipment with many mills built for commercial operation. The machines are standard size and the operations are conducted with tons of ore instead of pounds. The sponsors of this method of teaching are enthusiastic in their belief that it is the best for students, as it gives them opportunity to handle standard machines in a commercial way. They believe that the student can become familiar with principles by applying them in practice.

The other method has been far more extensive, for obvious reasons of expense of installation and cost of securing quantities of ore to work on, and perhaps those who advocate the new method regard the old as less efficient. Nevertheless, there is much to be said in favor of small operations for purposes of instruction. It has been the method pursued at the Massachusetts Institute of Technology, and as this school has acquired a reputation for the way in which its teaching work is done it may be well to consider the reasons which underlie the methods used there. The following is taken from recent correspondence on the subject:

The question of large versus small machines for teaching students in the laboratory is largely a question of the ability of the machines themselves. If the machines are well adapted and perfectly made the small machine can teach jigging and table work and small plant work just as well as the large machine. At the Institute of Technology the only large-size machine used is the vanner, and this has a width of 4 ft. instead of 6 ft., as is now common.

The question of size of machine seems to be partly a personal matter with the teacher, who may prefer large-size machines to small ones, and partly a question of locality. If ores can be had and given to the school by miners the large-size machines might be preferable, but where ores have to be brought a long way the small machine serves perfectly.

There are two other questions which come up in connection with settling this matter. One is the feeling of the profession of the neighborhood of the school, as to whether they are of the opinion that it is necessary to have large machines. If the public opinion says that the school must have large-size machines, then it must be so or the school will lose caste. The other is the question of testing ores for mines. If a school is located in a mining district it is pretty certain to have ores sent to it with the idea that the results of the tests shall be given to the owners of the ore, either for a small fee or as a return for the favor done in sending the ore, so that the mine may profit by the test, while the school profits by having the ore to work on.

It is the opinion at the Institute that if small machines are carefully designed there, they will do work that will rank with the very best large-scale machines ever made. It is further believed that the students working on small-size machines can do their preliminary illustrative work when they are learning what the machines are, at the same time they are listening to the lectures, and that this can be done just as well on small machines as on large. Finally, students can do just as careful testing work for their theses on small machines, and possibly better work, than they can on large.

The argument may be summed up by saying that if public opinion, which controls matters in the school, favors large machines, and if it will not ruin the treasury to buy these machines and pay for large quantities of ore to run them, then it may be best to buy them. If, on the other hand, public opinion in regard to this matter prefers leaving the selection to the teachers in charge, and if the cost of ore is a serious item, then the small machines would be selected by those in charge at the Institute.

On the score of ore tests, it is believed that it is a matter of extreme value to a mining company to have a lot of ore tested carefully on small-scale machines which have been carefully designed and properly run. From the results so obtained the company can get figures which will serve as a standard of perfection for them to try to attain on their large-scale operations; and if they attain the perfection which it is believed can be shown on well-designed small machines, they can feel satisfied that they have done the best that can be done according to the plan followed.

The Bureau of Mines and Metal Mining.

For various logical reasons the first work of the national Bureau of Mines has been in connection with the coal-mining interests. At the last session of Congress an effort was made to secure an appropriation for work in connection with metal mining, and $50,000 was set aside for such work as should be decided upon. It appears probable that this sum will be expended on the establishment of a western laboratory for investigations in metallurgy. Report has it that the laboratory will be located at San Francisco, and that its first work will be in the nature of conservation in metallurgical processes. Such problems as the prevention of smoke and fume loss, perfection of processes to minimize tailings losses, investigation of promising new fields of metallurgy and electrometallurgy, etc., probably will occupy the attention of this first laboratory. Prof. Frederick G. Cottrell, of the University of California, has been mentioned as the chief of the new laboratory, and judging from his former success as an investigator in metallurgical and chemical engineering the choice would be a good one.

Press comments are confused as to the function of a government laboratory, and some notion exists that it will be a free assay office and general information bureau. Certainly this is a trivial view to take of the matter. The mining industry has hitherto expended vast sums in overcoming obstacles in ore treatment and in investigating matters which promised uncertain returns. The effect has been good but of limited usefulness; because, being privately obtained, the information also was privately held. Again, many companies have duplicated work along certain lines, because each had to gain its own information or drop behind. Finally, much of the progress has been made by large companies, and is applicable only to large-scale operations and of little value to a small operator.

Thus the industry has progressed because some companies were rich enough to spend sums in investigation which is denied the larger part of the operators. It is believed that government laboratories can undertake many lines of investigation which will obviate some of the objections just raised and which will result in a general dissemination of information that will benefit the entire metallurgical industry.

Missouri Zinc Fields.

The unsettled state of affairs in Mexico has had the effect of sending more buyers into the zinc districts of Missouri, resulting in improvement in market conditions owing to greater competition. Additional interest in this district arises from the large operations of the American Zinc, Lead and Smelting Co., which is one of the largest factors in the field. Several weeks ago this company acquired under lease and bond 640 acres of land in the sheet-ground district near Webb City.

Following the policy of mining and milling its own ores, the company plans to erect a mill of over a thousand tons daily capacity, which will be the largest in the district. It is said that the mill will be patterned after the one owned by the same company at Plattsville, Wis., where the Huff electrostatic separators are in use. Among the other reported activities of this company is the erection of a new zinc smelter at Collinsville, Ill.

International Smelter, Utah.

The Tooele plant of the International Smelting & Refining Co., originally built and started as a copper smelter, is to be enlarged by the addition of two lead furnaces of a combined daily capacity of 500 tons. This is a part of the original plan of the company, although it was not known when the mining situation would warrant the starting of a new lead smelter in Utah. The company has been buying lead ores for some time, and a large tonnage is now on hand, which will provide at least six months' supply when the furnaces are ready.

This places the International company in competition with the American Smelting & Refining Co. and the U. S. Smelting Co., as regards lead ores as well as copper. The former operates a lead plant at Murray and a copper plant at Garfield, while the latter has both lead and copper smelters at Midvale. Apparently the general ore situation in Utah is improving, and with active competition on the part of three large corporations there should ensue a period of prosperity for the entire industry. In this respect Utah has an advantage over Colorado, which, in the height of its mining activity, had only one company bidding for smelting ores.

The Non-Ferrous Metal Market.

Dullness has characterized the market for all non-ferrous metals since our last report. Business has been so low in some cases that quotations are but nominal, and such business as has been transacted has been at shaded prices.

Copper.—Consumers are pursuing a policy of buying for immediate needs only, and the general belief is that most of them are not as well covered as good business policy would dictate. With a rise in prices there will undoubtedly be heavy purchases, especially if the upward tendency should appear permanent. Lake copper is quoted at 12⅛ cents to 12⅜ cents and electrolytic at 11.9 cents to 12 cents.

Lead.—The market is very dull and there has been a lower tendency in prices. The St. Louis market quotes lead at 4.22½ cents to 4.25 cents, and New York at 4.37½ cents to 4.40 cents.

Spelter.—Business has been negotiated at figures below the quotations, and the market has shown a tendency to go lower. Even though the supplies of spelter are required for work in this country, manufacturers show no disposition to buy heavily at present prices. St. Louis spelter is quoted at 5.20 cents to 5.22½ cents, and New York at 5.35 cents to 5.37½ cents.

Tin.—The market has been unusually quiet, and consumers can provide for their needs at figures below the import basis. The quotation for May tin in New York was nominally 42 cents.

Other Metals.—Aluminium prices are unchanged at 20½ cents to 21 cents per pound for No. 1 ingots. Antimony has moved slowly, and the prices range from 8 cents to 9.6 cents for various grades. Quicksilver has been steady at $46.50 per flask of 75 lb. in New York and $46 in San Francisco.

The Iron and Steel Market.

As this report is written the price structure in the finished rule a day or a week hence. On May 24 the Republic Iron & Steel Company began openly to sell steel bars at 1.25 cents, Pittsburgh, the recognized market having been 1.40 cents. It formally announced its abandonment of the policy of co-operation with other manufacturers in the matter of steel bar prices, directing attention to the impracticability of maintaining the 1.40-cent basis on soft steel bars when iron bars and hard steel bars rolled from old rails were being sold at prices so much lower, and intimating that there had been irregularities on the part of some other producers of soft steel bars.

Iron bars have sold as low as 1.20 cents, Chicago, while the 1.40-cent price on steel bars has meant 1.58 cents, Chicago. Bars rolled from old rails have sold at from 1.20 cents to 1.25 cents at various mills in the central West, and it has been quite obvious that the 1.40-cent, Pittsburgh, price on soft steel bars has been out of line.

Ultimately the break in the steel bar price is likely to lead to a general revision of prices of semi-finished and finished steel, but the readjustment may perhaps proceed slowly. In some respects the condition resembles that in February, 1909, when a general and rapid decline commenced, and in other respects the condition is quite different. In the previous instance there was a definite understanding among a number of producers that the entire steel market, outside of steel rails, should be opened; in the present instance the move is taken by a single independent steel interest, which does not make a full line of product. It is one of the heaviest producers of steel bars and is also prominent as a seller of billets and sheet bars, as well as of railroad spikes, but plates, shapes and wire products it does not make, while it has only lately commenced the manufacture of pipe.

The last week in April and the first three weeks in May constituted the dullest period in the booking of orders for finished steel products which the trade has seen for a very long time. The market seemed to be seized with an almost complete paralysis, the meager tonnage booked being far below any reasonable estimate of the minimum current requirements of the country. From this situation an improvement could not but result, and before the first break in steel prices there were signs of a slight improvement. The Pennsylvania Railroad System, which on April 1 issued general orders to discontinue shipments of rails and track supplies, rescinded the orders and began receiving material again. Several other roads which had issued somewhat similar orders began to take increased shipments, and some new buying began to appear.

By common consent the iron and steel trade has concluded that the second half of this year will be a large one from a tonnage standpoint, although at no time have prospects been that even fairly high prices would be realized. With a long period of liquidation in stocks after the unhealthy boom of the late months of 1909 the country has found itself bare of material in the hands of mills, jobbers and manufacturing consumers. Financial conditions are good and the country has had four years of growth since the last period when the capacity was fully employed.

Pig Iron.

The Southern pig-iron market dropped 50 cents a ton early in May as to deliveries in the second half, and fairly large sales were made for delivery to the end of the year on the basis of $10.50, Birmingham. There are predictions that this figure will prove the turning point after the long decline, which began in October, 1909, after the sharp rise just preceding. In the East basic iron sold down to $14.50, delivered Philadelphia district. In the Central West basic iron declined on practically each sale made, until $14, delivered Pittsburgh, or $13.10, valley, was done without much difficulty. Foundry iron has remained nominally at $13.75, valley, but buyers have no confidence in the market and pursue a strictly waiting policy. The Bessemer iron is held nominally at $15, valley, but a few odd lots have been sold by brokers at concessions of from 15 cents to 25 cents.

Production.

Production of pig iron and finished steel has declined steadily since about April 1. Pig iron is being made at the rate of between 21,000,000 tons and 22,000,000 tons a year, against a rate of 26,000,000 tons on April 1. In March finished steel was being made at the rate of about 75 per cent of total capacity, but at the close of May steel works operations do not average over about 50 per cent, and the decrease may continue through June. The almost universal expectation, however, is that the turning point will be reached by midsummer.

CORRESPONDENCE.

Generating Electricity from Heat.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The writer desires to point out a new possibility for converting heat into electricity by way of the primary battery.

It is generally known that some metals like copper and silver are decomposed by hot sulphuric acid with evolution of sulphurous acid. The latter is, however, the result of a secondary reaction and due to the reduction of the hot sulphuric acid by nascent hydrogen.

$$Cu + H_2SO_4 = CuSO_4 + H_2 \quad (1)$$

$$H_2 + H_2SO_4 = 2\,H_2O + SO_2 \quad (2)$$

Hence there is no material difference between the reactions of copper and zinc with sulphuric acid. The greater affinity of zinc for the acid sufficiently accounts for the fact that here the hydrogen is liberated at ordinary temperature, while in the case of copper a stronger acid and a higher temperature are required. By amalgamating the surface of a piece of zinc the evolution of hydrogen is much diminished; in other words, its chemical action on sulphuric acid is weaker, yet the faculty of the zinc to send its ions into solution is even greater when alloyed with mercury.

By raising the temperature of sulphuric acid in which a piece of copper is immersed we increase its tendency to pass into the ionic state, and it is probable that the electric pressure will rise more or less proportionately with the increment of temperature up to the moment when the SO_2 gas is liberated, it being then a maximum.

For the present it will be sufficient to indicate that there exists an e.m.f. between copper and sulphuric acid at a temperature somewhat below that at which SO_2 would be set free. To make this e.m.f. available for practical purposes we must avail ourselves of another electrode which is surrounded by a suitable depolarizer and the latter should be prevented from freely intermixing with the sulphuric acid. By fulfilling these requirements we create a modification of the Daniell battery.

Using like Daniell a solution of sulphate of copper as depolarizer in which a copper cathode is immersed, and using instead of the zinc an electrode of copper surrounded by hot sulphuric acid, we have as an interesting feature a battery with electrodes of the same material.

Since the hot electrode will always dissolve while the cold one will receive the deposition of metal from the hot one it is evident that the migration of the ions will gradually cause an accumulation of $CuSO_4$ about the dissolving copper anode and of free H_2SO_4 about the copper cathode upon which copper is deposited.

These conditions bring about a decline of the available voltage of the cell and in order to restore it to the original intensity it is now necessary to heat the compartment which before was kept cool and vice versa. An interchange of the hot and cold solutions by convection currents must be prevented, of course.

Since the cycle of reactions is complete the sole agent for the electric energy produced by this cell is the inequality of temperature prevailing at its electrodes.

Waterbury, Conn. R. V. Heuser.

Comparative Grinding Experiments with a Chilean Mill and a High-Speed Roll at the Detroit Copper Company's Concentrator, Morenci, Ariz.

BY RUDOLF GAHL, PH.D.

Several interesting articles have been published recently describing experiments carried out to establish the most economical method of crushing and grinding ore. Most of these experiments, as for instance those undertaken by Mr. Caldecott, deal particularly with the reduction of ores preliminary to cyanide treatment. Very few experimental data are available referring to the reduction of ore preliminary to concentration.

Some tests were made several years ago at this plant by Mr. Frank Probert, with the idea of determining whether Chilean mills or high-speed rolls are more suitable for grinding in concentrating plants, and his results were published.*

More recently the installation of Chilean mills by the Utah Copper Company has been made the subject of very severe criticism. In this connection the following experiments may be of interest which are essentially a very careful repetition of Mr. Probert's tests and were carried out in the course of the past year.

The Chilean mill used in these tests was a 5-ft. Monadnock mill made by the Trent E. & M. Company running at a speed of 36 r.p.m. The rolls made by the Gates Iron Works have a width of 15 in., a diameter of 36 in., and are operated at 140 r.p.m. The product of the rolls was in the first test screened by a shaking screen operated by a Wilfley motion, in the later tests by a trommel. The oversize was fed to an elevator and returned to the rolls. Operated in this manner the rolls made only one product, viz., the undersize of the screen, which could be compared to the product as discharged by the Chilean mill.

This undersize was carefully sampled for a number of days. Of a composite sample representing an average of the days over which the sampling was extended a careful screen analysis was made.

Test No. 1.—Feed: Tailings from jigs working on a trommel product, passing through 7 mm and remaining on 5 mm trommels.

Screens: 2½ mm round hole screens on Chilean mill, and on shaking screen screening roll discharge.

Average load: 109 tons per machine per day.

Oversize from shaking screen: 9.3 per cent of feed.

The results of screening the mill discharge and the roll screen undersize are tabulated in Table No. 1 and represented graphically in Fig. 1. The conclusions to be drawn from these figures are that when the same screen cloth is used on a Chilean mill and on the screen which screens the roll discharge the rolls produce very much less slime than the mills.

TABLE I.

	Rolls.	Mills.
0–20	23.10	11.00
20–30	16.00	7.00
30–100	23.40	21.20
100–200	6.60	9.30
200–∞	30.90	51.50
	100.00	100.00

The object of the following test was to determine whether the same difference in the sliming of the rolls and mills exists also when a finer material is reground.

Test No. 2.—Feed: Tailings from jigs working on a trommel product, passing through 5 mm and remaining on 2½ mm trommels.

Screens: 2½ mm round hole screens on Chilean mill, and of shaking screen, screening roll discharge.

Average load: 65.7 tons per machine per day.

Oversize from shaking screen: 6.0 per cent of feed.

Eng. & Min. Jour., Vol. LXXIX, 1905, p. 1089.

The results of screening the mill discharge and the roll screen undersize are tabulated in Table 2 and represented graphically in Fig. 2. (p. 294). The same conclusion may be drawn from these figures as those from test No. 1, that when the same screen is used for rolls and mills the rolls produce very much less slime than the mills.

The same result has been obtained by other experimenters and the conclusion been drawn that, inasmuch as the size of the maximum grains was the same for both machines, and the quantity of the undesirable slime so much smaller for the rolls, they were better suited for this class of work. In drawing this conclusion it has been taken for granted that the economic

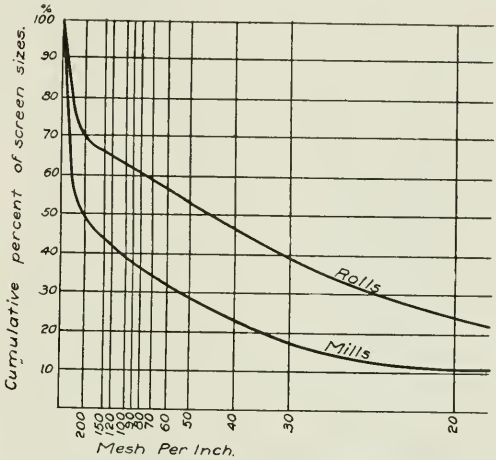


FIG. 1.—GRAPHICAL REPRESENTATION OF SCREENING RESULTS OF TABLE I.

mineral has been liberated equally well by the rolls as by the mills.

In order to determine whether this assumption is justified the different screen sizes were concentrated on a small experimental Wilfley table, with the exception of the material passing through 200 mesh. The amount of concentrates that could be expected to be saved from this slime was determined by panning. The results are also represented in Table 3 (p. 294).

It will be noticed that while a large amount of copper is lost by the mill in the form of slime, the sand tailings originating from the rolls contain more copper than the corresponding material from the mill. This accounts for the fact that actually more copper could be saved from the mill product than from the roll product.

The object of the following test was to make the same comparison as had been made in the preceding test for 2½-mm screens, with finer screens on the machines.

Test No. 3.—Feed: Tailings from jigs working on a trommel product, passing through 5-mm and remaining on 2¼-mm trommels.

Screens: One-mm round-hole screens on Chilean mill; 14-mesh No. 19 wire (opening, 0.79 mm) on trommel screening roll discharge.

Average Load: 48.8 tons per machine per day.

Oversize from Trommel: 21.8 per cent of feed. (This is equivalent to the statement that 68.6 per cent of the roll discharge is constantly returned. The high percentage of oversize returned is partly due to the fact that owing to the heavy load of fresh feed and returning oversize the trommel was heavily overloaded and its screening action very imperfect, which is shown by a screen analysis of the oversize tabulated in table 3.)

The results of screening the mill discharge and the roll

TABLE II.—Rolls.

	Concentrates.			Middlings.			Tailings.			Total.			
	Weight.	Assay	Cu	Weight	Assay	Cu.	Weight.	Assay.	Cu	Weight	Assay.	Cu. Direct.	Cu. Calculated
0–20	0.14	8.02	1.1	0.23	1.75	0.4	9.85	0.59	5.8	10.22	.76	7.8	7.3
20–30	0.37	7.19	2.7	0.18	1.86	0.3	15.52	0.56	8.7	16.07	.73	12.7	11.7
30–100	1.59	7.81	12.4	0.71	3.86	2.7	21.92	0.39	8.5	24.22	1.03	25.0	23.6
100–200	0.86	11.04	9.5	0.80	0.40	0.3	5.64	0.23	1.3	7.30	1.50	10.9	11.1
200–∞	1.15	15.86	18.2	4.14	0.34	1.4	36.90	0.54	19.9	42.19	0.94	39.7	39.5
	4.11	10.69	43.9	6.06	0.84	5.1	89.83	0.49	44.2	100.00	0.96	96.1	93.2

Mills.

	Concentrates			Middlings.			Tailings.			Total.			
	Weight.	Assay	Cu.	Weight.	Assay.	Cu.	Weight.	Assay.	Cu.	Weight	Assay.	Cu. Direct.	Cu. Calculated.
0–20	0.07	3.42	0.2	0.08	1.36	1.1	6.35	0.41	2.6	6.50	.48	3.1	3.9
20–30	0.10	5.30	0.5	0.21	1.46	0.3	5.89	0.40	2.4	6.20	.48	3.0	3.2
30–100	1.54	6.37	9.8	0.68	1.03	0.7	18.19	0.28	5.1	20.41	.78	15.9	15.6
100–200	0.98	10.45	10.2	0.45	0.64	0.3	8.19	0.24	2.0	9.62	1.26	12.1	12.5
200–∞	1.73	16.48	28.5	3.44	0.42	1.4	52.10	0.57	29.7	57.27	1.06	60.7	59.5
	4.42	11.12	49.2	4.86	0.78	3.8	90.72	0.49	41.8	100.00	0.95	94.8	94.7

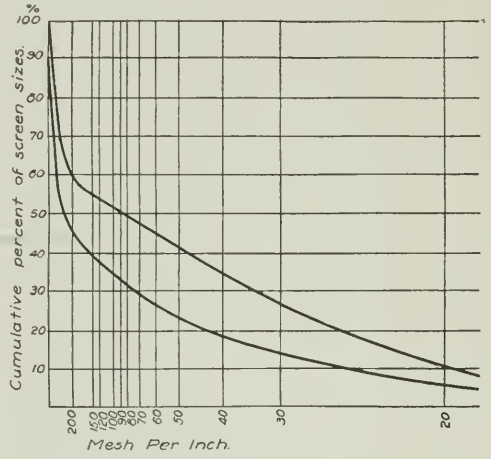


FIG. 2.—GRAPHICAL REPRESENTATION OF SCREENING RESULTS OF TABLE II.

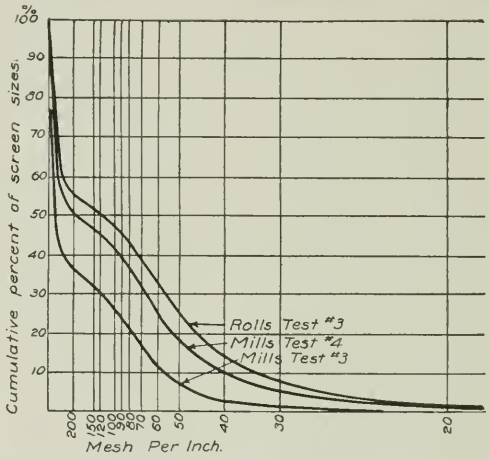


FIG. 3.—GRAPHICAL REPRESENTATION OF SCREENING RESULTS OF TABLES IV AND V.

screen undersize are tabulated in Table 4 and represented graphically in Fig. 3.

The result is practically a standoff between rolls and mills, the rolls losing more copper in the sands and the mills more copper in the slimes.

It appears, therefore, that the same saving of copper can be obtained by grinding in Chilean mills as by grinding in rolls, and that in a much simpler way mechanically. Reliable comparative operating costs are not available, but it seems reasonable to assume that on account of their simplicity the larger sizes of mills should be cheaper in operation than rolls, counting in the auxiliary screening and elevating machines. Elevating can, of course, be avoided by sending the oversize to other grinding machines.

TABLE III.

Trommel.	Oversize.
0–10	16.0
10–20	24.2
20–30	34.9
30–50	14.2
50–100	7.2
100–200	0.8
200–∞	2.7
	100.0

The conclusion drawn from the previous tests, that the same saving in copper can be obtained from ore ground in Chilean mills as from ore ground in high-speed rolls, holds true, however, only for the case that the grinding in these machines is final and is not followed by regrinding of any part of the pulp. Where the coarse part of the pulp is reground rolls are evidently preferable, as their product contains considerable amounts of copper mineral in the sands from which it can be set free by regrinding.

It was tried in the following test to decide if this point of superiority is really due to the characteristic grinding action of the rolls or more to the fact that the ore is not ground in one step to the desired fineness, but in stages, grinding alternating with screening. If the latter is the case, the same object should be attainable by Chilean mills using a coarse screen (or possibly no screen at all) on the mill itself, and a finer screen not connected with the mill, in addition, which would make the separation between oversize (to be returned) and undersize (to go to the concentrating machines).

Test No. 4.—Feed: Tailings from jigs working on a trommel product passing through 5 mm and remaining on 2½ mm.

Screens: 3 mm (slotted) on Chilean mill. The Chilean mill product passed to a trommel with 14-mesh No. 19 wire (opening, .79 mm), the oversize of which was returned to mill.

Average Load: 70.3 tons per machine per day.

Oversize from Trommel: 58.7 per cent of feed. (This is equivalent to the statement that 37 per cent of the mill discharge is constantly returned.)

The results of screening the mill discharge are tabulated in

TABLE IV.—ROLLS.

	Concentrates.			Middlings.			Tailings.			Total.			
	Weight.	Assay.	Cu.	Weight.	Assay.	Cu.	Weight.	Assay.	Cu.	Weight.	Assay.	Cu Direct.	Cu Calculated.
0–20	.....	.....	.....	.....	.....	.....	0.88	1.36	1.2	0.88	1.36	1.2	1.2
20–30	0.12	12.86	1.5	0.16	4.50	0.7	7.25	0.86	6.2	7.53	1.10	8.3	8.4
30–50	0.65	13.15	8.5	0.69	3.51	2.4	14.99	0.71	10.6	16.33	1.25	20.4	21.5
50–100	2.05	14.75	30.2	1.74	2.47	4.3	18.78	0.45	8.5	22.51	1.92	43.4	43.0
100–200	0.75	21.32	16.0	0.80	1.72	1.4	6.59	0.33	2.2	8.14	2.47	20.1	19.6
200–∞	1.85	19.66	36.4	.....		sand	19.20	0.44	8.4	44.55	1.39	62.0	62.7
						slime	23.50	0.76	17.9				
	5.42	17.08	92.6	3.39	2.60	8.8	91.19	0.60	55.0	100.00	1.55	155.4	156.4

MILLS.

	Concentrates.			Middlings.			Tailings.			Total			
	Weight.	Assay.	Cu.	Weight.	Assay.	Cu.	Weight.	Assay.	Cu.	Weight.	Assay.	Cu. Direct.	Cu. Calculated.
0–20	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....
20–30	.....	.....	.....	.....	.....	.....	1.21	0.76	0.9	1.21	0.76	0.9	0.9
30–50	0.23	6.55	1.5	0.22	2.85	0.6	5.34	0.44	2.3	5.79	0.80	4.6	4.4
50–100	1.45	12.33	17.9	1.56	2.48	3.9	16.88	0.36	6.1	19.89	1.39	27.6	27.9
100–200	1.01	17.62	17.8	1.64	0.84	1.4	7.19	0.32	2.3	9.84	2.19	21.5	21.5
200–∞	2.61	22.38	58.4	.....		sand	28.98	0.46	13.3	63.27	1.60	101.2	103.1
						slime	31.68	0.99	31.4				
	5.30	17.47	92.6	3.42	1.73	5.9	91.28	0.62	56.3	100.00	1.56	155.8	157.8

Table 5, and represented graphically in Fig. 3, together with the results of test No. 3.

The curve representing this mode of operating a mill in Table No. 7 (corresponding to the return of 40.6 per cent of mill discharge) approximates the curve obtained for the rolls when the product was screened on the same screen (with an oversize return of 68.5 per cent of the roll discharge). It seems to be safe to assume that for the same percentage of oversize the curve for the mill will reach the curve for the rolls.

It follows, therefore, that the rolls have not any inherent advantage over mills for grinding fine. It has to be considered, however, that to effect this reduction of sliming the mode of operating the mills becomes just as complicated as that of rolls, involving machinery for screening and elevating. (Elevating can be avoided by sending the oversize to other machines.) Under these conditions it is doubtful which class of machines can be operated at the lowest cost (the cost figures of this plant seem to point in favor of high-speed rolls), and the only serious objection to using them for this purpose would be the difficulty of keeping the faces true, which I understand has been found a serious drawback wherever fine grinding in rolls was attempted.

Regarding the point that the sliming of the mills can be reduced to the same amount as the sliming of the rolls, provided the mills make an oversize product of equal quantity, I want to add that this naturally only holds true for the conditions under which the test was made, that is, for reducing the maximum grains to about 1 mm. For coarser grinding the rolls, as hardly anybody doubts, are superior. In fact, some experiments along this line showed that the work of rolls operating in conjunction with a 2½-mm screen could, as far as minimum sliming is concerned, not be equaled by mills.

TABLE V.

Trommel.	Undersize
0–20	0.32
20–30	5.24
30–50	13.28
50–100	22.91
100–200	8.23
200–∞	50.02
	100.00

A fact which should be considered in judging the figures obtained in these tests is that in actual practice the results will be slightly different as the results of the concentration tests are based on the assumption that the concentration following the grinding is perfect. As a matter of fact the slime tailings from concentrators mostly show considerable amounts of free mineral on panning, while the coarser sizes are nearly free from such material.

This is an additional reason why the formation of slime should be avoided as far as possible by either replacing Chilean mills by high-speed rolls or by operating the Chilean mills in the manner suggested above, namely, grinding in stages. It has been reported that the Utah Copper Company is departing from the old method of running Chilean mills by using rather coarse screens on the mills and regrinding the coarser sizes of sands, which, judging from above tests, seems a commendable step.

The question whether high-speed rolls or Chilean mills with stage grinding are more economical can only be decided by tests on a practical scale extended over a long time. The experience of the Miami mill, where Mr. Burch is installing high-speed rolls of his own design in one section, to be compared with the Chilean mills of the other sections, may render such a decision possible.

Morenci, Ariz.

Duralumin

In a recent lecture of Mr. L. M. COHN before the Verein zur Beförderunz des Gewerbefleisses in Berlin, reported in *Electrotechn. Zeit.*, April 27, some interesting data on aluminium alloys are given, made by the Dürener Metallwerke in Düren i. Rh., Germany, and sold under the trade name "duralumin." These are alloys of aluminium with about 0.5 per cent magnesium, 3.5 to 5.5 copper, and 0.5 to 0.8 manganese. These additions are claimed to increase very materially the breaking strength, ductility, hardness and chemical resistivity of the metal.

The specific gravity of the different brands of duralumin varies between 2.75 and 2.84. The melting point is 650° C. The electric conductivity is "less than that of pure aluminium." The hardness of soft sheets, 7 mm thick, is 98 to 125, hence three times greater than that of pure aluminium and twice that of soft brass for cartridges. By rolling the hardness can be increased to 170. "The modulus of elasticity for alloy 681 A is 700,000 kg per square centimeter, the shearing strength 30 kg per square millimeter, the breaking strength about 50 kg per square millimeter, the elongation for a sheet, 2 mm, which is about 4 per cent." Since above 150° C. the strength rapidly decreases, duralumin should not be used above this temperature.

Special advantages of duralumin are its light weight and resistivity against chemical influences. But it should not be used in contact with copper or brass, but rivets of iron, or preferably duralumin, should be employed. Various kinds of vessels and fixtures are now made from this new alloy. For the airship just finished for the British army duralumin has been used throughout as material of construction.

Recent Progress in Calorimetry.

Details of Apparatus and Method.

By Walter P. White.

Last April, in an article in these columns describing recent progress in calorimetry, it was shown that most errors in the past have been due to uncertainty in the conditions attending a determination; that these uncertainties can be avoided by preventing evaporation, adjusting and regulating the speed of stirring and removing the calorimeter from the influence of all bodies of unknown or uncertain temperature; that the various lag effects can easily be made negligible; and that the preponderant *unavoidable* error is then in the temperature measurement.

The advantage which may be sought from the new methods will differ in different cases. For instance, one worker will wish chiefly to diminish his accidental errors—that is, to make his own results agree better among themselves—another may find it more important to reduce the systematic errors which produce discrepancies between the results of different observers; still others, satisfied with the accuracy they are now getting, may yet welcome an increase in the speed or convenience of their methods. Of these three advantages, the avoidance of accidental irregularities has presented the greatest difficulties and is most important. It will be further discussed presently. The securing of concordant results among different observers is often at present a great desideratum in commercial work. Its lack is sometimes acutely felt even where the accidental errors are satisfactorily small. It is, however, an easier problem than the other. It will be treated first.

Systematic variations between different observers may be due to four causes. (1) Error in some auxiliary process or observation (for instance, with the bomb calorimeter, incomplete combustion, impure oxygen, inaccurate weighing, etc.). (2) Error in the value taken for the heat capacity ("water equivalent") of the calorimeter. (3) Error in the thermometer calibration. (4) Various errors due to lag which, as already pointed out, produce the effect of an error in the value of the heat capacity. The simplest and by far the surest way to eliminate all these errors, it is generally recognized, is to make a direct calibration of the calorimeter under actual working conditions, using some standard substance of known thermal properties. A familiar instance is the calibration of bomb calorimeters by determining with them the heats of combustion of sugar or benzoic acid, whose true values are pretty well known. Results obtained with these calorimeters thus become definite, and comparable with those of others, and their absolute accuracy will increase with every increase in our knowledge of the standard values.

With this method of procedure, of course, it is essential that there be no change either during the calibration or afterward in the errors whose effect it is sought to eliminate. Changes in auxiliary processes, though important, do not fall within the plan of the present paper; changes in the water equivalent, though also important, do not seem to call for discussion here, and the same may be said of errors in the thermometer calibration. But errors due to lag are not so well understood, and should perhaps be discussed briefly.

The lags most likely to produce actual error are those due to bodies of uncertain temperature, such as covers not touching the water of either calorimeter or jacket, shields and other large masses within the jacket, or the jacket itself if it consists merely of shields and contains no water. These lags will be constant, and will, therefore, not be productive of appreciable error in a calibrated calorimeter as long as the conditions affecting them do not change too greatly. But the one who, therefore, counts on ignoring them, of course, does so at his own risk, and must, to avoid error, not only be able to keep the essential conditions always constant, but must also be sure that he fully understands what *all* the essential conditions are.

It should not be supposed, however, that the errors, if any occur, would be large. These lags only alter the cooling correction by a fraction of itself; the cooling correction is only about 2 per cent of the total heat quantity, and no change of conditions is likely to alter the lags consistently by more than a rather small fraction of themselves. Hence, while these lags might cause considerable discrepancies between the results of calorimeters whose water equivalents were merely computed, they are not likely to cause in calorimeters directly calibrated any systematic errors greater than 1 part or 2 parts per mille. Present data do not seem sufficient for a more definite statement, since most of those who have given any particular attention to these lags have preferred to eliminate them rather than to bother with them.

The preceding suggests one general consideration. In many cases the presence of large lag effects is doubtless due to a legitimate attempt at simplicity of apparatus, but it also often results from an effort to increase accuracy by diminishing the magnitude of the cooling correction. This, of course, results from the mistaken notion that the cooling is the main source of error. But appreciable error rarely or never arises from the main part of the cooling correction; it comes from the smaller but variable subsidiary effects; hence a very large decrease in the total value of the cooling constant is a positive disadvantage *provided*, that it is accompanied by even a slight increase in the variations and conversely. The failure to appreciate this fact has frequently led to an unfortunate design or procedure. For instance, calorimeters have been considerably used in which the jacket consists of two fiber pails. If the space between these pails were to be filled with water, slightly stirred, the cooling correction would increase, but the errors would decrease because the environing temperature would become more constant. Similarly, where metallic shields are used inside a water jacket. Again, the substitution of Dewar flasks for metallic calorimeters has seemed to many persons an enormous improvement on account of the accompanying decrease in the cooling rate. (It is perhaps not everywhere known that use as a calorimeter was the original object of the Dewar flask, which antedates liquid air.) Of course, such a decrease is, other things being equal, a desirable thing, though less important than many appear to have supposed; the rate is, to judge from two recently reported cases, about one-fifth the usual one, but this is a very long way from giving anything like five-fold accuracy. On the other hand, in the Dewar flask, where there is a continuous glass connection between the calorimeter proper (inside) and the jacket (outside), the lag effects and the water equivalent are hard to determine and vary as the water level changes. Hence, while these flasks, if used with proper precautions, often have great advantages, they are not likely to prove particularly valuable in cases where it is supposed that their small cooling factor is alone enough to secure accuracy without special precautions.

Another case where the advantages of reducing the cooling correction have been overestimated is the case of adiabatic methods. In these methods the jacket is heated so as always to be at the same temperature as the calorimeter; there is thus (theoretically) no direct heat flow at all from the calorimeter (though a correction for the effects of stirring and evaporation is, in general, still necessary). This total absence of direct cooling is generally supposed to increase accuracy. Of course, it does nothing of the sort (in determinations of ordinary length). In fact, Professor Richards, who has been a leading exponent of the method, did not expect to gain accuracy from this source as much as from the elimination of certain lag effects. But it is now known that the errors from these effects are negligible or non-existent, though this was not known at the time when the method was introduced. There is now no good reason to suppose that adiabatic methods have any advantage in accuracy in determinations of ordinary length. They may often have decided advantages in convenience.

The value of a small cooling correction has also loomed large to those experimenters who have suggested that the calorim-

eter should be of such a shape as to have the minimum surface. As a matter of fact, differences of shape, unless extravagant, have but slight influence on the cooling rate, and hence have an altogether insignificant effect on the accuracy. The shape of the calorimeter should be chosen with reference to facility in stirring, and, of course, to convenience in construction or manipulation.

Passing now to consider the *accidental errors*, which set the present limits to calorimetric accuracy, the principal conditions for good work are: Constancy of conditions and sufficient thermometric sensitiveness. The means of securing these are in detail much as follows:

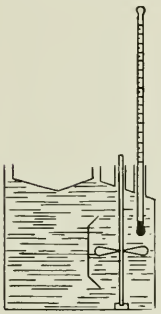


FIG. 1.—THIN SECTION OF CALORIMETER WITH FLOATING COVER IN CONTACT WITH WATER.

The Calorimeter.—To avoid lag errors projections should be avoided as far as possible, especially those of poor heat-conducting material. For the same reason, if there is any cover, it should be in direct contact with the water. Unless the calorimeter is to be always operated at a temperature lower than the jacket, the control of evaporation will require also a rather tight cover. A cover for the jacket alone will not answer at all. The simplest cover to manipulate is a floating cup, about 1 mm narrower than the calorimeter opening. The evaporation with such a cover will seldom reach 1 mg per minute, which means 0.6 calory heat loss a minute. If the cooling correction is (as usual) of the type which eliminates the effect of a *constant* evaporation, and if there are 2 or 3 liters of air space inside the jacket, the error due to evaporation then will seldom be as much as 0.2 mille for an ordintry determination, involving 6000 calories and lasting not over 10 minutes. Or, the edge of the cover can be made to rest in a narrow groove, and the joint sealed with 300 mg or so of oil, when evaporation is, of course, practically nil. Other fastenings are also possible, since absolute tightness is not needed. With such, since the water must touch the cover, the amount of water must be properly regulated. This is easily and quickly done by sucking the water out, down to a certain level. A tube shaped like the letter "J" is probably best for this purpose, and gives levels constant to one-tenth or two-tenths of a millimeter. Such a tube, stuck through a large cork which rests on the calorimeter rim, when in use, makes as good an adjuster as any.

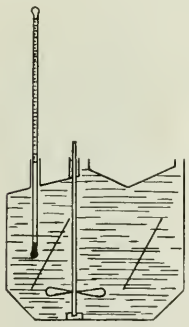


FIG. 2.—THIN SECTION OF CALORIMETER WITH FLOATING COVER. THE INCLINED TUBE GIVES CENTRAL STIRRING WITHOUT ENCUMBERING THE OPENING.

The stirrer shaft and thermometer can run through the movable cover, but are more conveniently inserted through a fixed cover, or half deck, as shown in Fig. 1. The stirrer, unless there is strong reason to the contrary, should be a propeller, because that is more efficient and causes less evaporation than an up-and-down stirrer. The effectiveness of propellers is usually promoted by inclosing them in side tubes, as in the figure. Such tubes should not be too small. Suppose, for instance, two calorimeters, in one of which the side tube is two-thirds as wide as the main calorimeter, while in the other it is one-third as wide. For any given effectiveness, then, the speed of the water in the narrow side tube is four times as great as in the other, and, since the heat is about proportional to the cube of the speed of flow, the heating is possibly 64 times as great. But friction is greater for the same speed in a narrow tube, so that the heating may be 100 times as great or more and is certainly far greater than in the wider tube. The disadvantage of throttling the stream of water is thus evident. Throttling is at a minimum if the stirrer is in a central tube. An arrangement by which this construction can be secured without having the stirrer shaft interfere with the calorimeter opening is shown in Fig. 2. This arrangement has proved very successful in reducing the heat of stirring.

The jacket, for the best results, should contain water, whose temperature should be kept uniform by stirring, and should be measured. The water should surround the calorimeter completely, so that the whole surrounding temperature is definite. The water, therefore, should be *above* the calorimeter, as well as below and at the sides. Several satisfactory methods of accomplishing this have been used. One of the best is as follows: The calorimeter chamber is a vessel open at the top and sunk in a larger vessel; the space between the two, partly open at the top, contains the water; the cover is a flat box, air-tight except for two large tubular openings which project downward and reach below the surface of the water in the lower vessel. The cover is filled with water, which is made to stay there by atmospheric pressure; the stirrer is directly under one of the tubular openings, and so produces circulation in the cover as well as in the lower vessel. (Fig. 3.) The opening of the chamber may be provided for in either of two ways: (1) The cover, tubes and all, is moved directly aside; this method agrees best with a design having a square outer ves-

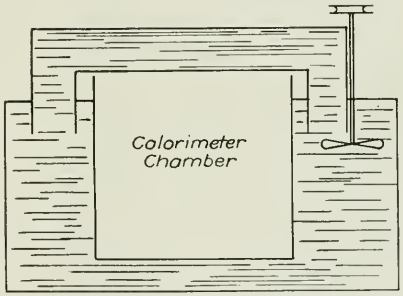


FIG. 3.—THIN SECTION OF CALORIMETER JACKET WHICH SURROUNDS THE CALORIMETER COMPLETELY WITH A SINGLE STIRRED BODY OF WATER AND IS READILY OPENED AT ANY TIME.

sel and two half covers, one of which transmits the thermometer, stirrer, etc.; this half can be kept stationary while the other is moved to expose the calorimeter opening. Both are moved apart to remove or insert the calorimeter. (2) In the other method the two tubular openings are near each other and the single cover swings around one of them as an axis in exposing the chamber and calorimeter; the thermometer passes through either a hole in the cover or a notch in its edge; partitions guide the circulation.

The upper bearing for the stirrer should be on the jacket, or, at any rate, not on the calorimeter itself. If the calorimeter is always below the temperature of the jacket the jacket may almost as well be left open at the top, since there is comparatively little tendency for heat to pass downward across an air space at ordinary temperatures. Of course, such a jacket is not to be unreservedly condemned, even where the calorimeter is sometimes warmer than it, if the highest accuracy is not required. A good deal will depend on how constant the room temperature is. There should be no difficulty in determining by actual observation how great is the direct influence of room temperature upon the calorimeter. A few hours' observation of the cooling rates, with different relations between jacket and room temperature, is all that is required. The error likely to result

through the use of an open jacket can then be readily estimated for any given variation in room temperature.

The necessity of measuring the jacket temperature can also be avoided if certain cooling correction methods (e.g., the Pfaundler) are used, and if that temperature is also kept practically constant. It can easily be shown that variations in the jacket temperature arising from the temperature change in the calorimeter will be less than 0.1 per mille if the jacket has ten times the heat capacity of the calorimeter. Changes in the jacket arising from the room temperature are more serious. If the difference between room and jacket is equal to the calorimetric interval the resulting error will often reach, and may exceed, 1 per mille. Here, also, the probable magnitude of the error for a given difference of room and jacket can readily be found experimentally—in this case by a little observation of the cooling rate of the jacket.

Summary.—Systematic errors in calorimetry can usually be eliminated by a direct calibration, except for certain lag errors, which can be avoided by proper construction and procedure. Accidental errors can be greatly reduced by various features, described above, in the design of calorimeter and jacket. Temperature measurement and the calculation of temperature corrections will be treated in a third paper.

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Positive Rays and Chemical Analysis

The discourse on "A New Method of Chemical Analysis" which Sir J. J. Thomson delivered at the Royal Institution in London on April 7 showed that a line of research, of which Professor Thomson gave a preliminary account before the Sheffield meeting of the British Association, has led to most remarkable results. The following account of the lecture is taken from London *Engineering* of April 14:

There is nothing chemical in the new method of analysis. But the study of the positive rays (Kanalstrahlen) tells the experimenter not only what gases he is actually dealing with in the apparent residual gas of his vacuum tubes, but in what state—atomic, molecular, or molecular aggregations—they are, all deduced from measurements of deflection of the rays, which measurements give a clue as to the atomic weight (or equivalent). This is the strength, and possibly also the weakness, of the new method, which bears some resemblance to spectroscopy.

The new method identifies the elements by their masses. It informs the experimenter that some substance having a certain e/m (ratio of electric charge to mass) is present, and as the experimenter knows with some degree of certainty what substances he starts with, the method affords a valuable help to the chemist, especially as it is wonderfully sensitive and records the presence and mass of elements or radicles which are far too evanescent to be isolated.

Sir Joseph said, in introducing his subject, that he approached a chemical problem with some trepidation, since chemists were fighting men. But his weapons were bullets traveling at 1000 miles per second. The positive rays were positively electrified particles attracted to the negative electrode. When that electrode was perforated (by Goldstein's Kanal) the particles would pass through the cathode in streams which, in neon, had a beautiful orange-red color, as was shown. The particles were deflected both by the magnetic and the electric field, and the research depended upon the measurement of these deflections.

There are several arrangements of the bulbs possible, and these bulbs are sometimes very large. In one of the new arrangements (Fig. 1) the tubular cathode is continued backward in a fine tube, terminating at two electrodes, between which an electric field is established. A magnetic field is also provided so that after passing through the two fields the particles strike the flat screen of willemite, closing the tube.

With the magnetic field alone in action, the original light spot O (Fig. 2) of the undeflected particles is shifted to N; the deflection ON being equal to Ae/mv, where A was a constant depending on the intensity of the field and the size of the tube, while m was the mass of the particle, e its charge, and v its velocity. The electric field similarly deflected the particle to OM, at right angles to ON, and $OM = Be/mv^2$, where B was another constant.

When both fields were excited the particles would be shifted to P, and it could easily be shown that the ratio of m to e should vary as the ratio of OM to ON^2. If, now, all the particles were of the same kind and had the same speed, the position of P would merely depend on m/e, and a number of spots P—as many as there were different kinds of particles—would be seen. But the particles of the same kind did not move all at the same speeds, and each spot was drawn out into a curve, which was a parabola if m were constant.

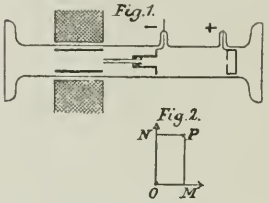


FIG. 1 AND 2.—ARRANGEMENT OF DISCHARGE TUBE.

The lecturer and his assistants had at first made the measurements directly on the screen while the experiment lasted. But that was very laborious, and they had substituted a photographic plate for the screen so that each particle photographed itself. They had placed the photographic plate first outside and then inside the bulb (as has also been done in similar researches conducted by Koenigsberger and others), and with the aid of a slide shutter, due to Sir Joseph's assistant, Mr. F. W. Aston, the exposure was made at the proper moment and several photographs were taken on the same plate.

In each case the first photograph was taken with the magnetic field producing a deflection upward, the second with the reversed field giving a deflection downward. Thus they obtained full parabolas (not only one branch), and they could ascertain the zero from which to measure with more definiteness. In the slide used the photographic plate was suspended by a silk thread wound round a plug worked in a glass-ground joint.

FIG. 3.—PHOTOGRAPHIC PICTURE OBTAINED.

Fig. 3 illustrates the beautiful curves obtained with a tube containing nitrogen isolated from the atmosphere. The central light spot indicates the original position of the undeflected ray. There were many faint and bright curves, sometimes beady; their number suggested the presence of many substances, constituents or impurities; the argon curve was present, for instance, while absent on another photograph taken with nitrogen prepared from ammonium nitrate. Most of the curves were on the right side of the photograph, in which direction the positive particles were deflected. But there were also some faint curves on the left-hand side, showing that some negatively-charged particles had found their way through the cathode.

The uppermost curve always corresponded to the hydrogen atom, and this was present on all plates from discharges through all substances. Traces of oxygen, carbon and mercury were also found on almost all the plates, no matter what substance they were experimenting with. The other substances followed in the order of their atomic or equivalent weights.

Thus, taking the ratio of m to e as unity in the case of

hydrogen, it is possible to calculate from measurement of the curves the relative value of this ratio for the other particles impressing themselves on the plate. We reproduce below figures deduced from the measurements of the parabolas of Fig. 3, showing the discharge through atmospheric nitrogen; the enumeration starts from the top of the diagram.

Positive Side.	Negative Side.
1.00 H +	1.0 H —
1.99 H_2 ++	11.2 C —
6.80 N ++	15.2 O —
11.40 C ++	
13.95 N +	
28.1 N_2 +	
39.0 A +	
101.0 Hg ++	
198.0 Hg +	

The + and — indicated, the lecturer explained, the simple or double charge carried by the particle. The H = 1 corresponded to the hydrogen atom carrying a single charge; the second, $H_2 = 1.99$ (instead of 2) to the hydrogen molecule; 11.40 C + to a carbon atom (C = 12); the 6.80 N ++ to a nitrogen atom carrying two charges (N = 14); the 13.95 N + to the nitrogen atom; the 28.1 N_2 + to the nitrogen molecule; the 39.0 A + to the argon atom (A = 40); and the two last curves, 101 Hg ++ and 198 Hg + to mercury (Hg = 200), carrying one and two charges respectively. The agreement between the deduced values and those assumed by chemists (figures added in brackets) was very close. The hydrogen, carbon, and oxygen found on the negative side are all in the atomic state.

Comparing the new method with spectroscopy, Sir Joseph pointed out that the spectroscopist, observing a new line, was not always certain whether he was not looking at a line of an old known substance produced under special thermal or electrical conditions. In his new method the particles wrote their names on the plate. The method was also exceedingly sensitive. He had traced the presence of helium in this way, when it could not otherwise be detected, requiring only a few particles of the substance confined in a vessel at a pressure of 0.03 mm. of mercury. The accuracy of the measurement was not dependent upon the weight of the substance present.

Although these researches had only been going on for a few months, he realized an accuracy of 1 per cent with 0.01 milligram of substance. No chemist would undertake a determination of atomic or molecular weights under similar circumstances. Another great advantage was that the purity of this substance—the great difficulty of the chemist—was of no consequence. The method sifted the particles, merely giving an additional curve for each constituent. It is further noteworthy that the atom or particle recorded itself within a millionth of a second of its production; thus substances were detected of which the free existence had not, so far, been proved by other means.

The lecturer then exhibited slides of the curves obtained with other gases—chemically-prepared nitrogen, carbon monoxide, carbon dioxide, methane, chloroform, carbon oxychloride, etc., and he reproduced also the tables of the measurements. A few characteristic features may be pointed out. The chemically-prepared nitrogen did not give the argon line, as it should not. A faint mercury curve 65.5 Hg +++ was frequently observed, but somewhat uncertain. The carbon compounds showed 6.0 C ++ in addition to the 12.02 C +.

In the experiments with methane, CH_4 (marsh gas), a veil in the neighborhood of the carbon line had finally been resolved into four curves, 11.95 C +, 12.9 CH +, 14.05 CH_2 +, 14.9 CH_3 +; these were familiar groups to the chemist, but had never been obtained in the free state.

They hoped to study the transformation products of radioactivity substances in the novel way. In the case of chloroform, $CHCl_3$ and some other compounds, two obscure faint lines were observed, 1.5 and 3, the 1.5 also on the negative side; the 3 might possibly be C ++++. The lecturer did not comment on the fact that none of his photographs exhibited any helium curves.

Passing to general conclusions, Sir J. J. Thomson remarked that, leaving accidental impurities out of consideration, this analysis always demonstrated the presence of the respective substance in several states or systems, atomic, molecular, or complex aggregations, carrying one or several positive and sometimes also negatives charges.

Oxygen, for instance, was found in nine modifications, comprising ozone, O_3 × (the suggestion is not new to chemists), O —, and uncharged atoms and molecules. In view of this variety, it did not appear surprising that substances gave many kinds of spectra under different conditions; the various systems might give rise to different spectra.

On the negative side were found the atoms of hydrogen, carbon, oxygen, and chlorine, but never in the molecular condition; the last two would naturally be expected on the negative side, and the association of hydrogen (though a highly electropositive element) with negative charge was no longer a novelty.

As regards these curves on the negative side, it would appear that negative particles were carried through the cathode and that they were picked up by neutral particles and held by attraction. Only the atoms, not the molecules, seemed to be able to do this, and the phenomena illustrated the different behavior of atoms and molecules. Apparently the atom, not the molecule, was capable of exerting momentarily a very powerful attraction. The explanation was probably that the atom had more freedom of motion than the molecule. When magnetism acted on a sphere of a magnetic substance, the electricity within that sphere was free to move and to readjust itself in accordance with the magnetic induction, and a strong attraction resulted. On a sphere of sulphur the freedom was limited and the attraction was weak.

In illustration of this view, about half a dozen compass needles had been mounted, each on its vertical pivot, on a piece of cardboard, which was held suspended by strings in front of an electromagnet. When the electromagnet was excited, the whole system on the cardboard was attracted toward the electromagnet; but when the needles were taken off their pivots and placed on the board, on which they could no longer move freely, hardly any attraction was observed. Similarly, Sir Joseph remarked, there were stiff and freely-moving atomic and molecular systems. It depended upon the numbers of corpuscles in the atom; one corpuscle more than a critical number would (as he had explained on other occasions) impart to the atomic system one freedom of motion, one bond or one valency; two free corpuscles would correspond to two bonds and two valencies. When the atoms joined one another to form molecules, the bonds were linked, the system became rigid, and it failed to attract bodies outside. These researches, Professor Thomson concluded, might help forward the solution of the problems of chemical combination.

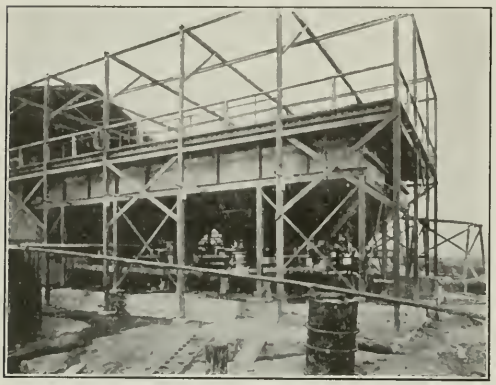
Chemical Analysis in British National Physical Laboratory.—Last year's report of the (British) National Physical Laboratory contains an account of some researches on chemical analysis (London *Eng'ing*, April 21). The small boats of fire-clay wanted for combustion tests were formerly molded and then pressed. The clay is now forced through a die and delivered as a U-shaped strip, which is pinched off at suitable lengths—a more rapid and neater process. The new electric oven for carrying on spontaneous-combustion experiments is a cubical box with asbestos walls, containing a copper box, 18-in. cube, in which the material is placed; a temperature of 500° C. can be kept constant within 1°. There is a free air space of 3 in. around the box. When materials like cotton wool or charcoal are examined a rise of the thermo-couples inside the material above those outside would indicate chemical changes which might lead to combustion.

The Butters Filter Plant at the Crown Mines, Ltd.

The February issue of the *Journal* of the Chemical, Metallurgical & Mining Society of South Africa contains an interesting account of the visit of the members of the society to the Crown Mines, Ltd.

The present metallurgical equipment is 675 stamps with 20 tube mills, and sand and slime plants, made up as follows:

A.	Crown Deep	300	stamps and	10	tube	mills.
B.	Langlaate Deep....	200	" "	6	"	"
C.	Crown Reef	120	" "	3	"	"
D.	Bonanza	55	" "	1	"	"



VIEW OF BUTTERS FILTER PLANT, FROM BENEATH.

With the present plant they have a capacity of 165,000 tons per month.

A fifth mill is being erected of more modern type, which, when completed, will increase the total crushing capacity to well over 200,000 tons per month. Special features are the tandem classifications and the tube milling at A plant; further, the installation of a 300-leaf Butters vacuum filter slime plant at C plant, which is unique not only on account of its being the first vacuum filter plant erected on the South African gold fields, but also because it occupies the site of the first slime plant designed and erected on the Rand by Mr. J. R. Williams.

The following is a description, by Mr. C. G. Patterson, of the Butters vacuum filter plant as installed and worked on the Crown Reef section of the Crown Mines:

The plant consists of two stock vats equipped (temporarily) with air agitation to keep the slime in suspension. After the values have been dissolved in Brown agitators the pulp at a dilution of 2 to 1 is transferred to the stock-pulp vats, from where it is fed to the filters as required.

There is one wash solution vat. All precipitated solution is delivered to this vat to be used subsequently for a wash in the filters. There are two Butters filter boxes, each fitted with 150 Butters vacuum filter leaves and one 14 x 14 Goulds vacuum pump. Each filter box is an independent filtering unit. One 10-in. Robeson-Davidson slime pump handles pulp and solution to and from the filter boxes and the stock vats, serving the boxes alternately.

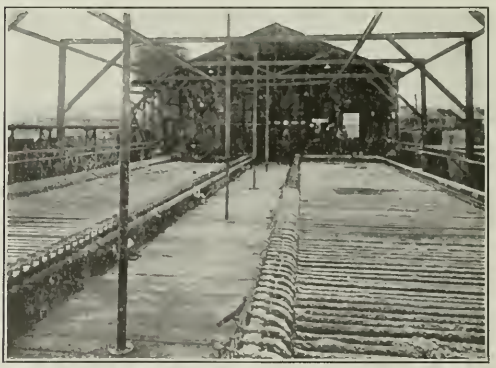
The piping is arranged round the pump in a loop, having four valves interposed so as to reverse the direction of flow through the pipes. The pump is set in either the emptying or filling position instantly, by means of one pilot valve introducing hydraulic pressure into the cylinders of the four valves simultaneously.

The motor for the 10-in. pump, and also valves in the pump systems, are operated from the switchboard. The style of plant is called the "pumping system" and the process of filtration is performed as follows:

Forming the Cake.—The filter box is filled with pulp to a point over the top of the filter leaves and a valve opened connecting the vacuum pump directly with the filters. Clear cyanide solution is drawn from within the filter leaves and delivered to the clarifying vat for precipitation, the slime remaining as a cake on the outside of the filters. The filters are kept submerged by refilling the box at intervals, and jets of air are introduced at the points of each hopper to keep the slime in suspension. When a cake 1 in. to 1½ in. thick has been formed, the surplus pulp is pumped back to the stock pulp vat and the box is then filled with solution to wash the cake. During the time of forming and washing the cake the vacuum is maintained at the highest possible point, but when the cake is exposed to air during the transfer of pulp and wash solution, the vacuum is reduced to 5 in. to prevent the cake cracking.

Washing.—The cake being formed by atmospheric pressure, the resistance and permeability are equal over the whole surface of the leaf, thus producing an ideal condition for the recovery of the valuable moisture contained therein, which equals 30 per cent to 35 per cent. Sufficient wash solution (about 2 tons per ton of slime) is drawn through the cake by the vacuum pump to effect a complete displacement of the original moisture. A portion of this solution goes to the clarifying vats for precipitation; the flow is then diverted to a stock vat to be used in making up new charges in the Brown treatment vats.

Discharging.—When the wash is complete the vacuum is disconnected and a reverse flow of solution (by gravity from the vat on the roof) is introduced to the interior of the filters, causing the cakes to drop. The surplus solution is then pumped from the filter box through decanters and returned to the wash solution vat. The mass of thick sludge remaining in the box is diluted with water and agitated with air for a few minutes to make a homogeneous pulp of 1 to 1, which is then pumped to the residue dam by a second 10-in. R. & D. pump, the delivery pipe of which is fitted with an automatic sampler.



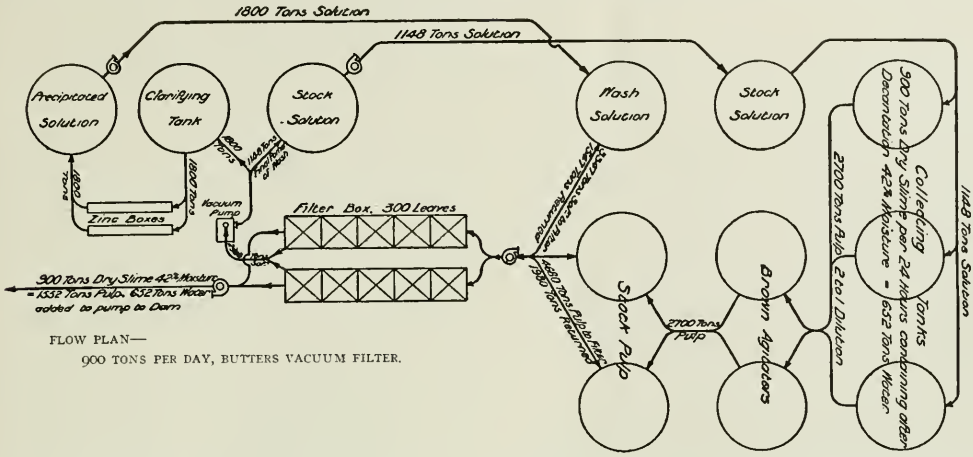
VIEW OF BUTTERS FILTER PLANT, ON OPERATING PLATFORM.

Cycle of Operations.—

Filling box with pulp and forming cake, 45 minutes.
Transferring and washing, 70 minutes.
Discharging, 20 minutes.
Total time of cycle, 2 hours 15 minutes.
Tons treated per cycle, 1¼-in. cake = 50 tons each box.

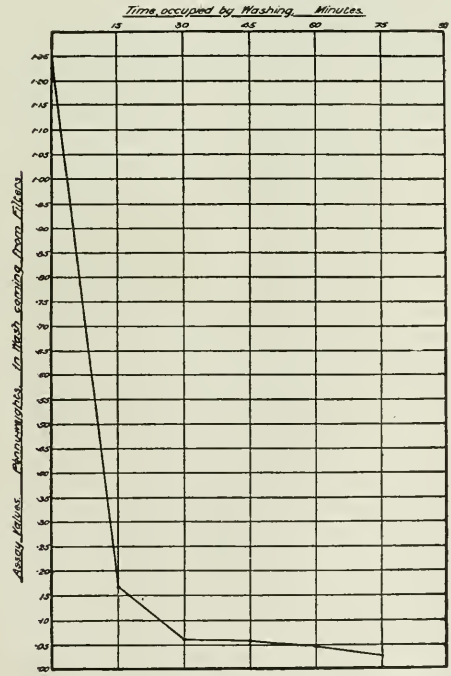
Acid Treatment.—Five leaves are removed from each filter box every day, thoroughly washed with a spray of water and air, and then immersed in a vat of 2 per cent hydrochloric acid solution for the purpose of dissolving the calcium carbonate and keeping the leaves soft, pliable and in good working condi-

tion. The HCl solution is passed through the leaves in reverse directions alternately by means of an automatic arrangement that requires no power or attention. The leaves are removed from the acid bath after four to six hours' treatment, again washed with the spray and returned to the filter box.


1800 Tons Solution
1148 Tons Solution
Precipitated Solution
Clarifying Tank
Stock Solution
Wash Solution
Stock Solution
900 Tons Dry Slime per 24 Hours containing 42% Moisture = 1552 Tons Water
Collecting Tanks
2700 Tons Pulp 2 to 1 Dilution
1148 Tons Solution
Zinc Boxes
Vacuum Pump
Filter Box, 300 Leaves
900 Tons Dry Slime 42% Moisture = 1552 Tons Pulp. 652 Tons Water added to pump to Dam
Stock Pulp
Brown Agitators
2700 Tons Pulp

FLOW PLAN—
900 TONS PER DAY, BUTTERS VACUUM FILTER.

Space will not permit of a complete analysis of the six months' test work, as conditions were not normal, owing to insufficient slimes to work the plant to full capacity and to the



WASHING CURVE CHARGE NO. 24, 18, 8, 10.

treatment of deleterious accumulated slime in combination with the current slime the major portion of the time.

In January a special test was made on current slime only, treated first by the Adair-Usher process, the residues from which were delivered to the filter for the purpose of determining what further recovery of dissolved values was possible by filtration. In this test, as also in the former operations with accumulated slime, the efficiency of the filter was fully demonstrated, assays of the residues showing consistently during the whole period over 99 per cent recovery of dissolved values.

There is some dissolving of gold, incidentally effected, during the process of filtration; however, the Butters filter is essentially a slime-washing machine, purely mechanical in its functions, and might be termed a process of forced decantation.

The plant was designed to treat 500 tons per day. Subsequent operations proved that, owing to the exceptional adaptability of Rand slime to filtration, it has a capacity of 900 tons per day.

The plant was put in commission on Aug. 12, 1910, and after a most exacting test over a period of six months has been formally taken over by the Crown Mines, Ltd., as a part of their permanent plant.

Electrolytic Iron.

Last year's report of the British National Physical Laboratory gives an account of an investigation made with electrolytic iron.

According to London *Engineering* of April 21 the experiments have so far been made on electrolytic iron obtained under a German patent, from a mixture of iron and calcium chlorides, by electrolysis at 110° C. with a current of 23 amp per square centimeter; a high current density is essential for the process.

The report does not give analysis of this iron, whose purity is perhaps doubtful. Amberg found nitrogen in iron deposited from chloride among other impurities.

The cathode is a hexagonal prism which yields six slabs of iron 3 in. x 1 in., ⅛ in. thick; these slabs are rolled in the cold and annealed in vacuo to get rid of hydrogen. The thin strips can be bent between the fingers and long crystals 1 in. long, of iron, as thick as the strip, may then split off; these crystals of pure iron are tough and ductile, while the metal is brittle along their boundaries.

Humfrey has heated long strips to high temperatures (say, 1000° C) at one end and to 500° C. at the other, the strip being enclosed in an evacuated porcelain tube; the strip is afterwards etched by admitting gaseous hydrochloric acid or bromide, and it then shows changes, corresponding to a wide range of temperature. The method is an amplification of one introduced by Baykoff.

The Methods of the United States Steel Corporation for the Technical Sampling and Analysis of Gases.

Copyright, 1911, by J. M. Camp, Chairman Chemists' Committee.

This paper is descriptive of the methods selected by the chemists of the U. S. Steel Corporation, acting through the Chemists' Committee, for the technical sampling and analysis of gases. With the increased significance attached to the economic control of those industrial processes involving the combustion of fuels, and with the application of keen scientific research to the various problems involved, it becomes readily apparent that accurate knowledge of the composition of the gases encountered is a matter of prime importance. This fact is exemplified by the rapid increase in the use of internal combustion engines, using blast furnace or producer gas as a fuel, and by the increasing necessity of greater watchfulness upon the efficiency of boilers, stoves and furnaces. The need is obvious for the use by the laboratories of the United States Steel Corporation of a standard system of sampling and analysis of gases, in order that comparable results may be obtained with the maximum degree of accuracy, consistent with the minimum of time requisite for execution.

This description was consummated through a review of the practices specially written for this purpose, of all the laboratories of the Corporation engaged in the sampling and analysis of gases. The apparatus shown for the analysis is to be considered only as a type of the permissible form. The essential points are the exclusive use of capillary tubes, the permanency of connection of the auxiliary gases, compactness, and the applicability of the apparatus to the accompanying description. This form of apparatus and the methods as here described, are to take precendence over all others employed for the analysis of gases throughout the laboratories of the Corporation.

It is the desire of the Chemists' Committee here to acknowledge their grateful appreciation of the services of Mr. D. A. Barkley and Mr. R. J. Wysor, assistants in the Duquesne laboratory, in the preparation of this paper.

In the preparation of the accompanying standard methods for the sampling and analysis of gases, there has been an unwavering purpose to eliminate, so far as possible, tedious analytical procedures, or the use of cumbersome and antiquated forms of apparatus. It has been desired to adopt methods, inherently correct in principle, in conjunction with simplified apparatus, which will insure the requisite expediency, at times so necessary in commercial work, without an appreciable sacrifice in accuracy of results. While no notable originality is claimed in general for the substance of this paper, it is believed that the application of well-known methods, coupled with certain novel features in execution, and in the forms of apparatus used, will be presented. A sense of vagueness or lack of definiteness with reference to practical descriptions, it is believed, pervades the bulk of literature concerned with this subject. No apology is offered for the unabridged nature of the descriptions and explanations appearing in this pamphlet. The delineation of the methods has been made in considerable detail so as to be clearly intelligible to amateurs as well as to those better versed in the subject. The goal simultaneously attained by accuracy and speed of execution, and so much desired in chemical methods, has been striven for, with what success the future will demonstrate.

Sampling.

The impossibility is recognized within the compass of this pamphlet of detailing a method for sampling gases, that would be applicable to all the works of the Corporation and the various metallurgical operations therein. It is preferred, rather, to make this description general in its scope, leaving to each individual operator the solution of the details to meet his own particular conditions. In the sampling of gas two kinds of samples are recognized, the regular and the irregular. The former comprises all samples taken over a continuous division of this time. The results so obtained will constitute the regular or official analysis and they only will enter into comparison with like results from other works. Under the head of the irregular samples will be included all those samples taken for a shorter period of time than the minimum above given, and will represent momentary conditions of the particular operation in question, and are for the works' guidance only. All analytical reports shall show the duration of the time of sampling in order that the status of the results may be seen at a glance.

In sampling blast furnace or producer gas, which are considered homogenous as regards cross section of the main, the gas is withdrawn through a petcock on a pipe screwed into the shell of the main and extending in beyond the brickwork lining. In the absence of lining the gas may be withdrawn through a petcock on the shell, or when the main is inaccessible, on a pipe leading from the main and through which a constant current of gas passes.

In sampling flue gas it is necessary that the gas be withdrawn through a perforated pipe, which enters with air-tight connection and extends entirely across the flue. The pipe is closed at the anterior end and provided at the outer end by an aspirating device, operated by air, water, or steam (see sketch). The perforations in this pipe should be an equal distance apart, and, to obtain an equal flow through each opening, their combined area should be less than the cross sectional area of the pipe, three-fourths being considered a safe figure. The gas sample is withdrawn from the sampling pipe through a petcock situated between the flue and the aspirating device. This pipe may also be used in sampling blast furnace or producer gas; however, in place of the aspirator, the gas is allowed to escape under its own pressure.

An apparatus for withdrawing an accumulative sample is shown in the accompanying photograph. This appliance is one of many that may be used and has the advantage of simplicity of construction and operation. In obtaining a sample, the petcock, permanently attached to the main, flue or intermediary pipe before mentioned, is connected by a rubber tube with one hole of the doubly perforated stopper in the two gallon aspirator bottle. This bottle is full of water which has been saturated with the gas to be sampled and which may also contain one per cent of sulphuric acid. The lower opening of this bottle is connected by a rubber tube with a similar opening of another bottle placed on a lower level, the rate of flow of water from the upper to the lower being controlled by a screw compressor. The petcock being open, the screw compressor is released and the gas rapidly displaces the water in the upper bottle. On closing the petcock, reversing the bottles, and opening the pinchcock on the rubber tube leading from the second hole in the rubber stopper, the gas is rapidly expelled into the air until the rubber tube constituting the exit is full of water. The pinchcock is closed, the bottles reversed, and the upper one filled with gas, the rate of inflow of gas being so controlled by the compressor that the bottle is filled in the designated time, one-half to twenty-four hours.

A portion for analysis is transferred from the bottle to the small sample tube (No. 2, Analytical Apparatus). This tube, made of suitable material, is two inches in diameter, eight inches long, has conically shaped ends terminated by petcocks and a capacity of about 300 cc. This sample tube being filled with water, including the upper petcock, to exclude all air, is attached to the rubber tubing constituting the exit from the gas bottle, which is also full of water. The position of the aspirator bottles is reversed, placing the gas under pressure, the pinchcock is opened, the upper then the lower cocks on the sample tube are opened and the gas is allowed to displace the water in the tube and flush through for a short time to saturate the moisture adhering to the sides. The lower, then the upper cock on the tube is then closed, leaving the gas under slight pressure. The sample is delivered to the laboratory and analyzed as soon as possible to minimize the loss of carbon dioxide.

When so desired, a gasometer may be used for withdrawing a larger sample of such size that calorimetric determinations may also be made on it. If rough gas is to be sampled continuously, it is first passed through a suitable apparatus for removing the dust. The water in the gasometer is used continuously and as long as practicable. A small sample for analysis is forced through a petcock on the holder and suitable rubber tubing connection into the sample tube in the manner previously described.

Analysis.

The Apparatus.—1. The apparatus shown in the accompanying Fig. 3 has been designed with particular reference to its adaptation to the methods subsequently described. The distinguishing features in its design are the capillary tube of 1 mm bore, the accessibility of the auxiliary gases, oxygen and hydrogen, and the central location of the burette in the apparatus, thus decreasing the error due to the capillary space. Drawings of this apparatus, which have been copyrighted, have been supplied to Messrs. Eimer & Amend of New York, the authorized makers, who are prepared to supply it in whole or in part.

The general disposition of the apparatus is obvious. On the extreme left end of the capillary the rubber tubing connection leading to the Kipp apparatus (No. 9) is placed, while the connection to the oxygen supply (No. 7) is made on the end of the first stopcock. To the first capillary stem is attached the explosion pipette (No. 8), the platinum leads within which are partly covered with glass to prevent the spark from short-circuiting; next in order the pipette containing potassium hydrate solution (No. 4) is placed, followed by the burette (No. 3). To the right of the burette are situated the pipettes containing stick phosphorus and fuming sulphuric acid (No. 6 and No. 5, respectively).

The stem of each pipette is ground so as just to clear the corresponding stem of the capillary with which it is connected. The stem of the burette must not be altered, but is to be precisely adjusted to the stem of the capillary tube by regulating the height of the support at the bottom. The calculated volume of the capillary tube is aprpoximately 0.2 cc, but since one-half is kept filled with water, the capacity of the burette and capillary is 100.1 cc. Since the majority of readings are made in the lower part of the burette, the first 75 cc are graduated in tenths, and by interpolation twentieths may be read. The remaining 25 cc are graduated in fifths, and may be read in tenths. The numerations are from the bottom upward. The burette is surrounded by a water jacket, thereby maintaining practically a constant internal temperature throughout an analysis.

The pipettes have two compartments, and to the open side a thin rubber bulb is attached. This protects the reagent from the atmosphere. The pipettes will conveniently hold 100 cc of gas and 260 cc of the reagent, with the exception of the one for phosphorus. This pipette was designed for the use of stick phosphorus, and when moderately filled with this reagent will hold 100 cc of gas, and sufficient water to form a seal. Thus the gas is exposed to the maximum surface area of the phosphorus, facilitating the removal of oxygen. In filling, the phosphorus is introduced through an opening at the lower end of the pipette. The stem rises into the base of the pipette for a distance of about ¼ in., thus preventing the sticks from obstructing the orifice.

2. A ½ pint Kipp apparatus for generating hydrogen from C. P. stick zinc and dilute hydrochloric acid.

3. An aspirator bottle containing oxygen and nitrogen in predetermined proportion.

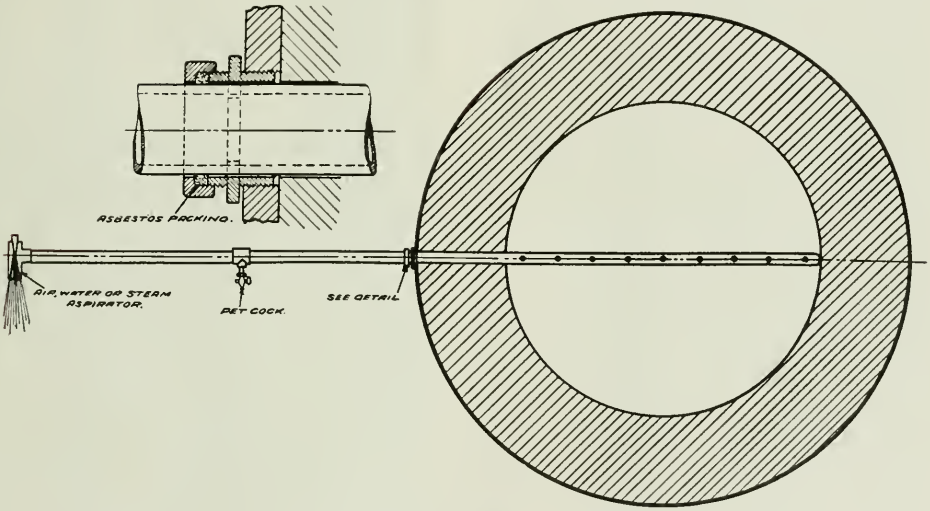


FIG. 1.—SAMPLING TUBE FOR FLUE GASES.

An induction coil of sufficient size to give a ⅛-in. spark. The current required for this coil is derived either from dry cells, a storage batter, or from direct-current lighting circuit with a suitable rheostat connection.

Reagents.—1. Potassium Hydrate. The pipette used for this reagent is filled with 260 cc of a solution of approximately 1.27 specific gravity. The absorbing power is in excess of 40 cc of carbon dioxide per cc of solution.

2. Stick Phosphorus. This reagent is prepared in sticks of about 4 mm or 5 mm diameter, and of suitable length for the absorption pipette, by melting the phosphorus under water in a test tube immersed in a vessel of water at a temperature of about 50° C. A glass tube of the proper internal diameter is inserted in the molten phosphorus, and a column of phosphorus is drawn into the tube by means of suction to the desired height, when the tube is withdrawn and immediately dipped into a beaker of cold water. The solidified phosphorus is then pushed out with a glass rod into the reagent pipette filled with water. When in use the water in the pipette is renewed occasionally to remove the oxides of phosphorus in solution. The need for this is apparent when the fumes in the pipette are

slowly or incompletely absorbed in one or two minutes. The chamber of the pipette containing the phosphorus is inclosed with a piece of black paper to protect it from the action of light.

3. Fuming Sulphuric Acid. Two hundred and sixty cubic centimeters are required. The acid should contain at least 20 per cent of sulphur trioxide in excess. It will become discolored owing to its action upon the rubber tubing, but this does not appear to interfere with its efficiency. This action tends to harden the tubing, after which it apparently ceases.

4. Dilute Hydrochloric Acid. A mixture of hydrochloric acid —100 cc, and distilled water—400 cc, is poured into the Kipp apparatus, the middle chamber of which is well filled with C. P. stick zinc.

5. Saturated Water. This is used to fill the burette, the explosion pipette, and the leveling bottles. Care should be taken not to contaminate this water with any of the reagent solutions, as the longer it is used, the less will be the liability of error due to the absorption of gas. It is obtained by causing the gas to bubble through distilled water contained in a bottle for a reasonable period of time. The water is filtered, 1 per cent of sulphuric acid is added, and the mixture well shaken.

Auxiliary Gases.—1. Hydrogen. This is generated in the middle chamber of the Kipp apparatus by the action of dilute hydrochloric acid upon C. P. stick zinc. The apparatus should be flushed several times with freshly generated hydrogen to remove all traces of air, and the rubber tubing connected to the left end of the main capillary tube.

2. Oxygen and Nitrogen Mixture. This mixture should contain from 35 per cent to 45 per cent oxygen and from 55 per cent to 65 per cent nitrogen. The container, a ½-gal. aspirator bottle, is assumed to have a volume in round numbers of 2000 cc. By calculation it has been found that a mixture of one-third oxygen, obtained from the oxygen tank, and containing between 95 per cent and 100 per cent oxygen, and two-thirds air will approximate 46 per cent oxygen and 54 per cent nitrogen in composition.

Suitable connections are made between two aspirator bottles, one being filled with water, the other containing enough to form a seal, and the rubber tubing connecting the two bottles is closed with a pinchcock. The volume of water in the former above the lower outlet is determined, and two-thirds of this amount is returned to the bottle, a mark being made on the side even with the water level; the bottle is then filled with water. A connection is made with the oxygen supply, the pinchcock between the two bottles released, and the oxygen is allowed to pass slowly into the bottle until the water reaches the indicated mark, the overflow passing into the second aspirator bottle. At this point the rubber tubing leading to the oxygen supply is disconnected, the second aspirator bottle is immediately lowered, and the air drawn in the upper bottle until the water level is even with the outlet. The end of the rubber tubing on the upper bottle is closed with a pinchcock, and the gases thoroughly mixed by agitation. The bottles are placed on their shelves, and the tubing connected to the stem of the three-way cock on the left of the gas analyzing apparatus.

Methods of Analysis.—1. Percentage of oxygen in oxygen and nitrogen mixture.

In the application of the following methods it has been found that a mixture of oxygen and nitrogen, in the proportion given before, is the most practical for use in the combustion of the component gases, but it is necessary to know the exact percentage of oxygen in this mixture in order that the amount of oxygen added for the explosions may be known. The object in introducing this mixture instead of oxygen alone is that a larger volume of the former can be added, and only a normal excess of oxygen will be present after the first explosion, thereby keeping the explosion ratio as high as practicable and reducing the flame temperature, thus minimizing errors due to the combustion of nitrogen. The explosion ratio is the proportion of the inert to the total gases entering into the combination. The oxidation of nitrogen cannot be entirely avoided since oxides of nitrogen are formed in all explosions in amounts varying chiefly with the temperature attained. The design of the explosion pipette, owing to its eudometric shape, is an aid in avoiding the combustion of nitrogen in that the flame movement is slower, and necessarily the production of heat is lower, than in the spherical form of pipette. Experiments have been made on the gases discussed in this paper, and approximately quantitative determinations made of the oxides of nitrogen formed in the combustions. It has been found that with this mixture, and the explosive ratios as high as practicable, the errors due to oxides of nitrogen are negligible. With pure oxygen, even though the explosive ratio is the same, the formation of oxides of nitrogen is markedly increased.

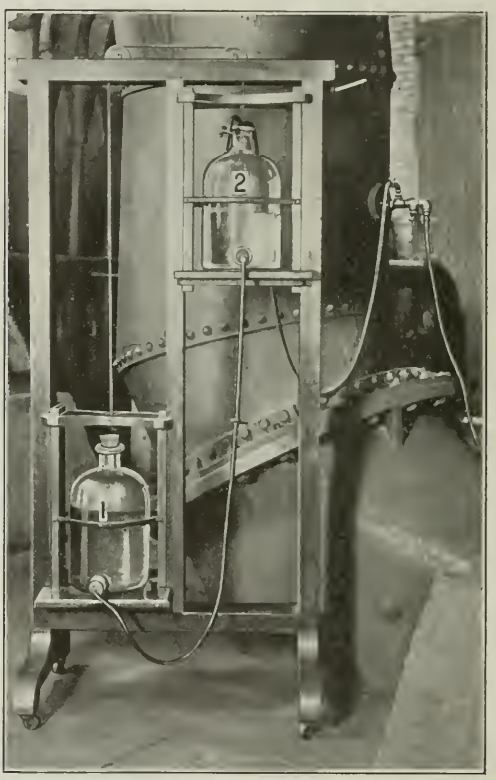


FIG. 2.—APPARATUS FOR THE SAMPLING OF GASES.

With the data of an analysis at hand, the explosion ratio for the first explosion is found by dividing the gas residue after Contraction II, by the sum of Contraction I and Contraction II, and the ratio for the second explosion by dividing the residue after Contraction III, by Contraction III. In the analysis of furnace, producer and flue gases, the most practical ratios of inert to the combining gases are those lying between 1.5 to 1 and 3 to 1; with producer gas the ratio obtained in the first explosion should approach as near as possible to the latter ratio, since the amount of heat liberated in the combustion of methane is relatively great.

Before beginning the analysis, it is advisable each day to agitate the bottle containing the oxygen and nitrogen mixture, and to flush out the connecting rubber tubing by lowering the leveling bottle (*L*), drawing in about 25 cc each of hydrogen

and oxygen and nitrogen mixture, and discarding through the three-way cock at the right. The solutions in the reagent pipettes are drawn flush with the main capillary tube, care being exercised that there are no unfilled spaces in the capillaries of the pipettes. The left half of the capillary is filled with water drawn over from the explosion pipette, the levelling bottle is now raised and any portion of the solutions that may have been drawn into the main capillary are expelled through the stem of the three-way cock at the right, to which is attached a piece of rubber tubing discharging into a suitable waste receptacle. While the levelling bottle is still raised the three-way cock is closed and in this manner the capillary tube is left filled with water. After the gas has been passed on the left side for absorption or explosion, or after the addition of each of the auxiliary gases, the capillary tube may be filled with water from the explosion pipette before a reading is taken, thereby avoiding the error due to the capillary space of 0.1 cc. The leveling bottle is placed at the base of the apparatus, and approximately 75 cc of hydrogen and 25 cc of oxygen and nitrogen mixture are added in succession, each being accurately measured by bringing the water in the leveling bottle to the same level as that in the burette. From a half to one minute should be allowed for the burette to drain before a reading is taken. The leveling bottle is raised and the cock leading to the explosion pipette (8) is opened, and the gas passed over and back two or three times to insure a thorough mixture, as an imperfect mixture may prevent the desired explosion.

The electric circuit is then closed causing the explosion. The bottle (Y), which receives the overflow from the explosion pipette, and the location of which on the lower shelf is never changed, the gases thereby being kept under partial vacuum, is connected to the latter by a piece of strong rubber tubing about 1 ft. long, partly constricted by a screw compressor near the bottle. The degree of the constriction is determined by increasing it until no bubbles appear in the pipette after an explosion. All the conditions of this arrangement tend to diminish the force of the resultant explosion.

After the explosion, the gas is immediately drawn back into the burette until the water from the pipette fills the left half of the capillary tube. The cock in the pipette is then closed, and after an interval of one minute for allowing the gas to cool, a reading is made. This less the previous reading represents the contraction due to the formation and condensation of water and is known as Contraction I. The oxygen that entered into the explosion is equal to one-third of the contraction, since

$2H_2 + O_2 = 2H_2O$. 2 volumes + 1 volume = liquid (disappearing).

$$\text{Percentage of oxygen in the mixture} = \frac{\text{oxygen and nitrogen mixture added}}{\text{oxygen found}} \times 100.$$

Example Data

	Burette Readings.	
	100.0	
Add hydrogen.....	74.6	Hydrogen added.
Add oxygen and nitrogen	25.4	
Mixture	25.2	Mixture added.
	.2	
Explosion	33.6	Contraction.
	33.8	
	11.2	⅓ Contraction or Oxygen found.

Calculations.

$$\frac{\text{oxygen found}}{\text{oxygen and nitrogen mixture added}} \times 100 = 44.4 \text{ per cent oxygen.}$$

Explosion ratio, 1.97.

Two or more determinations are made, which should agree within 0.1 per cent or 0.2 per cent. If there is question as to the accuracy of the results obtained, they may be proven by using air as a standard, and subjecting it to conditions similar to those just described. It being known that there is 20.9 per cent of oxygen in air, it follows that if the oxygen factor as previously obtained is correct, the determined amount of oxygen in the sample of air should equal the theoretical.

The residue of gases from the previous analysis is rejected and about 60 cc of air are drawn into the burette and the burette is allowed to drain. The leveling bottle is placed on top of the apparatus, the tube leading from it is pinched tightly, and the three-way cock opened, the pressure of the gas being thus released. The remaining portion is slowly forced out by releasing the pressure on the tube until the meniscus just reaches the 50-cc mark in the burette, and the three-way cock is closed. When the leveling bottle is applied

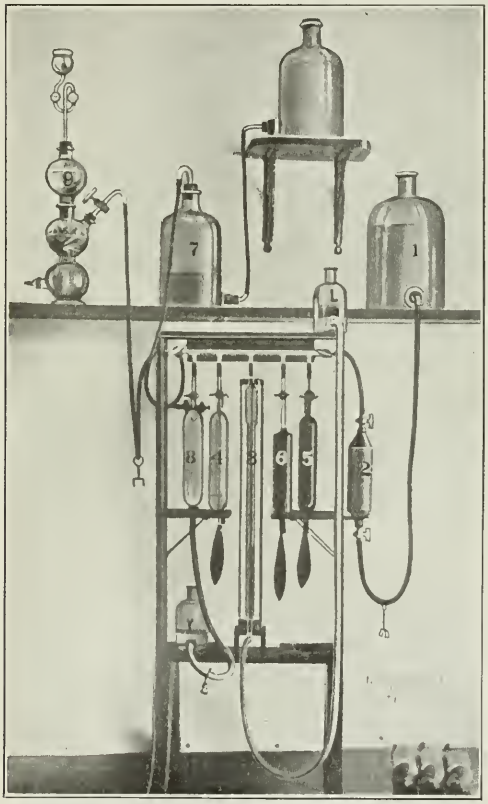


FIG. 3.—APPARATUS FOR THE ANALYSIS OF GASES.

there should be exactly 50 cc of air in the burette and to this approximately 20 cc of oxygen and nitrogen mixture, and a like amount of hydrogen, are added, a reading being taken in each case. The gases are mixed, exploded, drawn back into the burette and, after allowing one minute to drain and cool, Contraction I is noted. Hydrogen is then added until the volume of the mixture approaches 100 cc. The gases are again mixed and exploded, drawn back into the burette, and Contraction II is noted. It follows from the reaction previously stated that

$$\text{Total oxygen} = \frac{\text{Contraction I} + \text{Contraction II}}{3}$$

Oxygen in sample of 50 cc of air = total oxygen − oxygen added.

The result should approximate 10.45 and if so, proves the factor to be correct.

Example—Burette Readings.

100.0	
Add Air.....	50.0 Air added.
50.0	
Add Oxygen and Nitrogen Mixture...	20.0 O and N Mixture added (44.4%O_2). 8.88 Oxygen added.
30.0	
Add Hydrogen.....	19.6 Hydrogen added.
10.4	
Explosion	29.4 Contraction I.
39.8	
Add Hydrogen.....	38.8 Hydrogen added.
1.0	
Explosion	28.6 Contraction II.
29.6	

Calculations.

$$\text{Total Oxygen} = \frac{\text{Contraction I} + \text{Contraction II}}{3} = 19.33$$

Oxygen in sample of 50 cc of air = total oxygen — oxygen added = 10.45 cc.

Oxygen found in 100 cc of air = 20.90 cc.

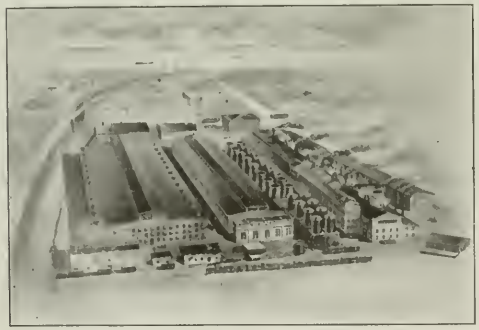
Explosion ratios: I, 2.05; II, 2.46.

The percentage of oxygen may also be found from the same data; the oxygen in the mixture is the difference between the total oxygen found and that contained in the air used, the percentage is found by dividing the oxygen in the mixture by the volume of mixture used and multiplying the quotient by 100.

(To be concluded in the next issue.)

Hooker Electrochemical Co. Works.

We are glad to show in the accompanying illustration the plant of the Hooker Electrochemical Company at Niagara Falls, N. Y., for the production of caustic soda and chloride of lime by the Townsend electrolytic cell, which they purchased in the early stages of its development and have since perfected and successfully operated. It is a splendid example of concrete steel construction and is the result of the gradual expansion of the business since 1905. The company now ranks among the foremost producers of heavy chemicals in this country. In 1910 a portion of the plant was seriously damaged by fire, but was quickly replaced by construction of the most permanent character, and full operation was resumed early



NIAGARA WORKS OF HOOKER ELECTROCHEMICAL CO.

last fall. During the shutdown the company fulfilled its contracts for the delivery of material, waiving the rights secured to it by fire clauses in its contracts. This action is indicative of the broad business policy which the Hooker company has pursued since its inception.

The Townsend cell, which is comparatively simple and inexpensive both in construction and operation, has been successfully used in the mining industry to produce chlorine for the chlorination of ores, and is recommended and licensed by the Hooker company for this purpose.

Evaporators and Vacuum Pans.

By B. Viola, M.E.

The Kestner Evaporator.

The principle of the Kestner evaporator is the climbing film and it introduces quite a novel condition in the evaporator business, having special advantages for the evaporation of delicate materials. Numerous experiments made have demonstrated that the liquor is in contact with the heating tubes for almost an infinitesimal time, thereby permitting the use of higher pressures without damage to the material, or lower pressures, thereby producing a material of high quality.

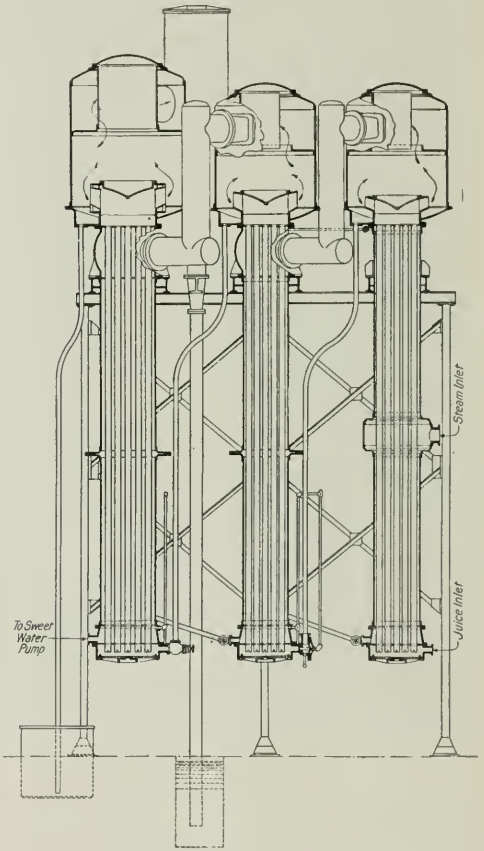


FIG. 29—KESTNER EVAPORATORS.

The liquid enters the base of No. 1 cell (*Fig.* 29). This should be fed in at a temperature approximating the boiling point in this body. The steam, as shown, surrounds the outside of the tubes, and as the liquor is at the boiling point it is, of course, ready to begin boiling at once. Small bubbles are formed at the base of the tubes, which readily grow larger until the entire core is filled with a solid mass of steam passing rapidly up the tube. The speed of this steam or vapor is such that it carries with it a thin film of liquid on the inside of the tube, and in virtue of the rushing steam this thin film is continually passing up and is pressed in close contact with the hot

walls of the tube. On reaching the upper tube sheet the mixed vapor and liquid pass through the so-called centrifugal separator, which consists of a series of curved blades having tangentially presented openings through which this mixture passes at high velocity. The liquid, in virtue of its greater specific gravity, is thrown farthest and is projected against the walls of the vapor belt or containing vessel. As this strikes these walls at an angle they impart to it a rotary motion,

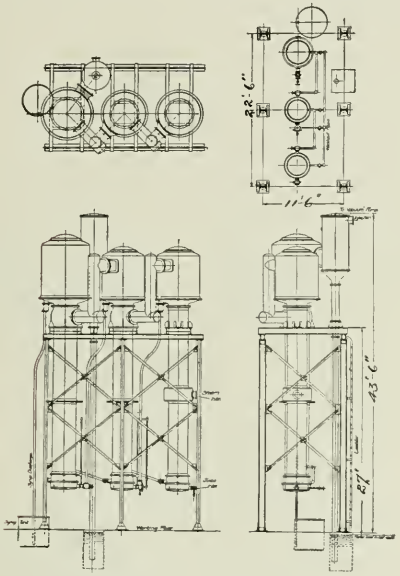


FIG. 30—TRIPLE-EFFECT KESTNER EVAPORATOR.

which further permits the complete separation of the vapor from the liquid. The major portion of the vapor, on account of its lower specific gravity, is not thrown so far as the liquid, and passes up at an intermediate point through the baffle or catchall, which forms the upper portion of the vapor belt. This catchall is arranged in the well-known manner and is inserted simply as a precautionary measure.

The vapor then passes through the vapor pipe between bodies No. 1 and No. 2 and enters the steam belt of No. 2, performing the same operation as in No. 1. The liquid, however, separated from the steam, passes down the pipe and through the automatic valve into the bottom of body No. 2, where the same operation takes place as in No. 1.

The automatic valve is a balance valve operated by a diaphragm. One side of the diaphragm is subjected to the pressure of the liquid in the pipe between bodies Nos. 1 and 2. The other side of the diaphragm is connected through what is termed the balance pipe with the steam belt of body No. 2. It is the practice to fill the vertical leg of this balance pipe with water. The hydrostatic head on one side of the diaphragm is, therefore, equal to the head of water in the balance pipe; and on the other side the head of juice in the liquor pipe. The pneumatic pressure on both of these columns is the same because steam belt No. 2 is at the same pressure as vapor belt No. 1. When these two hydrostatic heads are equal the valve is closed.

Should the head of liquor rise in the liquor pipe so that it exerts a greater hydrostatic pressure on its side of the diaphragm the valve will open and allow the liquor to pass until the hydrostatic head balances that of the balance pipe. In practice this valve remains partly open the whole time the evaporator is working. A similar valve might be used on the liquor pipe between 2 and 3. The drawing, however, shows an angle valve which can be adjusted by hand.

The water of condensation from body No. 1 is ordinarily carried into body No. 2 and use is made of the thermal units set free in virtue of its higher temperature. In the same way the water of condensation passes from No. 2 into No. 3. As the steam belt of No. 3 is at a partial vacuum, the water of condensation must be removed by means of a pump.

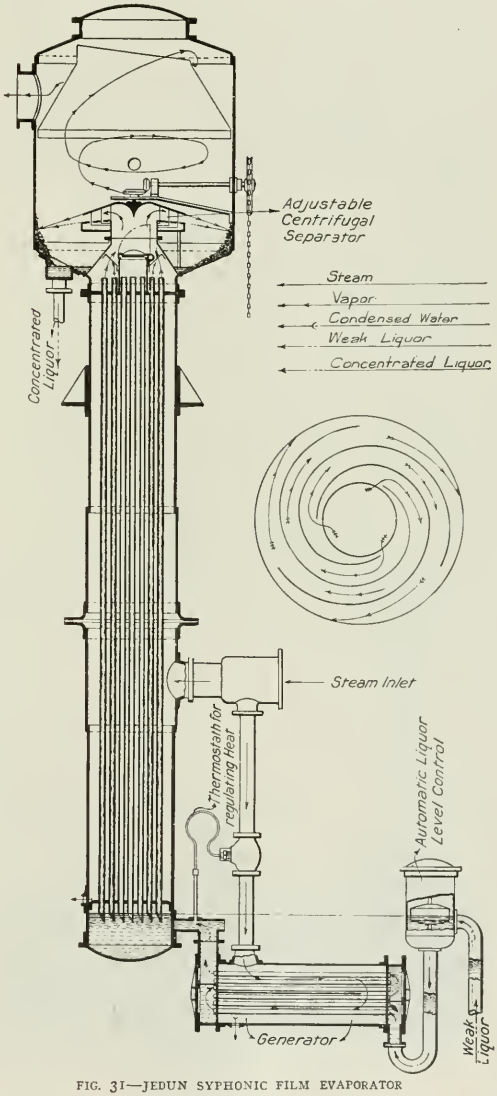


FIG. 31—JEDUN SYPHONIC FILM EVAPORATOR

The discharge of liquor from the last body takes place by virtue of the barometric height. The vapor from the last body passes into the condenser through the opening shown in the back of the last vapor belt. *Fig.* 30, the plan view of a triple effect, shows clearly the connection to the condenser.

Jedun Syphonic Film Evaporator.

As shown in *Fig.* 31, this evaporator's action, which has a

great similarity to the Kestner evaporator and works on the same principles, begins with a generator whose object is to generate a proportionate amount of vapor bubbles which enter the base of the evaporating tubes, carrying with them small pistons of layers of liquid which are contained in the expansion chamber. If the vapor bubbles are generated and supplied continuously the rising of the bubbles and layers of liquid will also be continuous within the evaporating tubes. The height

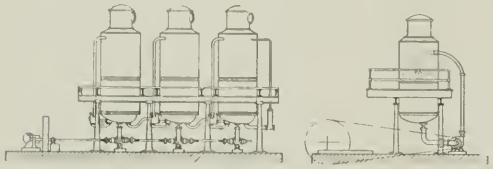


FIG. 32.—MEYER & ARBUCKLE'S REVOLVING DISTRIBUTOR ATTACHED TO A TRIPLE EFFECT.

to which the layers of liquid and vapor bubbles rise is governed by the amount and temperature of the liquid admitted to the expansion chamber.

The action of this evaporator is explained by the following: if a liquid and a vapor having the same temperature are allowed to rise in a current in small tubes the vapor bubbles, notwithstanding their lower specific gravity, will always rise through the tubes, carrying in front of them small pistons of liquid. Now, if the tubes which receive the bubbles of steam and liquid are surrounded by a heating medium, say steam of a temperature somewhat higher than the temperature under which the liquid or pistons of liquid and bubbles are to be evaporated, it is reasonable to say the evaporation of the liquid carried into the tubes by the vapor bubbles is very rapid, this depending upon the amount of vapor bubbles and liquid admitted into the tubes.

The vapor bubbles expand enormously and burst and the liquid evaporates owing to its contact with the heated surface of the tubes, which increases the volume of the vapor in a very large proportion, while the volume of the liquid decreases. As

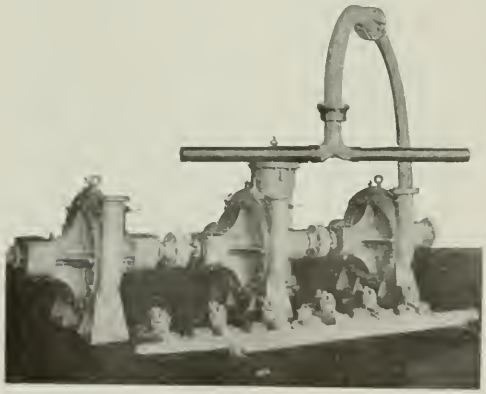
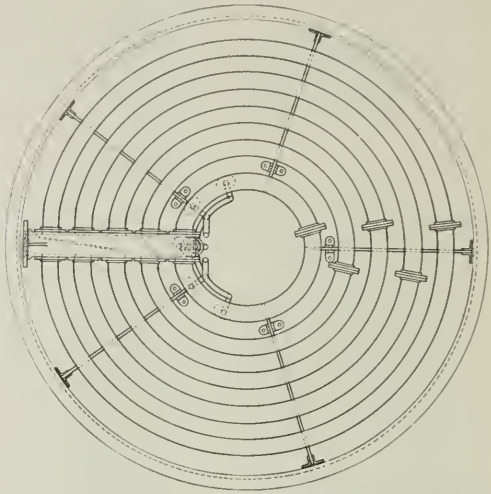


FIG. 33—MEYER & ARBUCKLE'S REVOLVING DISTRIBUTOR CONNECTED WITH THE CENTRIFUGAL PUMPS.

the vapor in proportion is many times larger in volume than the liquid, this causes it to rush through the central portion of the tubes at such a velocity that it syphons or pulls with it along the heated walls of the tubes a thin film of liquid, which is discharged together with the vapor through an adjustable centrifugal separator into the vapor belt or chamber.

The action of this centrifugal separator is based on gravity, centrifugal force and the converting of vapor bubbles into solid drops through causing them to burst by allowing the bubbles to expand from a higher to a lower temperature.

The vapor and liquid discharged from the evaporating tubes enter the centrifugal separator and by the arrangement of the centrifugal vanes therein have imparted to them a centrifugal motion, and also an increase in pressure or temperature through the adjustment of the head or cover of the separator. The average velocity of the vapor through the separator is from 80 to 120 ft. per sec., while the average velocity of the



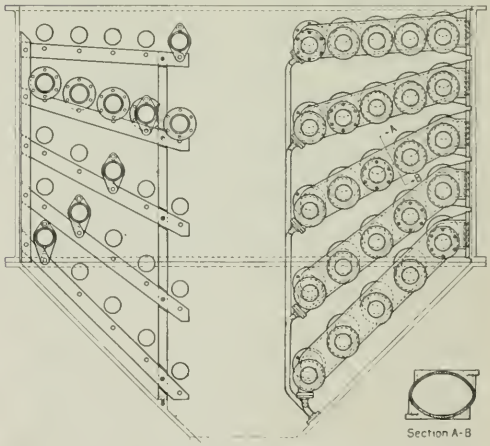


FIG. 34—DYER MANIFOLD HEATING COIL.

vapor ascending in the vapor belt into which the separator discharges is 3½ ft. per sec.

The drops of liquid and vapor thrown off from the separator retain their original velocity. The drops of liquid being heavier than the vapor, combined with the centrifugal motion or force imparted to them by the separator vanes, and the downward angle at which they are thrown from the periphery of the separator, cause them to strike against the wall of the vapor belt, where they flow downward to the discharge pipe, while the vapor ascends free from all moisture and entrainment and passes on through the vapor main to the next effect or condenser.

Arrangements are made also whereby the evaporator can be worked at full, half, quarter or any capacity that may be required. To meet these varying requirements a hood is arranged encircling any given number of tubes. Thus the capacity is controlled at will.

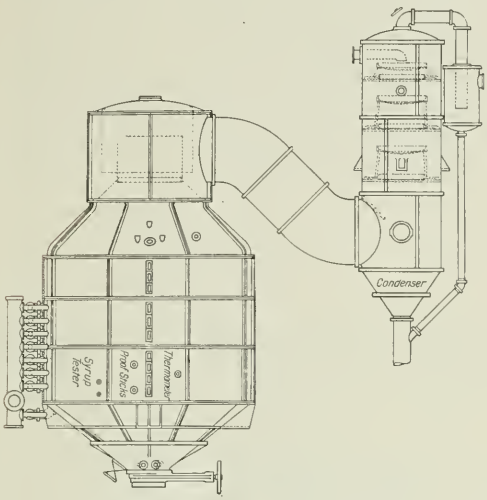


FIG. 35—DYER VACUUM PAN AND CONDENSER.

Meyer & Arbuckle's Film Evaporator.

This evaporator, or in other words, a film sprinkler, is a very original idea. It is in fact nothing else than an attachment to any evaporator with horizontal or vertical heating tubes. It is an automatic revolving distributor suspended over top tube plate in vessel, combined with centrifugal or other pumps for mechanical circulation, is the principle applied in obtaining an ideal film over the whole heating surface. Fig. 32 shows how it is attached to a triple effect, being provided with centrifugal pumps for bringing the liquid into the evaporator. The distributors, which are entirely automatic, are actuated by the escape of liquid through perforations in the arms bolted to the tee piece over the tube plate. The perforations in one arm are diametrically opposite to those in the other arm, thus there are no working parts beyond the ball bearings in the tee piece. The velocity is readily controlled by the increasing or diminishing angle perforations in relation to the top of tube plates.

Centrifugal velocity is imparted to the fine jets of liquor by distributors, resulting in a swirl down the interior heating surface of the tubes, leaving at the same time the center of the tubes free for rapid escape of vapors.

According to experience this attachment increases the efficiency of an evaporator 33 per cent (Fig. 33).

The Concentrating of Solutions Without Heat.

Experiments have been made lately for concentrating solutions without heating them, which process is based on the well-known osmosis. Vegetable membranes and artificially produced membranes of precipitates are known already which are semi-pervious, namely, which permit the solvent but not the dissolved crystalloids to pass through them. Numerous experiments made by A. Farago have demonstrated that most of the precipitates which cannot be dissolved in water or in other solvents possess the same property. In other words, they are semi-pervious, if prepared in a sufficiently thin layer.

The semi-pervious substances which are used in carrying out the process to which this invention relates are prepared as follows: Porous clay tubes, which can be closed at their edges, are filled with a solution of salt, one of the components of which is the component of the semi-pervious membrane to be produced. The tubes are then closed and placed in a bath which contains another solution of salt, one of the components of which is the second component of the semi-pervius membrane to be produced. The two solutions meet in the walls of the tubes and from there a new compound which in a finely divided condition fills the pores of the clay tubes. The tubes after being removed from the bath, emptied and washed out, will now be semi-pervious.

Now the essential novelty of this invention consists in the fact that the liquid to be concentrated is caused to pass through a system of tubes produced in the manner above described while a concentrated salt solution is caused to flow in the opposite direction around this system of tubes. If, from tubes of the character above described, there be constructed a system of tubes of sufficient length, surrounded by a second set of tubes made of glass, copper, stoneware, wrought-iron or any suitable material, and if the aqueous solution to be concentrated, be caused to flow slowly by means of a pressure pump through the inner of the two sets, while a concentrated salt solution is at the same time caused to flow in the opposite direction through the surrounding outer tube-system, there will arise between the two solutions an osmotic reaction through the semi-pervious clay tubes which will result in the passage of water from the aqueous solution into the concentrated salt solution.

As the two liquids flow in opposite directions it follows that the more concentrated liquid will gradually meet with the more

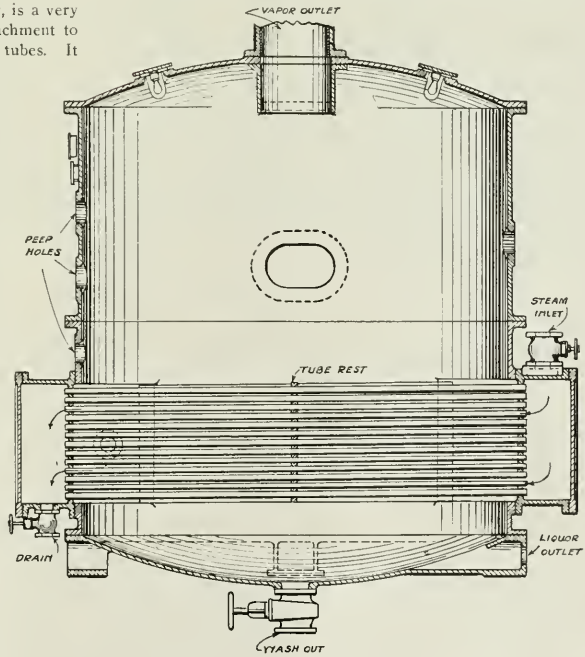


FIG. 35A—ZAREMBA VACUUM PAN.

concentrated salt solution, and as osmosis will continue as long as the two liquids differ in their degree of concentration the liquid to be concentrated will finally pass out of the tubular system with a degree of concentration corresponding to that

of the salt solution in the outer tubular system, while the salt solution used to effect concentration will pass out of the outer tubular system diluted to an extent corresponding to the dilution of the liquid to be concentrated.

The form of the tubular system used in carrying out the process may be of any desired kind, so long as it fulfills the condition of presenting as large surfaces as possible without taking up much room. The advantage of the process to which this invention relates, in comparison with those hitherto in use, consists in the fact that solutions which can be heated and evaporated only with a certain amount of risk, such as explosive substances or solutions which easily become decomposed when heated, can by means of this process be concentrated without any risk whatever.

This shows clearly that long heating coils will give hardly more than one-third of the efficiency compared with short, inclined tube heating systems. This is natural, for after steam enters the coils the condensing water will cover the lower part of the inner surface of the coil and the more it nears the end the more condensing water will be present. This water isolates the heating surface of that part of the coil entirely. Therefore the most efficient heating surfaces for vacuum pans should be made from short inclined tubes. By adopting this system, which has the advantage of greater economy and larger output in a shorter time, the present vacuum pans with long horizontal and vertical tubes will have to be reconstructed.

To improve this disadvantage of the heating-coil system, a very ingenious idea was worked out by a well-known Ameri-

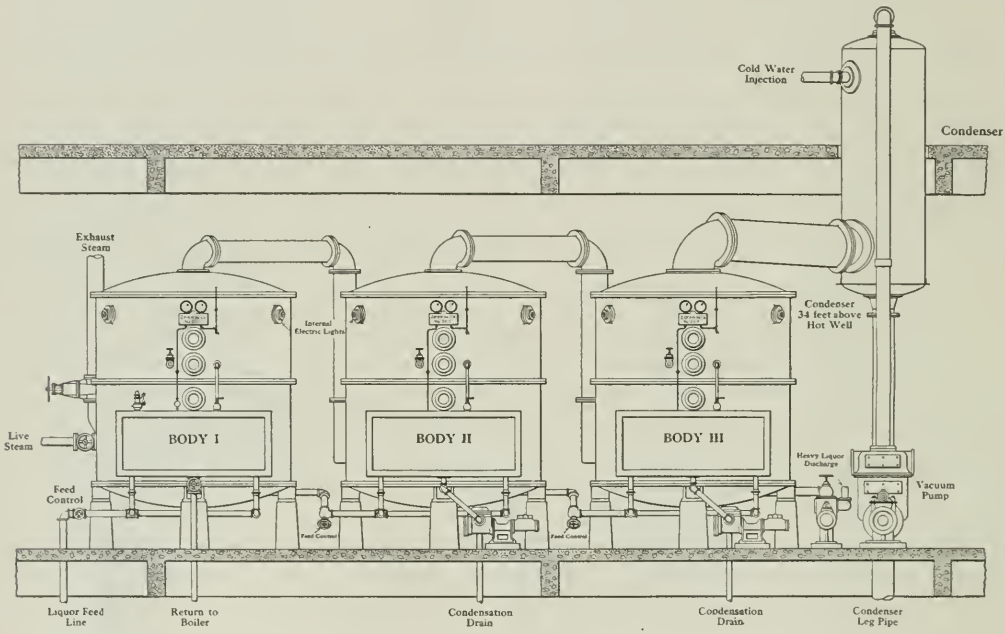


FIG. 35B—ZAREMBA TRIPLE-EFFECT EVAPORATOR.

Another advantage of this process, should it be practiced, consists in the fact that it is no longer necessary to employ complicated evaporation apparatus or to use fuel.

Vacuum Pans.

The vacuum pan is in fact nothing else than a single effect evaporator with the difference that in this apparatus is done the evaporating of the thick juice to massecuite in the sugar line. Formerly this process was effected in open kettles, which were treated by open fire or steam. Evaporation is produced now only in vacuum pans with the assistance of condenser and vacuum pumps.

It has been shown above how advantageous evaporation is by vacuum and low temperature. In vacuum pans mostly direct steam is used with higher pressure. Their shape is usually cylindrical and they are heated by heating coils, whose length is not more than 200 times the diameter. As mentioned before, the heating capacity of a heating coil is the largest where steam enters the coil, and decreases until it reaches the outlet. Researches inaugurated in the engineering laboratory of the Technical High School of Dresden by Dr. Eng. William Nusselt have led to the surprising result that the coefficient of transmission of heat is dependent not only on the difference of temperatures, but also on the length of the piping.

can vacuum pan building firm as follows: The manifold form of the coils is arranged in such a way that the number of pipes through which the heating system circulates is reduced from time to time, and this is in proportion to the estimated reduction in volume, so that the velocity of the vapor is maintained, as is also the pressure in all parts of the heating system (*Fig.* 34).

One of the most modern improved vacuum pans supplied with the heating coils shown in *Fig.* 34 can be seen in *Fig.* 35. This is a Dyer vacuum pan, 16 feet in diameter and equipped for live and exhaust steam heating, and is connected with a very efficient barometric counter-current condenser. There the drips or drains from all heating coils are connected to two manifolds outside of the pan.

One manifold, called the "live steam drip manifold," receives the drain water from all coils which at any particular time may be using live steam. The exhaust steam manifold in the same way receives the drains from the coils which are using exhaust steam. The water of condensation from the live steam drip manifold flows by gravity to a closed receiver, from which the boiler feed pump delivers it directly into the boilers without loss of temperature due to reduction of pressure. The drain from the exhaust steam manifold is handled in the same

way by a separate receiver and pump. The total heating surface of this pan is 2280 sq. ft., it has a juice catcher in the top and a very suitable connection to the condenser.

A cylindrical vacuum pan, with horizontal heating tubes is shown in Fig. 35*a*. This is the Zaremba system, with cast steel; it is also built of steel, copper or other metal. The heating surface is made of seamless drawn tubing of copper, brass, steel, etc., as the conditions may demand. The tubes are all of one length and are secured in place by an improved type of elastic packing that is absolutely tight and at the same time is easily taken out when the tubes are to be removed. Sagging of tubes at the center, in large machines, is prevented by the use of a tube rest. Steam chests can be so arranged as to make possible a large increase in heating surface whenever desired, by merely inserting additional tubes.

The steam circulation is from right to left through the tubes, which are rapidly drained of their condensation by being given ample incline. Liquor circulation is upward around the tubes and downward through large segment-shaped downtakes between the tube nest and the sides of the evaporator. Both steam and liquor circulation is very rapid, thus yielding a great efficiency of the heating surface.

The steel structure is simple, with only three readily kept tight joints; no internal bracing is required and yet an internal pressure can be carried with perfect safety. The bottom plates are so designed as to give either much settling space under the tube nest, or no such space, as may be desired.

The vapor space above the tube nest is made high and is so arranged that its entire volume is efficiently used to prevent entrainment loss. As an additional safeguard against entrainment the pans are equipped with an internal separator which returns to the liquor body any liquor particles that may have escaped with the vapor.

This construction (Fig. 35b), combined with a multiple effect, is used with excellent result as an evaporator.

(*To be continued.*)

Pittsburgh Meeting of American Foundrymen's Association

While going to press this year's convention of the American Foundrymen's Association and allied societies is still in progress in Pittsburgh. The attendance is again enormous, and it may be stated that there are an unusual number of chemists, metallurgists and assayers in attendance. In the following we give brief notes on some of the papers presented.

Standard Test Bars for Cast Iron.

Dr. Richard Moldenke, the untiring secretary of the association, presented a memorandum on standard tests for cast iron.

At the Copenhagen Congress of the International Society for Testing Materials the so-called "arbitration test bar for cast iron" was fully discussed, and comparisons made of the practice prevalent in the several interested countries of the world. The test bar in question is 1¼ in. in diameter, and 14 in. long, being broken transversely on supports 12 in. apart.

The comparison of the practice showed that so far as the diameter of the bar is concerned there is a general accord. Germany, Italy and the United States have adopted to 1¼ in. diameter, or metric dimensions very close to this. England, while quite favorably inclined, has not yet acted officially on the subject.

Further, the manner of preparing the test bar seems to be in accord everywhere in its main features. Bars are to be cast on end, in dry sand. Germany differs only in pouring from bottom up, whereas all the other countries pour directly into the top. German practice is undoubtedly the better one, though more expensive.

The radical difference in the several standards seems to lie entirely in the length of the bars for test. We break at 12 in. between supports. Germany makes it almost two feet, while Italy takes the distance only six inches. It was, therefore, suggested at the Copenhagen Congress that a series of tests be made in the several countries interested, with bars ranging from six to twenty-four inches between supports in breaking them, and from the results obtained that length be selected which will give the best all-around values.

Mr. Walter Wood, of R. D. Wood & Company, the chairman of the Commission on Cast Iron, took up this work for the United States, and reported at the last meeting of the American Society for Testing Materials. Both transverse and tensile tests were made. One set of the transverse tests would indicate a steady downward progression of the modulus of rupture for cast iron, as the span increases. But from the results so far obtained the need has become evident to extend the series of tests to smaller spans, say as low as 6 in. With the results extended over spans from 6 in. to 20 in. it would be possible to plot a curve showing the relation of the span to the modulus of rupture of the material, the turning point being found when too great a stiffness and consequent high results change to too great a weakness, or possible lack of sensitiveness on the part of the test, on the other side. This point, when found for the several classes of iron at issue, would be the desirable length of the span for a standard test.

Foundry Coke and Foundry Chemistry.

Mr. H. E. Diller, as secretary of the company, presented a set of the proposed standard methods for determining the constituents of foundry coke. This will be published in full in one of the next issues.

An instruction paper on foundry chemistry by Mr. Herbert E. Field gave definitions of entectics and segregation, and discussed the effect of phosphorus on cast iron.

The Electric Furnace for Steel Castings and General Foundry Work.

This interesting subject was discussed in a brief note by Dr. Paul Héroult.

At the present time some of the Héroult electric furnaces are busy on steel castings; in fact, some are exclusively erected for that purpose; for instance, at the Fischer Steel Foundry, Schaffhausen, Switzerland; Lake and Elliot, Baintree, England, and out of forty others in operation most of them are intermittently employed for castings, as for instance, at Worcester and South Chicago. The steel is perfectly sound, of high quality, and is good and hot when cast. It is so dead when poured that they do not need to increase the percentage of silicon and they run 0.15 and lower.

This, however, has been proved long ago. A more interesting feature of the electric furnace is that it is capable of improving the quality of foundry iron, and this at very low cost and by a very simple operation.

The main feature of what is called strong iron is the low content of sulphur. The removal of sulphur is one of the easiest and most effective operations that can be performed in the electric furnace. It consists simply in pouring into the furnace a charge of molten pig iron, if possible direct from the blast furnace; if not, from a cupola or other melting apparatus, then heating the metal under a basic slag which does not have to be scraped or removed except when it is teemed into a ladle with the metal ready for pouring. The contents in carbon, silicon, manganese and phosphorus are not affected by this operation unless this be desired. Common Bessemer iron, worth anywhere from $10 to $14 per ton can be changed into strong iron, charcoal iron, carwheel iron, or so-called cold-blast charcoal iron for a cost of about $1 per ton.

It seems probable that on account of the high grade of material so cheaply produced most iron castings of the future will be made of this improved quality. This new improved product will help to give new life to the iron foundry industry and will enable it to compete with the steel-casting industry.

Titanium in Iron.

The effect of titanium in iron was discussed in a paper by Mr. Charles V. Slocum, of the Titanium Alloy Manufacturing Co. The gist of the paper is contained in the following conclusions: Very small percentages, as low sometimes as 1/10 of 1 per cent (0.1) of alloy (only 0.01 Ti), are sufficient to cleanse the iron of impurities not touched by limestone or fluor-spar, while larger percentages increase the improvement in other directions, making the iron more fluid (usually hotter), making the castings sharper, finer grained, free from blow-holes and pin-holes, and easily machined, while at the same time the expense runs from 25 cents per ton of castings up to say $3 as a maximum, all of which is often repaid in the reduction of bad work and in improving poor irons.

Manganese and Silicon in the Foundry.

A paper by Mr. Alexander E. Outerbridge, Jr., deals with the use of manganese and silicon for foundry purposes. With respect to manganese he referred to the long series of his investigations with this element in connection with the special kind of cast iron used for making chilled cast iron car wheels. Formerly such wheels were made entirely from cold-blast charcoal iron, but this was replaced by the author's manganese treatment in the ladle. (*Jour. Franklin Institute*, March, 1888.)

Mr. Outerbridge points out that manganese acts in two different and opposite ways in cast iron.

When alloyed therewith in the cupola in considerable quantity, say 2 per cent or over, it has a chilling and hardening effect, producing what he terms a spurious chill of coarse crystalline nature, in contradistinction to the normal chill in a good car wheel which has a fine and closely interwoven crystalline structure.

When the alloy ferromanganese is added in a ladle of molten car-wheel iron in small quantity (1 pound of alloy, containing about 80 per cent manganese, in 600 pounds of iron), it acts not as an additional contribution of 0.133 per cent manganese to the metal in the ladle, but simply as a deoxidizing and desulphurizing flux, cleansing the metal from impurities, softening it and greatly increasing the ductility and strength without injuring the chilled tread of the wheel.

In view of these two directly opposite effects of manganese the author thinks that the ignorant and improper use of ferromanganese in general foundries is sure to lead to disappointments.

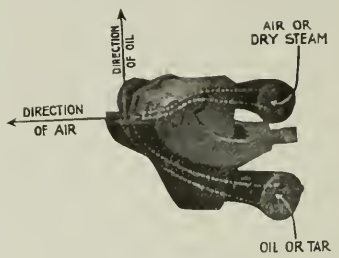
While ferromanganese is best adapted to the treatment of high chilling iron for carwheels or chilled rolls, or other chilled castings, where the proportion of combined carbon is large, on the other hand ferrosilicon is best adapted to treatment of foundry iron when from any cause it is hard and brittle, since it possesses the peculiar property of softening and at the same time strengthening such iron. It also enables the founder to vary the grade of iron in individual ladles to suit individual castings or groups of castings. It gives the founder control of his iron after it has been withdrawn from the source of melting, a matter of great importance and value, and it enables him to use cheaper grades of iron.

But neither ferromanganese nor ferrosilicon can be regarded as universal panaceas, and while each in its proper place is of great value, they must, like all other good things, be used intelligently, and if impure adulterated materials are employed they will not only prove to be of no benefit but must be absolutely harmful.

Oil as Fuel for Air Furnaces.

Mr. W. N. Best pointed out that, according to analysis, oil has a much higher calorific value than coal, and therefore ought to be able to melt the metal in a much shorter period of time than coal. Further, when oil is used as fuel the castings contain no more sulphur than the metal did when charged into the furnace, and the tensile strength is consequently greater than of metal melted by coal fire. As the melter has the furnace under perfect control the heats can be taken off much quicker than while burning coal, and the temperature of the charge while being tapped can be maintained without varying more than 25° Fahr. until all the charge has been run from the furnace.

The change from coal to oil is said to be a very simple matter. In the original firebox he constructs a combustion chamber of such form and proportions that the air necessary for perfect combustion can unite with the atomized fuel before it



USE OF OIL FUEL FOR AIR FURNACES

reaches the furnace, which prevents oxidation of the charge. Also this chamber causes the heat to be deflected upon the entire surface of the bath.

In the end of this combustion chamber he places a hydrocarbon burner of peculiar form, which makes a fan-shaped blaze, filling the entire chamber with flame. This burner exteriorly atomizes any gravity of liquid fuel which will flow through a ½-in. pipe, such as fuel oil, crude oil and tar. Its construction is such that it cannot carbonize, and there are no parts to wear away or get out of order.

A very small quantity of compressed air is used through the burner to atomize the fuel and distribute the heat, while the balance of the air necessary for perfect combustion is supplied at from 3 to 6 oz. pressure through a volume air nozzle.

The furnace is charged in the usual manner. The burner is started by opening the air valve, holding a piece of burning waste (which has been well saturated with kerosene) by means of a pair of pick-up tongs under the burner, and then turning on the oil.

The reduction in the time required to get the charge ready for tapping is not the only point wherein oil is more economical than coal. There is no handling of fuel and ashes, consequently the service of the fireman and coal passers are dispensed with. There is great saving in floor space for the oil tank is placed underground and the former coal bins used for other purposes. The fire brick lining of the furnace lasts 20 per cent longer than with coal. Poor castings or imperfect ones caused by the metal being cool or sluggish are obviated entirely.

This method is used by the Barlow Foundry Company, in Newark, N. J., and the Malleable Iron Company, Oriskany, N. Y.

Cupola Melting Practice.

A paper by Mr. P. Munnoch dealt with various phases of cupola practice, with special reference to British practice. A few points were considered more in detail.

One of these is oxidation during melting. There are two effects which result from oxidation, one of which is the reduction in the amount of silicon and manganese during melting, the other is the oxidation of the iron. The amount of oxidation depends largely upon conditions of melting.

Dr. Moldenke has pointed out that the method of using large charges in the cupola is the chief cause of this trouble, and by reducing the size of the charges and therefore preventing any great variation in the height of the coke bed above the tuyeres this oxidation is prevented.

A charge of below two hundred pounds per square foot of cross sectional area might be considered a small or suitable

charge and a charge above this amount is called a large charge.

The lower the melting ratio and the lower the density of the coke, the smaller the charge desirable. Therefore, in some cases one hundred pounds per square foot sectional area might be ample.

Other special points discussed in the paper are the total coke ratio, the changes occurring during melting, the removal of the sulphur, and the action of slags in regard to oxide in metal. The author considers the use of limestone in the slag as useful for removing oxygen from the oxidized iron.

As to the limestone requirements of the cupola charge the author considers the fact that in the cupola slags seldom contain more than one part of lime or two parts of silica to be a suitable basis for calculation. This leads to the rule that equal weights of limestone to silica in the components of the charge used will give suitable amount to add, after making allowance for silica from lining.

Generally the amount required will vary between thirty and sixty pounds per 2000 pounds of iron.

Gas Cavities, Shot and Chilled Iron in Iron Castings.

This subject was discussed in a long paper by Mr. THOMAS D. WEST, of Cleveland, Ohio.

The reason of the creation of the small, hard particles or metal shot found in some castings is that they become suddenly separated from the general mass of metal during the pouring of a mold. When thus separated they solidify so quickly that not sufficient time is given for the carbon to be separated out as graphite. Hence the shot does not have the softness of the rest of the casting which had more time to cool.

The longer the distance a shot is sent through the air before it lodges and the cooler the air it passes through, the harder the shot. Affecting this, also, is the degree of dampness of the spot upon which the shot may lodge. Where shot is caught and covered by incoming metal before it has time to become cooled below a red heat the chances for their being hard and discoverable in a casting are not, of course, nearly as great as when they can become of a dark color before being encased by the liquid metal.

Particles of iron or shot may be formed during the falling of the metal from the ladle's pouring lip by striking the sides of a pouring basin or gate. Again, by falling upon a flat bottom inlet gate. Or when the metal, reaching the mold, strikes some obstruction fronting the gates to splatter the metal. In ordinary plain work there is probably no feature so harmful in causing a splattering as having flat-gate bottoms. The bottom of all pouring gates should be more or less of a well, as by this formation, if there is hesitation for a moment or so when starting to pour, the first droppings from the ladle fall into the well, to say there is a liquid bulk and not be spattered to create shot.

In starting to pour the majority of molds, especially those of a light work and stove-plate character, there should be no momentary stoppage for starting, but from the instant the falling metal first strikes the bottom of a gate there should be a steady and unbroken stream to rapidly fill the pouring gate, and have it kept so until the mold is filled. This applies to the flat-top pouring gates generally used for stove plate, etc., as well as those of the joint character.

The too free use of a swab or sponge in wetting the joint of a mold, before drawing a pattern, or after this is done in finishing a mold, can easily harden or chill the metal parts of a casting formed where this excessive dampness existed. Again, also, by too damp a mixture of sand hard ramming or insufficient venting. Metal striking these damp sections will often bubble and splatter to a greater or less degree. This action can in some cases create suddenly chilled shot that will be carried to other sections of a mold to become encased in liquid metal and thereby give the white iron or chilled shot found in some castings.

The author next discusses globules in gas cavities and blow-holes. These defects can be due to two conditions, but emanating from one cause, and that is the creation of excessive gases or steam that cannot find its liberation from imprisonment within a body of solidifying metal. They are found chiefly in the upper body or cast end of castings.

Excessive gas, causing blow holes or cavities, may come directly from the metal or wholly from defects in molding or pouring. Those due to the iron may be caused by oxides of iron or manganese reacting on the iron's carbon, the former producing carbon monoxide gas, or on liberated graphite creating kish (although of the latter there is a very little in remelted iron, it being chiefly found in direct metal at blast furnaces). Also indirectly through sulphur, by reason of the formation of sulphide of manganese.

And again by a foreign body like that possibly created by the mixture of iron oxide and dross (the latter may contain some kish in connection with the dirt generally coming from clays of the cupola tap hole and spout, and other sources) forming a slag on the top of a ladle's metal that could pass in small bodies into a mold at the starting of pouring, or later on before a mold is filled, through defective skimming. Thereby often large as well as small blow holes or gas cavities are formed.

As a side issue it can be said that a ladle's surface dross can easily cause dirt holes, pin holes and a dirty surface to the finish parts as well as the rough surface of castings when skimming is at fault in pouring a casting.

Gas cavities and blow holes due directly to defects in molding and pouring are often due to not having properly tempered sand, regulated degrees of hardness in ramming, efficient means of venting and correct methods of finishing cores and molds, combined with needed experience in grating and pouring. These are all factors demanding the experience of a decade or more in the actual work of general molding.

Molding machines, owing to the following of an exact routine adopted after much experimenting as the best to insure desired results, cannot but be helpful to a large degree in removing the liability of stream and gas cavities, or blow holes, chargeable to molding.

The author then discusses methods of deciding whether globules are due to defects in metal or molding. Sulphur and phosphorus causing blow holes are said to be a weak excuse for defects.

The work of Mr. P. Munnock, of the British Foundrymen's Association, is then discussed. The present author attributes the defects cited by Mr. Munnock to the molds.

The present author's contention that the molds are more responsible as a general thing than the iron for the creation of blow holes, etc., is not to be taken as meaning that there is nothing to be feared by gases that may emanate directly from some irons.

The author briefly discusses the use of alloys for dioxidizing and purifying iron and finally discusses the very rare and mysterious occurrence of "inside chill," or white iron inside of gray or soft iron.

Electricity in the Foundry.

A paper by Mr. BRENT WILEY, of Pittsburgh, Pa., on "Electrical Applications in the Foundry," gives some interesting and valuable figures on the electric power requirements of foundries with a capacity of 1500 to 2000 tons per month. The figures are based on the experience of a number of modern foundries of this size with electric drive throughout, and equipped with overhead traveling cranes in each department. The weight of the castings range from 100 to 100,000 pounds, with an average of 250 pounds. The electrical equipment was 220-volt, direct-current, with one exception, which included 400-volt, three-phase motors. The average conditions for these foundries are as follows:

Floor motors, one 75-hp air-compressor, one 20-hp cold saw, two 30-hp sand mills, eight 5-hp grinders, six machine-shop

motors of together 50 hp, one 25-hp repair-shop motor, six miscellaneous motors of together 58 hp. Total, 323 hp.

Crane motors, two of 50 tons each (280 hp), three of 25 tons each (250 hp), four of 10 tons each (180 hp), two of 5 tons each (90 hp). Total, 800 hp.

Crane hoist motors. Total, 400 hp.

A general rule for power-house capacities is that the average load will be approximately one-half of the total capacities of floor motors and crane hoist motors. In figuring the crane load on the power station it is to be noted that as the hoisting motion of the crane gives the heaviest load, and as it is seldom that more than one motion is used at a time, it is sufficient to consider the load on this motor only. As the crane work is very intermittent the load is very fluctuating, and the total capacity of power generators should not be less than twice the average load. As a general rule a spare generating unit will prove a paying investment to insure ample power at all times.

For the above conditions the average power house load is $723 \div 3 = 241$ hp $= 180\frac{3}{4}$ kw, and two 175-kw generators would be sufficient.

Or, figured on another basis, it requires approximately 35 kw hours per ton of steel castings, and approximately 45 kw hours per ton of iron castings, for foundries ranging in capacity as given above. The blast for the cupolas of the iron foundry and the greater amount of small work increases the electric power per ton output.

The values given for power-house capacities include power for lamps and other auxiliaries, such as electric welding for the steel foundries, repair shop and pattern shop.

The following general conditions for foundries, as mentioned above, have been established. The capacity of the floor or constant-speed motors is approximately equal to the capacity of the crane hoist motors.

The average station load equals one-half the total capacity of floor motors and crane hoist motors.

The maximum station load equals approximately twice the average load.

The average distance for distribution of power is 600 feet or less.

The applications of varying-speed motors are few except where considerable machine work is done, in which case the power required will seldom be over 15 per cent of the total station capacity.

The lighting load will average from 10 to 12 per cent of the total load.

As to the question of direct current versus alternating current the author points out that for the small plant, with a power station capacity of 300 to 500 kw, and an average distributing distance of approximately 600 ft., the 250-volt, direct-current system offers many advantages. Direct-current motors can be used for all the applications.

However, if a foundry has its own power plant the cost of generating power is high on account of the rapidly fluctuating load. It is doubtful if the average steam power plant of the foundry, under the most favorable conditions regarding price of fuel, etc., can produce power for less than 2 cents per kw-hour, and for the majority of the plants the cost will be even higher.

A central station, being of large capacities, would not feel the direct effects of the load fluctuations. The load, being practically continuous, improves the load factor. Thus, to the small plant, especially, the central station companies can often offer good inducements to purchase power.

Lighting is the principal load of the central station, and the system mostly adapted is 60-cycle, alternating current, which offers decided advantages regarding economy of distribution of power. In the case of an existing plant having direct-current motors installed, a rotary converter could be used to convert the power to direct current of suitable voltage.

If the proposition relates to a new plant then a more economical plan will be to use the alternating current direct, although in a majority of the cases it will be necessary to reduce the voltage. Usually the transmission voltage will be over 1000 volts, and transformers should be used to give 240 volts for distribution within the foundry.

Considerable discussion has taken place between the electrical engineers of the iron and steel industry during the last two years regarding the most suitable voltage for inner works distribution, and from a safety standpoint it has been decided that 240 volts for 220-volt, alternating-current motors, and 250 volts for 230-volt, direct-current motors should be used, and these voltages are becoming standard for this class of industry.

The alternating-current, squirrel-cage, wound motor offers several advantages over the direct-current motor for the floor machines, with the exception of the applications requiring varying speed, such as machine tools, and for these applications it would be best to instal a small motor generator set to give direct current.

Lifting Magnets in Foundries.

The application of lifting magnets for foundry service was discussed in a paper by Mr. H. F. Stratton, of Cleveland, Ohio. While the lifting magnet has found very extended application in the steel industry, and is also used largely by the railroads for handling their scrap material, the foundries have been reluctant to introduce it.

However, to prove the adaptability of the lifting magnet for foundry work the author gives the following estimate of cost for a foundry melting 35 tons of metal daily, 300 days in the year. The cost of the magnet and crane installation, with a lifting capacity of 1350 pounds of pig iron, is $4,000. A crane and a magnet of this size can handle 35 tons per hour at an operating cost (wages, fuel, oil) of 48 cents per hour, or 1.4 cents per ton.

The annual depreciation on the $4,000 equipment at 12 per cent would be $480, and as 35 tons handled twice per day for 300 days represent 21,000 tons handled annually the depreciation cost on this tonning basis is 2.3 cents per ton. This brings the total cost of handling, including wages, fuel, oil and depreciation, up to 3.7 cents per ton.

The author is told by a gentleman well versed in foundry practice that 10 cents per ton is a fair figure to assume for the cost of loading or unloading pig by hand, and on this basis the saving would be $1,323 annually, or 33 per cent on an investment of $4,000.

To sum up, a foundry melting 35 tons of metal daily can install both a crane and a magnet, and expect a return upon the investment, after allowing all charges, of more than 30 per cent. If it happens that a foundry is already equipped with either an electric or locomotive crane a magnet can then be installed on a very profitable basis when the tonnage to be handled is considerably less. For instance, if an assumption be made that a foundry melts 20 tons daily, and that a magnet be installed on an existing crane, the cost of the magnet being about $900, then the cost of handling, per ton, including wages, fuel, oil and depreciation on the magnet, is about 2.3 cents, which would represent a saving in this foundry of about $924 each year, or more than 100 per cent on the investment.

"It is very earnestly hoped that these figures be not brushed aside with the assumption that they are theoretical and must be largely discounted. As a matter of fact, the costs of handling by a magnet are stated conservatively, and are being bettered in service daily."

Handling scrap, in general, will be more expensive than handling pig iron, whether it be done by hand or by magnet, but the advantage in favor of the magnet is more marked in the case of scrap than in the case of pig.

Efficiency.

"The Efficiency Movement in the Foundry" was discussed in a paper by Mr. C. E. Knoeppel. He emphasized two conclusions from practice: That a man can accomplish considerably more than he does, and that the management, as business is at

present conducted, does not know what constitutes the best a man can do.

It is necessary to build up an effective organization from the bottom upwards. This requires a constructive type of organization that erects on a firm foundation and in a scientific manner a structure that will consider the various elements entering into making the enterprise successful. These elements are: (1) The man. (2) The work to be done. (3) The conditions under which he does his work. (4) The planning of the work he has to do. (5) Reward for individual effort. (6) The records affecting the factors of the business. These six different items are discussed in succession.

Brass Founders' Association.

Bronzes.

A paper by Mr. JAMES R. HUBER, of the Lumen Bearing Company of Buffalo, N. Y., on the analysis of manganese bronze gave an account of some improvements on the method described in this journal, August, 1910. The complete paper by Mr. Huber will be published in one of our next issues.

A paper by Mr. C. P. KARR gave results of determinations of the pouring temperatures of various bronzes under foundry conditions, and on the use of the pyrometer, especially the radiation pyrometer, in the foundry.

Non-Ferrous Foundry Economics.

A paper by Mr. E. A. BARNES, of the Ft. Wayne Electric Works of Ft. Wayne, Ind., gave interesting and concise notes on refinements in non-ferrous foundry installations and operations. He discussed the design of the building, heating and ventilating systems, office, pattern storage, metal storage, sand bin, crucible storage, core oven, welfare work, power plant arrangements, meters, air compressor radiators, etc.

In the matter of furnaces, ladle heaters, scrap-recovering furnaces, enormous saving can be effected by intelligent handling. With the old system of coke and hard coal pit furnaces, which of course are adhered to for many kinds of production, the greatest injury done to the crucibles is from the rough handling with the tongs and the uneven heating and cooling. With the fuel oil system and tilting furnaces, which are becoming more and more popular, combined with the propagation of suitable reducing flame in a combustion chamber of sufficient volume and absolute control of temperature, this crucible loss is greatly reduced.

One of the greatest items of loss in a brass and aluminum foundry is the shrinkage. This is occasioned in many ways. Poor system is responsible for all of them. Volatilization loss in the methods of melting is the most important.

In their furnace they employ the concentric burner, using compressed air at about 35 pounds per square inch to pulverize to a mist the fuel oil which is supplied at about 25 pounds pressure at the nozzle. This fuel oil and atomizing air are both preheated by proper arrangements before they issue from the burner.

The volume air for combustion which is supplied through a truncated nozzle surrounding the concentric portion is maintained at about 8 ounces pressure and by valves and gates both the compressed air and fuel oil and the volume air are controlled, maintaining the "reducing flame" whether a slow or a fast fire is desired.

It is very necessary to have absolute control of combustion in order to produce the reducing flame, otherwise at these high temperatures the oxygen of the air will combine with the free graphitic carbon and other components of the crucible and furnace lining and very rapidly disintegrate the same through chemical combination, and the metal itself will suffer loss and injury through oxidation.

Ladle Heaters.—Their ladle heaters, which are arranged to use the same burner, are made with a telescoping hood that is raised up when the pouring crucible is introduced and then dropped down enveloping the same. In this way the crucible is evenly heated. These ladle heaters are very useful for getting out small batches of work, as the heat generated in them can be as high as that produced in the larger furnaces.

Conservation of Heat in Spent Gases.—On visiting any foundry one is struck with the enormous amount of apparent lost heat and energy that issues from the vents. This heat, in the form of spent gases, can be utilized for many useful purposes if proper arrangements are made in the design of the foundry to conserve them. Core ovens can be heated, as can also wash water for use in the lavatories; these are two of the most important. The preheating of air and oil is another. This, of course, involves piping and arrangements which may to some appear complicated and unnecessary, but they are nevertheless well worth taking into consideration.

Scrap Melting Furnaces.—In a manufacturing industry like the one with which the author is connected they get skeleton brass from the punch press department, borings and turnings from the machine shop in large quantities each day. The regular practice is to run these through an efficient magnetic separator connected with a blower system which takes out the iron, paper and light particles, passing the good metal into bins. In place of selling this stuff to the junk dealer at prices depending on varying conditions they save and separate these borings, etc., and use them in their foundry at once.

The copper and red brass borings are taken to the melting floor and are used up with gates, sprues and new metal. Much of this stuff, however, is of a mixed nature and cannot be so handled; it is therefore taken to the scrap melting furnace and run into ingots. These ingots are analyzed by the chemist, who determines closely their components. These ingots are further alloyed with pure copper and other new metals in varying proportions and first-class metal for all general purposes produced from them.

They also handle in their scrap melting furnace copper wire which they get from burned-out field coils, armature coils and scrap generally that is known to contain pure copper. The resultant pigs are analyzed and used as base copper for more important mixtures. The furnace is cleaned out, skeleton brass, borings, yellow brass, etc., are run off and stored.

With their scrap melting furnace they developed an endless chain arrangement for carrying the chills. The contents of these chills are automatically dumped into water, which cools, cleanses and softens the metal simultaneously. This system involves the continuous melting and pouring principle which they have found in practice to be preferable to the "hot bath" system where the metal is melted and tapped out into crucibles and repoured. This does not, however, indicate that the "hot bath" system is not better adapted to some other foundry requirements.

As to the actual runs of scrag and the losses incident to the recovery of the same the following figures are given:

Bare copper wire: Total wire melted, 9054 lbs.; total ingot recovered, 8888 lbs.; loss or shrinkage, 166 lbs., or 1.83 per cent.

Fine wire (includes burning off of insulation): Total wire melted, 3089 lbs.; total ingot recovered, 2773 lbs.; loss or shrinkage, 316 lbs., or 10.24 per cent.

Yellow brass punched scrap: Total scrap melted, 3000 lbs.; total ingot recovered, 2860 lbs.; loss or shrinkage, 140 lbs., or 4.5 per cent.

Aluminium.—In their aluminium department for years they had been using the open-flame form of reverberatory furnace and were always confronted with a heavy loss amounting often to 6¾ per cent. Their aluminium work is of a peculiar nature—light and of an odd pattern. For every pound of casting they produced they had to pour one pound for the runners and one pound for the riser or header. They therefore only received 33 1/3 per cent net of the gross metal melted. The loss, therefore, had to be multiplied by three, making 20 per cent net. They figure gross weight, because they charge so much metal to the melting floor, making a bi-monthly inventory of the castings turned over to the factory and the metal in the form of ingots, sprues and risers remaining over.

They could not see any immediate relief from this loss, so carried on experiments from time to time and finally decided that a tilting furnace similar to their regular crucible furnaces, but arranged with an iron kettle of the proper dimensions and shape, was the solution.

With this new type of furnace they are able to handle the same number of pounds of aluminium per heat as with the old open-flame furnace with a loss of only 2½ per cent gross, as it eliminates entirely the losses due to the cutting action of the flame. In the iron kettle the flame does not touch the metal at all and while the time employed in getting out the first heat is necessarily longer, it does not materially affect the total output.

Another great advantage in the kettle type of furnace is that the slag and oxide which form on the top of the pot act as a protection of the metal, and the spatter, skimmings, etc., instead of being thrown into the scrap pile, are returned to the melting pot, where the globules of metal held in suspension in the slag separates from the slag and mix with the metal below, the slag itself remaining on top in the form of a protecting blanket.

In the melting of their metals such as aluminium they employ pyrometers. They have found by using these instruments that the overheating of the metal is avoided.

A great economy results from the pre-alloying of metals. In alloying aluminium and copper it is a well-known fact that the copper, which melts at a much higher temperature than the aluminium, must be melted first. The aluminium is then added, combines with the copper and through chemical action the temperature is liable to rise to a point that vaporizes off much of the aluminium.

To rob this phenomenon of its most harmful effects they have made a practice of compounding a rich alloy of half and half copper and aluminium, which they thrown into pigs. The copper is melted first and the aluminium is added in small quantities, allowing the temperature to gradually reduce with each addition of aluminium, thus preventing overheating any considerable quantity of metal. This rich alloy, which now necessarily melts at a very much lower temperature than the original copper, is used in the final compounding of their aluminium mixture. All risk of overheating or mis-mixing large and individual batches is thus eliminated. Not only this, but the remelting has a refining effect and a much better product is insured.

The use of fluxes, such as phosphor tin, baron, magnesium, etc., is much more effectively carried on under the direction of a chemist who makes a special study of the action of these fluxes on the metals. The author is convinced that the employment of a chemist or foundry chemical engineer in a foundry of any size soon pays a handsome dividend on his sallary. As a rule the practical foundryman has too many other things to attend to in producing to spend much time in developing.

Alloys.

A paper by Dr. W. R. Whitney, director of the Research Laboratory of the General Electric Company, called attention to the possibilities in the way of useful discoveries, which may well lie more nearly within reach of some of the foundrymen than they realize, because of their particular knowledge or possession of special materials. These discoveries should be made by the more extended use of the "more recent metals," particularly in the manufacture of alloys.

Dr. Whitney thinks it highly improbable that the cost of copper will ever be greatly changed by the discovery of new uses. On the other hand, with aluminium it is still possible that considerably wider uses and reduction of production costs may develop in its future.

Dr. Whitney then gave an interesting, concise résumé of those elements which only recently have come into commercial use.

The element *calcium*, which a few years ago was listed only as museum specimens and at several dollars per gram, was sold in 1908 at $1.50 per pound, and could certainly be sold for a small part of this price if a greater use could be found for the metal. It slowly decomposes water, giving hydrogen, and it differs from the alkali metals in producing such a feeble alkaline solution that it is generally harmless. It ought to serve as a good deoxidizer, and should be a very cheap metal. It is not fair to relegate it to a list of useless metals. History of the metallic arts points to there being no such list.

Thallium is an element quite similar to lead, but probably possessing some property which will some day warrant its exploitation. It is softer and heavier, and could be obtained in quantity if a demand were created.

The elements *chromium, molybdenum, tungsten* and *tantalum*, the three latter now obtainable in wire-form, are tempting elements to study in mixtures with others. Who knows the useful properties of a chromium bronze, for example?

Tellurium has long been an apparently useless metal, and any market price is fictitious, as there is but little isolated in metallic state. It is not necessary that a great use, such as a substitute for zinc in brass, should be found for it. Our industries are so great that if a pound of tellurium added to the ton of aluminium was of benefit to the latter the production of the necessary tellurium would be real industry.

Consider *cobalt* a moment. The world's rate of supply of ore has been greatly augmented. It may take time to actually realize a greatly reduced cost of metallic cobalt, but we ought, notwithstanding, to realize it when uses have been developed. Our natural impulse in such a case is to try direct substitution of one metal for another in some well-developed use. Cobalt, for example, might replace nickel in most uses when the cost fell below that of nickel, but this is a second order use. A first order use would be the supplying of a want which no metal previously supplied, or supplied distinctly less perfectly.

In this connection an interesting alloy of cobalt and chromium has just been developed by Elwood Haynes, and it is altogether probable that technical use will soon be made of it. (See our Vol. VIII, p. 652.)

Many tons of metals are annually consumed as resistance wire for electrical purposes. At one time iron was the element most used. German silver replaced it in some cases, where a lower temperature coefficient was needed and the increased cost was permissible. Now there are a dozen or more special alloys for this particular electrical use. The new ones have far outclassed the old in most of those properties for which the electrical engineer uses them. In such alloys nickel, chromium, manganese and others are now used by the ton.

Silicon, which in 1900 was a curiosity and sold for 40 cents per gram, is now a necessary component of special iron alloys and of high-grade transformer iron, and the world uses thousands of tons of the alloy annually. Silicon is now sold at about 5 cents per pound. The use of this metal in other alloys is still quite limited. In the case of iron, it greatly decreases hysteresis loss and increases electrical resistance.

Boron, still a quite expensive material in metallic state, is coming into commercial use in assisting the making of solid copper castings of high electrical conductivity.

Vanadium seems to be a young wonder-working metal. Its use has increased very rapidly in the past few years, but the quantities consumed are not known to the author. As several companies are producing the iron alloy it is safe to assume that it is being sold by the ton. The price for the metal in the alloy is not far from $5 per pound.

Cadmium is a beautiful metal in many respects, and it is certainly awaiting use. It is whiter and less crystalline than zinc, and doubtless the high price of nearly a dollar a pound keeps practical workers from trying it in their experiments. It should be produced as cheaply as aluminium, if there were a good demand for it.

Titanium is an element long the subject of criminal negligence. It is a high-melting, ductile, white metal, which, at present, is only separable from its ores at high cost. It exists in

many cheap ores widely distributed in nature. It is now apparently coming into use in steel manufacture, particularly for railroad rails, and for this purpose it is fortunately unnecessary to isolate the pure titanium from its ores, an iron titanium alloy being produced directly. What will happen when the pure element has been tried in special field can only be surmised. The optimist sees great chances. The pessimist feels himself busy living with the optimist.

If one omits the common alloys, brass, bronze, solder, etc., and considers only possible alloys of two metals, and still confines himself to 20 of the common metals, like vanadium, manganese, chromium, boron, etc., he is interested at once to recognize that there must be one hundred and ninety different pairs of binary alloys. When, in addition, the effect of varying proportions in these alloys is considered, it becomes evident that the field of alloy research is truly a large one. Many of the alloys apparently unstudied are those which melt at extremely high temperatures.

The brass founder who knows the upper limiting temperature of his melting furnaces may at once point out that this temperature is fixed both by the life of crucibles and the particular coke or oil-heating schemes with which he is familiar. If he thought that a molybdenum bronze of 86 per cent molybdenum would have useful properties compared with all other alloys he might at once conclude that he must give up this alloy because of the difficulty of melting it.

If it were not for the advances in our available temperatures there would seem to be little more than amusement in considering alloys high in tantalum, in chromium, in titanium, in molybdenum, in silicon, in uranium, in vanadium and a number of other high-melting metals, but hand in hand with the discoveries leading to isolation of such metals go also discoveries of aluminothermics, oxyacetylene and oxyhydrogen temperatures and electric furnace processes.

The time is always ripe for the study of new alloys with new tensile strengths, elasticities, colors and wearing powers. The automobile and the aeroplane have forced the aluminium and iron alloys to make rapid strides, and it is natural that we should want to inventory our possibilities.

The physical chemist has started along the way of a systematic co-ordination of certain properties of binary, and in a few cases tertiary alloys. He has shown how to plot a few freezing points of two-metal and three-metal mixtures and to construct therefrom curves showing not only all possible metals but what may be expected in the way of segregation and structure and such effects as caused by annealing or quenching.

He has found that there is a solubility of metals in one another which varies just about as the solubility of substances in water varies. Metals may be melted together and well mixed, but the quality and permanency of the mixture is determined by just such solubility laws as control ordinary solutions. We know that in some cases well-mixed melted metals will separate into two layers if allowed to remain even a few moments in molten condition at low temperatures. Such a pair are zinc and lead. They act like a mixture of water and ether. The two separated layers contain both metals, no matter what the temperature, but the quantitative compositions depends on the temperature.

The other extreme of metal solubility is found in such a case as zinc-cadmium, which acts much like a mixture of alcohol and water, the two components going into solution in all proportions and remaining in solution at all temperatures. Having seen this analogy between the facts of solubility of substances in water it is natural to search among the metal mixtures for all the peculiar kinds of solution observed in aqueous solutions.

Two such classes interest us at once. They are those corresponding to aqueous solubilities where temperature widely influences the quantities dissolved, and those in which the solvent (as water) combines with the dissolved substance more intimately than by simple solutions, as by chemical combination. In the case of zinc and lead we have one of the metal alloys of limited solubility. If these two metals are well mixed in liquid state they separate into layers—one, the zinc, carrying a few per cent dissolved lead floats on an alloy made up of lead carrying a few per cent of dissolved zinc. In general, the quantity of the one metal dissolved and held in solution by the other depends on the temperature, and the higher the temperature the greater the solubility. Between 900 and 1,000 deg. C., they are apparently completely soluble. It follows from this that when a dissolved pair of metals is cooled slowly one of them may separate on cooling, if the limiting solubility is reached, and the extent of effective separation may depend on the rate of cooling.

Our second case, that of chemical combination between the metals, is made most evident by the form of the freezing point curve of the possible alloys. A compound of two metals which is stable at a temperature above the melting point of one or both of the metals shows very clearly on the melting-point curve and acts towards each of the elements just as a new or third element. Its melting point cannot be predicted from any knowledge of the component metals. It may even melt higher than either of the components. Such cases are seen in alloys of aluminium-antimony, in lead-tellurium, etc.

Man first used the metals as he found them; then, as he reduced them from the ores, and finally, when specified requirements became more and more exacting, he not only brought into use previously unused metals, but also greatly modified the old familiar ones. For a harder iron he used steel, a carbon alloy; for a harder steel, or one capable of cutting iron more readily, he added tungsten, nickel, chromium and other metals. For permanent magnets molybdenum was added, for high electrical resistance nickel, chromium, etc., were added; for low electrical hysteresis, silicon and aluminium were added; for toughness in springs a little vanadium was used, and for wearing qualities titanium is now introduced. These are only a few of the successful alloying experiments with iron. They will probably be repeated with other metals, such as copper, zinc, and aluminium, where the cost of the base metal is not high.

On the other hand, the study of those metals which have not yet advanced to a stage where first order cost reduction is impossible, is equally interesting. Consider again the element chromium. What do we know about it? Is it a workable metal? Can it be hammered or cast? Is it permanent in the air? Is there a considerable possible ore supply? Has the cost of obtaining the metal been reduced to what seems a reasonable rate? etc., etc.

As it is unlikely that such an element will suggest itself for use by men as did copper and iron, it is probable that its properties must first be determined and made known. As a metal it is only about fifteen years old. It is made in the metallic state by reduction of the oxide by metallic aluminium and also by electrolysis of its salt solutions. It cannot yet be produced at a lower cost than that of the aluminium required, and it now sells at about 80 cents per pound. In the oxide from which it was made it may be had for less than half this cost, and in its alloys with iron, which are made by direct reduction with carbon; it is sold for 29 cents per pound. This gives a rough idea that ultimately, by perfection of metallurgical processes, etc., we may possibly obtain the metal much below 80 cents per pound.

It withstands heat exceedingly well. When pure it melts at very high temperature (Ostwald, about 3000 deg. C.) and it does not scale when heated red hot in air, as copper and iron do. It is for this reason that it is used in resistance alloys for electric heating devices. It has been plated onto metals, and then looks and acts like nickel plate. Doubtless its use will quite rapidly increase in special alloys, as it has already come into use in tool steel.

* * *

In the following we give an incomplete alphabetical list of the exhibitors at the sixth annual exhibit of the Foundry and Mining Exhibition Company held in connection with the Foun-

drymen's Association convention. This exhibition was really an immense affair and all the exhibits were looked over with greatest interest by the visiting foundrymen.

The Adams Company, Dubuque, Iowa.
Albany Sand & Supply Company, Albany, N. Y.
American Vanadium Company, Pittsburgh, Pa.—Vanadium cast steel and cast-iron specimens.
Arcade Manufacturing Company, Freeport, Ill.
Atlas Car & Manufacturing Company, Cleveland, Ohio.—Storage battery locomotive, new type sand-blast car, portable track and side-dump sand car.
Jonathan Bartley Crucible Company, Trenton, N. J.—Crucibles, retorts, stoppers and a variety of graphite specialties.
Carborundum Company, Niagara Falls, N. Y.—Carborundum and aloxite grinding wheels, carborundum fire sand and carborundum rubbing bricks.
Chicago Pneumatic Tool Company, Chicago, Ill.—Franklin tube G C B compound bell, driven air compressors, improved Keller sand rammers, tripod sand sifter, pneumatic geared wire rope hoists, Boyer-Keller chipping hammers, electric grinder, side spindle electric drill, Little Giant pneumatic grinder and castings cleaner, Chicago Universal hose coupling.
The Chisholm & Moore Manufacturing Company, Cleveland, Ohio.—Chain hoists, trolleys, hand-power traveling cranes and malleable castings.
Cutler Hammer Manufacturing Company, Milwaukee, Wis.—Lifting magnet adapted for foundry use.
The Deming Company, Salem, Ohio.—Hand and power pumps, whitewashing machines.
Jos. Dixon Crucible Company, Jersey City, N. J.—Crucibles for both brass and steel melting, special shapes and formulas for various purposes, phosphorizers, stirrers, skimmers, stoppers, nozzles, sleeves, etc.; graphite refractories, foundry facings, motor brushes and graphite products.
Stanley Doggett, New York City, N. Y.—Foundry specialties, parting compounds, manganese dioxide, soapstone, graphite facing, facings, powdered charcoal, dioxide of manganese flux, ferro-manganese, ferro-silicon, iron and steel cement, metal workers' soapstone, crayons and pencils.
Electric Controller & Manufacturing Company, Cleveland, Ohio.—Lifting magnets and controllers, automatic motor starters.
General Electric Company, Schenectady, N. Y.
Goldschmidt Thermit Company, New York City, N. Y.—Full line of metals produced free from carbon, heating thermit cans, titanium thermit cans, thermit welding process and appliances.
Harbison-Walker Refractories Company, Pittsburgh, Pa.—Ordinary shapes and standard cupola blocks in fire brick.
Hauck Manufacturing Company, New York City, N. Y.—Oil-burning appliances, including cupola lighters, ladle heaters, core oven and furnace burners, mold dryers, preheating and brazing outfits.
Hawley Down-Draft Furnace Company, Chicago, Ill.—Metal melting furnaces.
Ideal Furnace Company, Chester, Pa.—Brass melting furnaces.
Ingersoll-Rand Company, New York City, N. Y.—Twelve-inch stroke "NE-1" air compressor, pneumatic chipping, caulking and scaling hammers, pneumatic riveting hammers, sand rammers for bench and floor work, pneumatic motor hoists, pneumatic stationary motors and pneumatic piston and rotary drills.
Monarch Engineering & Manufacturing Company, Baltimore, Md.—Latest improved furnaces using oil or gas as fuel for melting and heating, Acme core oven, steel foundry ladle, heating equipment, aluminum furnaces, Alls' "Eclipse" bolt-heating furnaces, blowers, etc.
Northern Engineering Works, Detroit, Mich.—Type E electric crane trolley and joint exhibit with Cutler Hammer Manufacturing Company, Milwaukee, Wis.
Norton Company, Worcester, Mass.—Grinding wheels, alundum and crystolon, abrasive materials, India oil stones, crystolon sharpening stones, Norton alundum refractories consisting of small electric furnace parts, cores, tubes, muffles, crucibles, combustion boats, filtering dishes, etc., for laboratory use.
Thomas W. Pangborn Company, New York City, N. Y.—Sand-blasting machinery.
Pittsburgh Steel Foundry, Pittsburgh, Pa.
Pittsburgh Valve Foundry & Construction Company, Pittsburgh, Pa.
Rockwell Furnace Company, New York City, N. Y.—Centric pouring crucible furnace.
Rogers, Brown & Company, Cincinnati, Ohio.—Pig iron, coke, ferro manganese "Rosiclare" fluor spar.
Shepard Electric Crane & Hoist Company, Montour Falls, N. Y.—Trolley for three motor electric traveling crane with cage, including the controllers, cage controlled electric traveling hoist and back-geared electric motor.
The W. W. Sly Manufacturing Company, Cleveland, Ohio.—Miniature display of Cleaning mills, cinder mill and dust arrester and sand-blast equipment and machines.
J. D. Smith Foundry Supply Co., Cleveland, Ohio.
Frederick B. Stevens, Detroit, Mich.
Tate, Jones & Company, Inc., Pittsburgh, Pa.—Brass crucible furnace, oil pumping, heating and regulating system, oil burners, portable burner for ladle drying and cupola lighting.
Taylor Instrument Co., Rochester, N. Y.—Recording pyrometer, radiation pyrometer and other small instruments.
United States Graphite Company, Saginaw, Mich.
Westinghouse Electric & Manufacturing Company, Pittsburgh, Pa.—Motors and controllers for direct and alternating current and variable speed work.
Whiting Foundry Equipment Company, Harvey, Ill.—Standard heavy duty exhaust tumbler with steel plate barrel and lever type door fastener, spur-geared crane ladle, with standard gear cover, worm-geared crane ladle, truck for crane ladle, standard turntable.
J. B. Wise, Watertown, N. Y.—M. R. V. brass-melting furnaces, tilting type, permanent crucible using coke as fuel.
Yale & Towne Manufacturing Company, New York City, N. Y.—Hand and electric hoists with trolley running on I-beam.

Resistance Wire.—A recent British patent (14,743, March 30, 1911) of the British Thomson-Houston Company (General Electric Company of this country) refers to an alloy which can be rolled and drawn and which is suitable for resistance wire or heating elements. It consists of twenty parts by weight of iron, sixty-two parts nickel, twelve and one-half parts chromium, five parts manganese, and between 0.1 per cent and 0.4 per cent of carbon. This alloy oxidizes only very slowly at a red heat. (London *Elec. Eng'ing*, April 6.)

Electrodeposition of Copper Free from Arsenic from Electrolytes Containing Arsenic

The effect of organic and inorganic "addition agents" upon the electrodeposition of copper from electrolytes containing arsenic is the title of a thesis submitted by Mr. Ching Yu Wen, to Columbia University, for the degree of Ph.D.

The investigation is interesting in two respects. First, it has a distinctly practical bearing on copper refining, and, secondly, it is of interest in connection with the problems of the action of addition agents in electrolysis which originated on a large scale with Mr. A. G. Betts in his lead-refining process. They have been studied quite extensively, especially by Dr. E. F. Kern, of Columbia University, under whose direction the present research was also conducted.

In the production of good pure copper by electrolysis, the composition of the electrolyte is one of the important factors. The cause of poor copper is chiefly due to the impurities which accumulate in the electrolyte and which, under usual conditions, are precipitated along with the copper. Of those impurities the most harmful and troublesome ones are arsenic and antimony, the presence of which in the deposited copper makes it brittle and nodular.

It has been a known fact that during electrolysis part of the arsenic and antimony contained in the anode dissolves and remains in solution. These two elements, especially the arsenic, are allowed to accumulate in the electrolyte till a critical point is reached, which has not yet been definitely determined. When this point is passed they begin to be deposited with the copper on the cathode and render the deposit bad and brittle.

To prevent this it is therefore of utmost importance to maintain the electrolyte within a certain degree of purity, in other words to keep the amount of arsenic and antimony in the electrolyte below the critical point. This is usually accomplished in practice by withdrawing a certain portion of the electrolyte and replacing it with an equal quantity of fresh solution, and the copper in the impure electrolyte is recovered either by crystallization or by electrolysis with insoluble lead anodes. This not only complicates the process of electrolytic refining of copper, but also entails an extra item of expenditure in the production of electrolytically refined copper.

Another thing that is observed during the electrolysis is that "sprouts" or dendritic "trees" often form, especially along the edges of the cathode. The formation of such "trees" interferes with the work, renders it more difficult to operate and prevents the electrodes from being placed together, as there is danger that the electric current would be short-circuited. In conducting the electrolysis in a commercial way, the removal of these "trees" becomes absolutely necessary and is usually done by the tank inspectors, thus increasing the cost of refining.

The object of Dr. Ching Yu Wen's investigation was, therefore, two-fold; first, to prevent the deposition of arsenic and antimony on the cathode, and second, to prevent the formation of dentritic "trees." This problem was worked out, having in mind the production of solid and smooth deposits, from copper electrolytes containing high percentages of arsenic, by means of organic and inorganic "addition agents."

Dr. Ching Yu Wen's thesis is quite voluminous; first abstracts of former literature relating to the subject are given and then his own experiments are described in great detail. For the description of the apparatus and details of the results the reader must be referred to the original thesis. In the following we give the chief conclusions.

The experiments as performed may be grouped under four main headings: (1) Impure electrolytes, which contained no "addition agents." (2) Impure electrolytes, which contained inorganic "addition agents." (3) Impure electrolytes, which contained organic "addition agents." (4) Impure electrolytes, which contained organic and inorganic "addition agents."

(1) With electrolytes which contained no "addition agent" much arsenic and antimony was deposited with the copper, even

when the electrolytes contain 1.5 per cent arsenic. With electrolytes containing 2 per cent and 3 per cent arsenic a still greater amount of these two impurities were deposited with the copper, and large dentritic "trees" also formed on the surface of the deposits. These impurities rendered the copper dull colored and brittle.

At temperatures between 50° C. and 60° C., and with electrolytes containing over 6 per cent arsenic, the arsenic acid appeared to act as an "addition agent," as it prevented, to some extent, the deposition of the impurities (arsenic and antimony) with the copper and also retarded the formation of dentritic "trees." At temperature of 40° C. and below, however, this action of arsenic as "addition agent" in the electrolyte did not appear to take place in the electrolyte containing under 6 per cent arsenic, whereas in the electrolyte containing 8 per cent arsenic the deposited copper was purer, brighter, more solid and coherent, and less brittle.

This shows that the good effect of arsenic acid depends not only upon the temperature of the electrolyte, but also upon the amount which is present in the electrolyte. This phenomenon cannot be explained from the data obtained in these experiments, but it may be attributed as the result of the hydrolization which converts the arsenic sulphate $As_2(SO_4)_3$ into arsenous acid H_3AsO_3 and sulphuric acid H_2SO_4. The reaction is $As_2(SO_4)_3 + 6H_2O = 2H_3AsO_3 + 3H_2SO_4$.

Temperature and the amount of arsenic in solution both appearing in this case to be functions of the reaction. As the arsenic is in the form of an acid radical, it would not be precipitated on the cathode, but would act as a reducing agent in the electrolyte and thereby give results similar to those of the ordinary "addition agents."

(2). Hydrochloric acid, sodium sulphate, aluminium chloride and sodium chloride, all, when present in small amounts, have a distinct action upon the improvement of the deposited copper both chemically and physically. They cause the deposits to become more or less smooth, dense, pure and free from "trees."

Of these inorganic "addition agents" the best and most effective is sodium chloride, hydrochloric acid to a slight degree. The effect of sodium sulphate is still less; it produces little or no effect if present in too small amounts, especially at temperatures of 40° C., and when the electrolyte contains 3 per cent arsenic.

Aluminium chloride, though it improves the copper deposits to some extent, does not seem to be a suitable "addition agent," while aluminium sulphate is, on the other hand, a satisfactory one, it causing the copper to deposit low in impurities and more ductile.

From the results of the trial of inorganic "addition agents" it appears to be true that the salts of those metals, which stand by far higher in the emf series than copper, are generally satisfactory "addition agents." They possess the property of preventing, to a considerable extent, the deposition of arsenic and antimony and the formation of "trees." The latter action of inorganic "addition agents" is difficult to explain, but, according to Dr. Edward F. Kern,* it is due to the reducing effect of these metals (in ionic state). The higher that the metal of the "addition agent" stands in the emf series the purer, smoother and less brittle the deposit appears.

Temperature plays an important part in the electrolysis of copper. With higher temperature the ductility of the copper seems to be increased and the potential between the electrodes decreases. In the case of sodium sulphate as "addition agent" with higher temperature it becomes more effective.

(3). In the case of organic "addition agents," both gelatine and tannin produce a remarkably beneficial effect upon the copper deposits and are satisfactory "addition agents." The presence of gelatine at the beginning of the electrolysis has the effect of causing the formation of small fern-like "trees," but after operating for a period of time the deposits are smooth and ductile. Peptone, unlike gelatine, exerts a detrimental influence on the deposits and, therefore, is an unsatisfactory "addition agent."

To explain the good and bad action of the organic "addition agents" mentioned above is no easy matter, as their chemical structure, except tannin, is not yet fully and definitely known. It may, however, be of value to cite their properties. Tannin $C_{12}H_3(OH)_3(COOH)_2$ possesses characteristic acid properties and contains three hydroxyls per molecule. Gelatine, whose chemical structure is not yet fully known, is also essentially acid in character, as it possesses, when pure, an acid reaction and dissociates carbonates.

Peptone* is of two kinds, which differ from each other in one molecule of H_2O. Their chemical compositions may be shown as follows: $C_{25}H_{34}N_6O_9$ and $C_{22}H_{36}N_6O_{10}$. The former is called A-peptone and the latter B-peptone. Peptones are pronounced acids, which redden litmus paper and form salts with carbonates after having expelled the carbonic acid gas; adopting the simplest formula, peptones are monobasic acids, but such a simple formula has to be multiplied.

(4). The deposits of copper obtained from electrolytes containing the combined "addition agents" are all satisfactory, particularly when the combination of gelatine and sodium chloride is used. The striking effect of the inorganic substances, when present with the organic, is that it counteracts the bad effect of the latter. Of the two inorganic compounds (sodium chloride and hydrochloric acid) that were used for the combined "addition agents," sodium chloride is in every case better, more satisfactory and more effective in producing good copper.

This shows that the better effect is due to the presence of the sodium ion in the electrolyte. As evidence a particular case may be cited, that is, a case in which peptone and sodium chloride and peptone and hydrochloride acid were used as combined "addition agents." The copper deposits obtained in the former case were found better, both in physical character and purity.

In considering what has been said above, the question may now arise: What causes the organic compounds to behave so differently in the electrolytes when an addition of a small amount of either sodium chloride or hydrochloric acid is made? Does the presence of either of these two inorganic compounds cause the organic ones to undergo a chemical change? These questions cannot be answered without special investigations and were not within the scope of the present research.

With the results that have been obtained, and with the effects of various "addition agents" that have been tried, this may be said: The combined "addition agents" of gelatine and sodium chloride, when present in a small amount (say 0.01 to 0.02 per cent gelatine and 0.01 per cent Cl as NaCl), proves to be the most suitable and most satisfactory "addition agent" for copper sulphate electrolytes containing, especially, a high proportion of arsenic, as the deposited copper possesses the greatest ductility and the highest purity.

Calcium and Molybdenum for Hard Copper Wire.—A recent British patent (7657, April 20, 1911) of the Electrochemische Werke Ges. (Berlin) refers to a process in which up to 1 per cent of calcium or molybdenum or smaller percentages of each are added to electrolytic copper instead of aluminium and magnesium as is now usual to produce hard wires for overhead lines. Whereas a usual requirement for such wires is a conductivity of 95 per cent with a tensile strength of 40 kg to 46 kg per square millimeter, a wire with 0.35 per cent of calcium is stated to give on test 98 per cent conductivity and a tensile strength of 54 kg per square millimeter. Similarly, a wire with 0.05 per cent of molybdenum is stated to give 96.2 per cent conductivity and a tensile strength of 52.9 kg per square millimeter. By adding 0.35 per cent of both calcium and molybdenum to pure copper a conductivity of 98 per cent and a tensile strength of 52 kg per square millimeter are said to be obtained.—London *Elec. Eng'ing*, April 27.

Trans. Amer. Electrochem. Soc., 1909, Vol. 15, p. 473.

**"Chemistry of Proteids," Mann, p. 188.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our London Correspondent.)

The Causes of Corrosion.

The corrosion research committee of the Institute of Metals is making arrangements for conducting an extensive series of investigations into the causes of corrosion in brass condenser tubes, for which purpose a condenser is being made with 48 tubes, 24 of which are of commercially pure brass, while the rest will severally contain one of the following metals: Lead, tin, aluminium, manganese, or any other substance the effects of which the committee may desire to ascertain. The water speeds will be varied in numerical ratios, 2 being the average speed generally employed. The experiments will be continued for several months and will be carried out under the personal supervision of Mr. G. D. Bengough at the University of Liverpool.

An Unusual Cause of Rusting in a Shipyard.

At last month's meeting of the Newcastle Section of the Society of Chemical Industry, Mr. F. C. Garrett said it was generally found that during the building of a ship rusting proceeded only to such extent that when ready for painting the plates could be easily cleaned of scale, but a case had recently occurred where rusting was so extensive that the extra scraping was very expensive and caused some anxiety on the part of the builders lest it should recur after completion.

The ship in question was near the eastern boundary of the yard; and another ship, about a quarter of a mile further to the west, built of the same make of plates, was but slightly more rusty than usual, showing that the fault was not with the plates. The aqueous extract from the rust of the western ship gave 0.342 per cent of a brown substance which slowly absorbed moisture from the air, while the rust from the eastern ship yielded 0.804 per cent of a nearly white and very deliquescent solid.

Analyses of these substances showed for the eastern ship 10.43 per cent calcium, 7.72 magnesium, with 41.6 chlorine, and for the western ship 5.68 per cent calcium, 4.24 magnesium, with 21.78 chlorine. The chlorine calculated for the Ca and Mg amounted respectively to 41.38 and 20.7. The rust from the western ship, therefore, contained 0.1047 per cent of mixed chloride of calcium and magnesium, and that from the eastern ship, 0.478 per cent of the mixed chlorides; and as the rust on the latter was at least twice as much as on the other, it must have contained about 10 times the quantity of chlorides per square foot.

It was found that excavation had been carried on just over the eastern boundary of the yard, involving disturbance of some alkali waste. A sample of this waste could not be obtained, but the local limestones commonly contain much magnesium, and the air was found to contain a chloride—probably hydrogen chloride.

A Ferro-Concrete Pier.

At a meeting of the Concrete Institute Mr. C. P. Taylor mentioned some novel features in the design and construction of the ferro-concrete pier at the Swanscombe works of the Associated Portland Cement Manufacturers.

The foundations consisted of 32 columns, which, except at the corner, were constructed in pairs on 25-ft. centers, the longitudinal pitch being 28 ft. Each pair of columns was connected just above low-water level by a horizontal brace and at the top by a cross-beam. Between the last three pairs of columns at the outer end of the pier there was also a reinforced diagonal strut. The top cross-beams were extended at each end to form a cantilever and the ends of the nine rows of longitudinal beams were built into them.

The nature of the river bed made it desirable to use a construction involving a *minimum* of long piles. The other chief idea underlying the designs adopted was the desirability of carrying the full diameter of the columns right down to the bed of the river without necessitating the deposition of concrete through water. The author considered that the practice of depositing concrete down a shoot inside a shell through a considerable depth of water produced a structure of very questionable strength. Mr. R. G. Clark, therefore, worked out a system of constructing the columns in which the concrete carrying the weight of the structure was in the form of blocks molded on shore and matured before being placed in position. The blocks were laid one on the other surrounding a central pile, and keyed together, and also by rails passing through vertical holes around the pile. The only concrete which had to be deposited under water was that used for filling around the pile and the rails. The central pile alone was not sufficient to carry the total weight borne by each column (about 150 tons) and the base of the column rested on piles driven round the center one and cut off near the ground level.

When the columns were completed the longitudinal beams, all of identical design, were lifted into place by the crane and were temporarily supported on rolled steel joists reaching across the pier on each side of the tops of the column. This plan saved much heavy shuttering which would have been necessary to mold the beams in place and also falsework to support the shuttering. After placing the nine longitudinal beams, the crane and other plant were moved forward for building the next pair of columns, and the molding of the main cross-beam and stiffener was put in hand.

Novelty was not claimed for making the longitudinal beams separately and lifting them into place, although the author did not know of any case where such large beams had been dealt with in this way. The author concluded with a detailed discussion of the especial stresses involved.

Concrete Water-Pipes.

The laying of the reinforced concrete water pipes on the Bonna system for the conveyance of the new water supply to Clydebank has been completed, and the pipes are now taking the water from Burncrooks to the filters at Cochno, distant about 10 miles. There is good authority for stating that the jointing problem has been solved and that no trace of leakage can now be found. The work was begun by contractors, but the corporation had to complete it themselves. The entire cost, including repairs while the work was suspended, is said to be less than that of iron pipes.

Tungsten and Tool Steel.

In an article in the *Times* of April 19 the statement was made that "the first commercial application of tungsten in steel manufacture was by the Bethlehem Steel Company of U. S. A. at the Paris Exhibition in 1900." Mr. Julius L. F. Vogel, a week later, wrote to point out that, although the Taylor-White steel then demonstrated was perhaps the first tool steel of which tungsten was publicly stated to be an important ingredient, Messrs. Osborne, of Sheffield, had very many years earlier incorporated a certain amount of tungsten in what was known as "Mushet" steel, and long before the Paris Exhibition this was employed in a small way in engineering works where specially hard cutting edges were required. Machines in use some 10 years ago were quite incapable of employing such steels to advantage, if at all, as they had not sufficient strength to do the work.

The growth of the high-speed tool was, therefore, dependent upon their replacement by others of special strength and design to work with the heavy cuts and high speeds of which the improved tools were capable. At the present day suitable machines for using high-speed tools are the rule and not the exception; hence the consumption of tungsten has continued to grow steadily.

In view of the fact that the supply of tungsten is comparatively limited it is fairly obvious that competition for the control of tungsten between three rival manufacturing countries such as Great Britain, United States and Germany must inevitably follow. The importance of Great Britain being self-supporting is apparent.

Rapid Method for Estimation of Sulphur in Coal-Gas, or of Ammonium Sulphate.

At the meeting of the Newcastle Section of the Society of Chemical Industry Mr. H. BLAIR read a paper on this subject. Ammonia can be determined rapidly and accurately by utilizing the reaction of Malfatti, viz., the formation of hexamethylenetetramine and liberation of acid on adding formaldehyde to solution of an ammonium compound:

$$4\,NH_4Cl + 6\,CH_2O = N_4(CH_2)_6 + 4\,HCl + 6\,H_2O$$

The liberated acid is titrated by standard alkali with phenolphthalein. For the determination of sulphur in coal-gas the liquor obtained in the "referee's test" is made up to 500 c.c., and 50 c.c. are boiled for a few minutes to expel carbon dioxide; to the hot solution 10 c.c. of formaldehyde solution (about 30 per cent) are added, and the titration is at once made with decinormal potash, 1 c.c. of which is equivalent to 0.0247 grain of sulphur. The potash solution is best standardized by means of a standard solution of ammonium sulphate containing a little ammonium carbonate.

The results come out consistently and a little higher than those obtained by the barium sulphate method, but the discrepancy is not important—figures obtained being 5.064 against 4.53; 10.89 against 10.19; 33.97 against 32.81; and 25.04 against 24.98. The method can be employed for the estimation of sulphur in a smaller volume of gas, such as 1 cu. ft. or 2 cu. ft. The required volume is burned under an adapter and the products of combustion are aspirated through solution of ammonium carbonate which is treated with formaldehyde as above.

For the estimation of ammonium sulphate the salt is dissolved in water and its acidity neutralized with caustic potash; an aliquot part is boiled, formaldehyde is added, and the whole is again boiled for a few seconds, phenolphthalein is added and the titration is made with decinormal potash: 1 c.c. $= 0.0049\ H_2SO_4$; $0.0017\ NH_3$; $0.0066\ Am_2SO_4$. The results agree closely with those obtained by the barium sulphate method.

Volumetric Determination of Tungsten.

E. Knecht and E. Hibbert reduce the tungstic acid or its salts with granulated zinc in a strong hydrochloric acid solution until a brown solution of tetrachloride is produced. Potassium thiocyanate is added as an indicator, and the tetrachloride is reoxidized by standard solution of iron alum in an atmosphere of carbon dioxide. During titration the brown color changes to blue, which persists until the end-point and may serve as an indicator instead of the thiocyanate. The presence of ferrous iron does not affect the titration. In the case of ferro-tungsten the authors recommend decomposition with potassium persulphate—fusing 0.1 gram of the alloy with 4 grams of persulphate in a porcelain crucible, taking up with water, making alkaline with sodium hydroxide and filtering. The tungsten is determined in the filtrate, and the iron on the filter may be dissolved and estimated.

Recovery of Zinc from Ores and Waste Material.

Mr. A. Gutensohn (Eng. Pat. 9818, 1910) mixes the ore, or other material containing zinc, in a powdered state, and after roasting if necessary—with carbon and a collecting material consisting of 20 parts lime, free as possible from silica, with one part each of iron borate and manganese borate. The quantity of carbon employed is the same as the estimated weight of the zinc in the ore, etc., and the collecting material used is one-fifth of that weight. The mixture is heated in a retort at such a temperature that the zinc is the only ingredient melted. For the production of zinc oxide the condenser pipes are fitted with air inlets.

Uranium.

W. P. Jorrissen and A. P. H. Trevelli have found that on exposing commercial metallic uranium to the action of cathode rays nitrogen was given off in considerable quantity. Further research demonstrated the presence of 13 per cent of uranium nitride, U_3N_4, and 1.25 per cent of carbon.

Hydraulic Bronze Castings.

Sergius Kern states that castings suitable for high-pressure hydraulic and steam apparatus can be made by melting 78 parts of best copper with 10 of good brass in a covered crucible holding at least 2 cwt. and then adding successively 12 parts of tin, 1.4 of zinc and 0.12 of cupro-silicon containing 10 per cent of silicon. As soon as diffusion of the components is complete the alloy should be poured into the molds.

Market Prices.

April, 1911.

Copper has remained fairly steady, between the limits of £53.12.0 and £54.15.0 per ton. It opened at £54.11.3, was depressed from 10th to 20th to £53.12.0 or about, then rallied slightly, and is now quoted at £54.

Tin has, over the whole month, risen. It opened at £188.15.0, rose to £192 by the 5th, fell smartly to former price, recovered on 7th, and has since kept fairly steadily on the upgrade though always a nervous market. Closes, £193.15.0.

Lead steady at £13.2.6 to £13.5.0 throughout the month. Opened at £13.5.0, closes at £13.3.9.

Iron Hæmatite opened at 65/-, but continued to fall, reaching 62/6 on the 20th; since recovering to 63/3.

Scotch Pig also inclined to be lower. Opened at 53/7½, and dropped to 52/9 by the 5th, recovering 3d. by the 12th, since steady at 52/7½.

Cleveland Warrants opened at 47/7½, and had dropped to 46/4½ by the 19th, since recovering to 46/6½.

	£	s.	d.
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphite powder, per ton	22	0	0
Borax, British Refined Crystal, per ton	16	0	0
Copper ore, 10 to 25 per cent, per unit	0	9	6
Copper sulphate, per ton	19	15	0
Carbolic acid, liquid, 97/99 per cent, per gal	0	1	5
Creosote, ordinary good liquid, per gal	0	0	2
Camphor, 1-oz. tablets		1	9
Caustic soda, ash, 48 per cent, ordinary, per ton	5	10	0
Litharge, flake, per ton	15	10	0
Mica, small slab, original cases, per lb.		2	0
Naphtha solvent, 90 per cent at 160° C., per gal			9
Petroleum, Russian spot			4⅞
Sal ammoniac, lump, firsts, delivered U.K., per ton	42	0	0
Sulphate of ammonia, f.o.b. Liverpool, per ton	13	15	0
Sulphur, recovered, per ton	5	0	0
Shellac, Standard T. N., orange spots, per cwt	4	4	0
Tin ore, 70 per cent, per ton	£123	to	125 0 0
Hydrochloric acid, per cwt	0	5	0

Platinum, per ounce, has been steady at £8.6.6 throughout the month.

India Rubber, Para fine, has been steadier and lower. Opening at 5/9½ it rose to 6/- by the 5th, and thence dropped to 5/3, and ultimately 5/0½. Now quoted at 5/3½.

On the month the following differences are noticed:

Lower.	*Amount.* £	s.	d.
Copper	0	15	0
Lead		1	3
Hæmatite		1	0
Scotch pig		1	0
Cleveland warrants		1	1
India rubber, Para fine		0	6
Copper sulphate		2	6
Camphor			1
Sulphate of Ammonia		12	6

Higher.	*Amount.* £	s.	d.
Tin	8	2	6
Mica		1	0
Shellac		11	0
Tin ore	11	0	0

Synopsis of Metallurgical and Chemical Literature.

Gold and Silver.

Amalgamation of Gold in Banket Ore.—In the March, 1911, issue of the *Journal* of the Chem. Met. & Mining Soc. of South Africa, W. R. Dowling presents a review of the methods of amalgamation used on the Rand before and since the war. The improvement in methods of metallurgy on the Rand has made it possible to treat lower grades of ore, and the large scale of operations has made it necessary to simplify procedure.

In the early days of the Rand the main source of gold recovery was amalgamation, assisted by concentration and Chlorination. To obtain maximum recovery it was considered necessary to catch the gold as soon after its release as possible, and therefore inside amalgamation was practised. Outside were the usual apron plates with mercury wells between. Mercury also was fed to the mortar. It appears that the percentage recovery in the early days on an oxidized and rich ore by the various methods then in use was not much less than is obtained by the present fine crushing from low-grade pyritic ore by the ordinary straight mill plate and shaking tube mill plates as used to-day.

A most important change was brought about in 1890 by the introduction of the cyanide process. It was at first applied only to coarse leachable ore, and as it was the practice to make as little slime as possible it follows that amalgamation recovery dropped on account of the coarse crushing. When the slime methods came into vogue in 1894 the fine grinding was again possible and the amalgamation recovery assumed its normal condition. The use of lime as a settling factor also was of benefit to amalgamation as it caused the slime to agglomerate

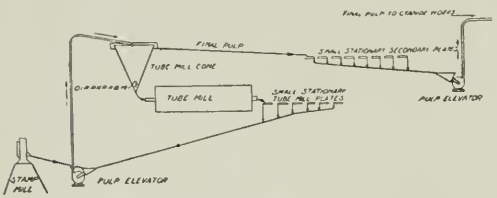


FIG. 1.—RANDFONTEIN CENTRAL ARRANGEMENT. SIX SMALL STATIONARY TUBE MILL PLATES AND EIGHT SMALL STATIONARY SECONDARY PLATES FOR FINAL PULP.

in particles which could come in contact with the plates better than fine disseminated slime. Finally the introduction of tube mills raised the recovery by amalgamation to 65 to 70 per cent of the contents of the ore, of which some 10 to 15 per cent was from the tube mill plates.

Coming nearer to the present time we find that the increased use of tube mills for regrinding permits of coarser crushing at the stamps and consequently the necessity of recovering the plates from the stamps on account of the scouring of the coarse pulp. The author finds that this change gives rise to five possibilities of arrangement of the tubes and plates, and these are illustrated in diagrams, Figs. 1 to 5.

(1) Amalgamation of the tube mill pulp by one set of plates and the overflow final pulp by another set.

(2) Amalgamation of the tube mill pulp only.

(3) Amalgamation of the mixed pulp after regrinding of the coarse portion of tube mills, as in recently erected plants.

(4) Common arrangement, and (5) author's proposal.

It will be noted that plans 1 and 3 involve two elevations, while plan 2 involves only one. In this respect the latter is preferable. Plan 4 also involves but one elevation of pulp, but it also has stamp mill plates, which, the author points out, are better removed, as they make immediately available the gold which otherwise is tied up in them, and permits placing all the plates in a separate house.

The number of plates required after tube mills, and the question as to whether they should shake or be stationary, is referred to by the author. We refer our readers to our May issue, p. 279, for data on this subject.

To ensure successful and economical work the ratio of water to solid should be reduced to the minimum, as not only does this decrease the velocity of the stream over the plate, but the cost for re-elevation is also less. The fall given to stationary plates is 18 per cent, and with this fall it is found that the minimum percentage of water in the pulp is 55 per cent. The

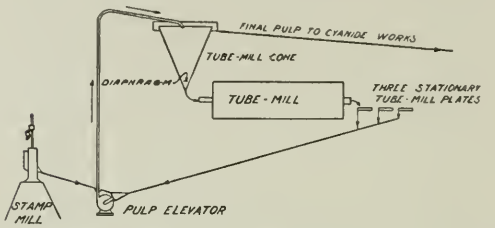


FIG. 2.—SIMMER AND JACK ARRANGEMENT. THREE STATIONARY PLATES IN TUBE MILL CIRCUIT.

point to be reached in water ratio and fall is just short of the banking of sand on the plate.

The author favors the removal of plates from the stamp mill to a separate house, as such an arrangement will result in increased running time by not having to hang up the stamps for cleaning the plates and more attention can be given the crushing apparatus. Better amalgamation also should result by the closer attention made possible, and there would be less opportunity for pilfering gold by employees who necessarily have to work about the stamp mill, but who could be kept from the plate house. Where the plates are removed from the stamp mill the water ratio may be considerably reduced.

The author is of the opinion that the ratio of amalgamation plate surface to stamp unit is entirely too large. At the Randfontein Central the area amounts to 9.6 sq. ft. per stamp unit, but it is now proposed to reduce this to one-half that area. In the Simmer and Jack rearrangement the plate area is about 2 sq. ft. per stamp unit. Advantages resulting from this reduced plate area are reduced cost of installation and operation and reduced cost of mercury consumption. Less gold also will be required to set the plates, and as a well set plate will have about 1 oz. of gold per square foot it will be seen that the sum thus tied up can amount to considerable in a large mill.

Dry Washing for Placer Gold.—The Altar district, So-

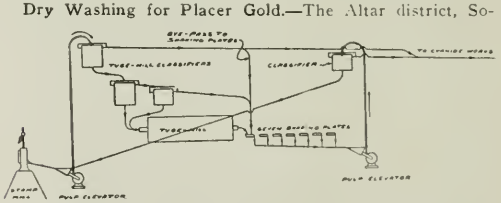


FIG. 3.—RECENT ARRANGEMENT. SEVEN PLATES AMALGAMATING TOTAL MIXED PULP.

nora, Mexico, is for the most part a desert, with but little rainfall and few running streams. On account of this dry condition it is necessary to "dry-wash" the cement gravels containing gold. The operation is described by J. V. Richards, of Spokane, Wash., in the April *Bulletin* of the American Institute of Mining Engineers.

It is said that as long as a century ago the Yaqui Indians discovered rich cement gravel, known as *argo masa*, at Las Palomas, and that there has been a more or less steady pro-

duction of gold from the property ever since. The *argo masa* is a fairly coarse gravel, the pebbles being held firmly together with a lime cement. Gold is scattered through this cement in particles ranging from fine specks to pieces several grams in weight, and occasional large nuggets are encountered. The contents vary so widely that it is difficult to estimate the value of the gravel, but returns of $10 to $50 per cubic yard are common, and it is doubtful if the Indians considered anything less than $4 to $5 per yard as pay dirt. The *argo masa* occurs in channels varying in thickness from a few inches to several

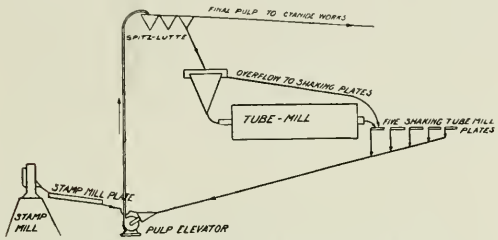


FIG. 4.—COMMON ARRANGEMENT. ONE PLATE FOR FIVE STAMPS AND FIVE SHAKING TUBE MILL PLATES FOR ONE TUBE MILL.

feet, and in width up to 300 ft. These channels are probably ancient stream beds and are overlaid with more recent gravel garying from 30 to 50 ft. in thickness. The overburden contains values as high as $1.50 per yard.

The old Yaqui method of working the ground was to sink shafts and gouge out the cement gravel as far as possible from the bottom of the shaft. Owing to lack of timber for supports, the ground frequently caved, and but little work could be done from any one shaft. After the cement is hoisted, it is pounded between boulders, the coarse gravel removed and the fine cement panned in *bateas*. While this method is laborious, the Indians and Mexicans are expert in the operation, and but little coarse gold is lost. More material can be handled by the dry washer, but the preliminary pulverizing is the same as for the *batea*.

The dry washer is an air blast machine working on the difference in specific gravity of metal and gangue. It is a small table with a cloth top and a bellows below. The table is about 4 ft. long by 2 ft. wide, and has a hopper at one end through which the gravel is fed. The top of the table contains a fine wire netting like ordinary fly screen. On this is placed one and sometimes two layers of burlap, and over this a layer of thin cotton cloth; old flour sacks are sometimes used. Four ½-in. riffles are fastened at equal intervals across the surface, and detachable sides are used to keep the gravel from spilling over. The machine is usually set on the edge of a dump at an angle of 20 deg. from the horizontal, but is carefully leveled in cross-section.

A hand crank belted to a small drive wheel operates the bellows, which has the ordinary clap valves. Intermittent puffs of air come through the cloth at the rate of 150 per minute and cause the sand and gravel to jump the riffles and travel off the table. Most of the gold and black sand is caught behind the first two riffles. The capacity of such a machine is from 2 to 2½ cu. yd. per hour.

After several yards of gravel have been run the machine is stopped and the material behind the riffles brushed into a gold pan. It is then fed back to the machine and nearly all the free gold is caught behind the first riffle. The resulting gold, sand and black sand are then tossed in a *batea* and the gold recovered clean.

The material must be absolutely dry for this operation, only a small amount of dampness tending to clog the cloth and disturb operations. Hand pulverizing has to some extent been displaced by the Quennor disintegrator, which is a cylinder 7 ft. long and 4 ft. in diameter, made up of 1½-in. steel strips set ⅛ in. apart. In the center is a shaft to which are attached chains having tough, manganese-steel lugs at the free ends. The cylinder revolves at the rate of 40 r.p.m. in one direction, and the center shaft revolves in the opposite direction at four or five times this speed. The cement gravel fed in at one end is thoroughly disintegrated by the lugs and falls through the slits, the coarse pebbles being rejected at the end.

At first sight the dry washer appears to be a crude machine, but a test made by the author showed that gravel worth 70 cents per yard gave tailings worth only 10 cents, or a saving of over 85 per cent, and he believes that the general recovery of fairly coarse gold is in excess of 80 per cent.

Zinc.

Modern Zinc Problems.—In his presidential address delivered before the Institution of Mining and Metallurgy, London, Mr. H. LIVINGSTON SULMAN gave an interesting and instructive review of zinc metallurgy and the problems yet to be solved.

In no department of metallurgy has development been more rapid than in the treatment of "complex ores." Just as the terms "free milling" and "refractory" applied to gold ores have lost their older significance, so a few years hence the word "complex" will cease to be used as a prefix to zinc-lead-silver-copper ores, and as a reproach to reduction practice. Many millions of tons of such material have been found, and in some instances partly developed, whose contained metals at current prices render them equivalent in value to two-ounce gold ore. The "complexity" usually is due to blende, usually more or less ferruginous, in intimate physical association with galena and other minerals. Most lead mines are also zinc producers, and every zinc deposit carries lead. When it is possible to dress these ores so as to produce a clean galena concentrate and a zinc portion containing 40 to 45 per cent metal, the ores are not "complex," but the finer parts and slime are still complex in the sense that their separation still awaits solution. The Broken Hill deposits with their intercrystallized galena and ferruginous blende, rhodonite and garnet gangue, which formerly prevented the separation of zinc, are of decreasing complexity since the advent of flotation methods. The slime reserves, however, still present difficulties of treatment.

A more truly complex problem is that which occurs in Tasmania, where the presence of copper adds to the problem. The difficulty in this instance is that the ore contains but little gangue, and concentration by any method is of little or no effect. Succesful reduction must be sought by direct smelting, or the removal of the zinc by leaching processes. Complex carbonates also are known, as those of the Rhodesia Broken

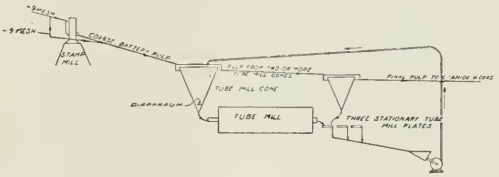


FIG. 5.—PROPOSED ARRANGEMENT. THREE STATIONARY TUBE MILL PLATES.

Hill region, where zinc and lead occur as mixed carbonates. Again, there are those ores in which the presence of fluorspar prevents the usual retort reduction methods for zinc, and which probably will have to be reduced in the electric furnace.

The methods of attacking such ores may be divided into proximate or mechanical, and ultimate or leaching and smelting processes. Proximate methods are limited by the conditions encountered in crushing heterogeneous materials to free the component parts, and usually are inefficient by reason of having to produce several grades of mineral, or a low grade

if all be mixed. Among the proximate methods which have come into prominence is the flotation group.

The simplest type of flotation methods is represented in the DeBavay process used at Broken Hill, and in the Macquisten tube, reference to which was made in our issue for May, 1911, p. 236. In these the surface tension of water is used to float off the sulphide particles, which are not easily wetted, whereas the more readily wetted gangue particles sink when a mixture of mineral and gangue is presented to a surface of flowing water. A second variety is the Potter process, in which submerged sulphide particles are caused to rise to the surface of a hot acidified pulp by the fixation of gas bubbles; the gas in this instance being carbon dioxide generated by the acid on siderite. The third variation includes the well known oil processes of Elmore and Minerals Separation, Ltd., which secure the flotation of submerged minerals by means of bubbles of air, such mineral particles having previously received a slight film covering of oil for the purpose of effecting the air or gas attachment. To Elmore undoubtedly belongs the honor of first bringing oil-mineral processes into the realm of commercial practicability. To Froment and especially to Cattermole we are indebted for the next important development. The latter first demonstrates the efficiency of a very small quantity of oil in place of the large bulk hitherto employed.

The physical reactions which underlie all such methods have been insufficiently explored, and in many respects are still obscure; the striking practical results so far achieved have been reached quite empirically. The author mentions a new fact in relation to the measurements of the "contact angles" which exist between the surfaces of aqueous liquids and those of certain sulphides, silicates and other solids. In investigating previously accepted figures for these, he finds discrepancies in the readings, which on further investigation prove to be due to the existence of a variable range for the contact angle between various liquids and solids, though of constant magnitude of variation for each. This range of angular variation has been called by C. G. Lamb, "the angle of hysteresis," and its existence and magnitude in regard to many substances have been independently confirmed and quantified by A. Howard Higgins. It is not a fact of merely academic interest, as it appears to be intimately connected with the ability of a given solid to condense upon itself a gas film; and when submerged in a liquid to determine the attachment of a gas when generated, or on the point of generation, from a solution. It follows that angular hysteresis reaches its highest values for minerals which are most susceptible of flotation, and *vice versa*. Hundreds of observations have been made which place this relationship beyond doubt. Besides roughly quantifying the gas condensing power due to the surface energy of solids, it brings us somewhat closer to an explanation of the efficiency of acidification. Thus, whereas the angular hysteresis of silica in plain water may exceed 30 deg., thus indicating that substance to have a definite power to occlude gas and to float, it drops from 4 deg. to 0 deg. in water acidified with sulphuric acid. Galena, on the other hand, retains its full measure of angular variation or is but slightly affected.

While certain difficulties remain to be solved, especially in regard to blende and galena slime, the Broken Hill problem may be regarded as solved by the flotation processes. It must be remembered however, that the methods referred to work effectively only on particles of sensible size.

The slime problem has been studied by Horwood, who operates by superficially attacking one particle while leaving the other unaltered, and thereafter submits the product to oil flotation. He usually employs heat to effect this differentiation, a low temperature oxidizing the surface of the galena while the blende remains unaltred. The Minerals Separation process is preferred for this work as it deals better with slime.

Magnetic separators employing fields of high intensity still have a wide range of usefulnes, although on dry ore have given trouble on account of dust. Wet types have been extensively experimented with, and lately Mr. Ullrich has introduced a multipolar ring machine which overcomes the difficulty of removing small particles from a liquid. Several of these machines are already working, and their possibilies regarding slime mineral are receiving close attention.

Electrostatic methods are revived in the Huff machine. To L. I. Blake is due the credit for first introducing a commercial separator on the electrostatic principle. He was followed by Sutton and Steele, in whose machine the electrode was charged inductively by a brush discharge, and was furnished with an ingenious interrupter whereby any desired degree of saturation of the separating roller could be obtained to suit the varying electrical capacities of the minerals to be separated. An electrostatic generator also was employed in this machine. The main advance in the Huff machine is the substitution of a dynamo current of suitable intensity for the electrostatic energy of a Wimhurst machine. The limiting factor of electrostatic separators is their inability to handle fine powders, and their liability to dust troubles unless the dust first be removed.

In the mechanical treatment of slime we reach the limit of proximate methods of handling zinc ores, and must take up ultimate methods such as smelting and hydrometallurgical processes. Dealing with smelting, the author mentions H. Pape's process in Germany. Low grade zinc-lead ores are smelted with sufficient silicious ore to slag off the zinc and reduce the lead. The slag of zinc silicate is then crushed and briquetted with sufficient carbon and a binding agent, and charged into a blowing furnace, fitted with a special grate on which the briquets are charged to a depth of three or four feet. Upon ignition, the zinc is first reduced and then blown off as oxide, while the depleted slag flows through the grate. The separation goes on alomst to completeness, and the zinc oxide is collected in chambers and bags.

Referring to leaching methods, the bisulphite process is described as follows: Roasted ore is subjected to the action of water and dissolved sulphur dioxide gas, the latter being derived from the roasting operation. The zinc is dissolved as bisulphite and separated from the residues which contain lead, silver and other metals. From the solution the zinc is thrown down as insoluble monosulhphite by removing one of the equivalents of sulphurous acid gas. The precipitate is then calcined, resulting in the production of a dense zinc oxide suitable for reduction to spelter. Sulphate of zinc accumulates gradually in the liquors until they are strong enough for recovery of the salt by crystallization.

A means for the reduction of zinc sulphate is provided by electrolysis, the deposited zinc being stripped from the cathode and remelted to spelter, while the acid regenerated at the anode is used for the leaching of fresh oxidized zinc from roasted ore. In practice it is found that about 5,500 electrical horsepower, as a mean figure, is required to produce a ton of deposited zinc, or roughly two-thirds of an electrical horsepower-year. Wherefore, localities favored with cheap water power will hold an advantage in the production of electrolytic zinc, for the process could hardly supersede the present distillation processes if located at existing reduction centers.

The author concludes with a review of electric processes for zinc, but as these have been given full attention in our issue for May, 1911 (see also our Vol. VIII., pages 49, 99, 289, 317, 688, 689), they will not be mentioned here.

Physical Chemistry and Electrochemistry.

Specific Heats of the Elements.—In a recent number of the *Zeit. f. Elektrochemie* J. Koenigsberger publishes a compilation of values obtained by himself, Nernst, Richards, Dewar and others for the atomic specific heats of the elements. It is shown that there is a distinct difference between the specific heat of a good conductor and that of an insulator. The values for metals cooled to very low temperatures are very near those

for insulators at high temperatures. The value c_p for carbon, in the shape of diamond, is almost zero at $-180°$ and at 1500° it has increased to $c_p = 6$.

Colloids.—*In the Zeit. f. Elektrochemie,* April 15, A. LOTTERMOSER gives a very careful review of all important papers and patents on colloids that appeared in 1908. Among the headings under which the subject matter is treated are: Electro-osmosis, the theory of dyeing, floculation, production of colloidal metals and metalloids on a commercial scale, lubricants and albumen.

Thermodynamics of Galvanic Cells.—In the same number R. LUTHER publishes a note on *Lead-Mercury Cells.* He finds that according to electromotive force determinations the heat of reaction of $Pb + 2HgCl = PbCl_2 + 2Hg$ is equal to 21,900 calories; from thermodynamical data of Nernst and Thomson the value 20,100 is derived. This leaves a difference of 18000 calories between the two values. No satisfactory explanation of this discrepancy has as yet been found.

Sharpening of Tools by Electrolytic Etching.—In *Dingler's Polytechn. Journal,* Vol. 35, 1910, page 747 (abstracted in *Elek. Zeit.,* Jan. 26, page 93), E. SCHNECKENBERG discusses the use of the electric current for the sharpening of tools by electrolytic action. Barthel formerly used a mixture of three parts of 66 per cent sulphuric acid and 6 parts of 40 per cent hydrochloric acid. The file or cutter to be sharpened was first cleaned from fat and dirt and then suspended together with a carbon electrode into the above solution. This represented a short-circuited cell and the file or cutter was automatically sharpened. Gratwohl has recently modified the method by employing an inert load electrode instead of the carbon electrode and a primary battery or storage battery giving 1 volt or 2 volts as an external source of current. He uses an electrolyte of a sulphuric acid solution of specific gravity 1.18.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Briquets of fine iron ore may be made according to the formula of GEORGE K. HOLLISTER, JR., of New York City. He proceeds preferably as follows: Say 2,000 pounds of fine iron ore are mixed with 15 pounds of clay in a dry and pulverized state. To this is added, during a mixing operation, 3½ gallons of molasses slightly acidified with two ounces of hydrochloric acid, and three gallons of water containing two pounds of sal soda and one pound of powdered alum. When the ingredients are thoroughly mixed the mass is placed in a pug mill and heated evenly to a temperature of about 350° F. until the material has lost its wet appearance and feels dry to the touch. It is then passed through a briqueting machine. By this procedure the inventor claims to obtain a hard, practically unbreakable briquet, due to the bond formed by the salts and clay, and to the carbonization of the molasses. It is claimed that the briquets can be used in the smelting furnace immediately after leaving the machine. (990,049, April 18, 1911.)

Copper.

Improving Conductivity and Tensile Strength.—The improvement of copper and its alloys for electrical work by increasing the tensile strength and conductivity is the basis of a patent granted to RUDOLF FRANKE, of Eisleben, Germany. The process has been assigned to the Elektrochemische Werke (G. M. B. H.), of Berlin, Germany.

Three classes of wire are used in telegraphic and telephonic work in Germany, viz., chemically pure copper wires, bronze wires and double bronze wires. The bronze and double bronze wires are treated with a composition consisting of 64 per cent aluminium and 36 per cent magnesium, together with a little tin. The inventor has experimented to see whether the composition could not be replaced by calcium or by molybdenum. He produced both soft and hard-drawn wires as follows: 1. Wires without any addition. 2. Wires with the addition of calcium in the form of calcium-copper containing 12.2 per cent calcium. 3. Wires with the addition of molybdenum in the form of metallic powder containing 97.1 per cent molybdenum and 3 to 4 per cent oxide, 1.4 per cent insoluble matter, 0.65 per cent iron and aluminium as carbide. 4. Wires with the addition of aluminium and calcium in the proportion of 42.3 per cent calcium and 55.85 per cent aluminium. 5. Wires with the addition of molybdenum and calcium, the latter in the form of calcium-copper. 6. Wires with the addition of aluminium-molybdenum-calcium, the calcium in the form of aluminium-calcium and the molybdenum as metallic powder.

The results of the tests showed that the addition of calcium as calcium-copper was far superior to the magnesium-aluminium composition; that molybdenum similarly gives better results than the composition referred to and about equal to those of calcium; that the addition of aluminium-calcium gives poor results; that calcium-molybdenum additions give the good results shown by the addition of either metal; and finally, the aluminium-molybdenum-calcium addition gives poor results and should not be used on account of the presence of the aluminium.

The tests establish to the satisfaction of the inventor that (1) calcium and molybdenum addition can be advantageously substituted for the aluminium-magnesium composition for hard-drawn bronze wires, and that (2) with an addition of at the most 0.35 per cent of calcium or molybdenum or both, hard-drawn wires can be produced with a tensile strength of 50 to 52 kg. per sq. mm., and a conductivity of 95 per cent. The patent specifications give the engineering results on many tests which are instructive and bear out the inventor's claims. (990,040, April 18, 1911.)

Aluminium.

A new solder formula recently has been patented by JOSEPH N. DAUDELIN, of St. Hyacinthe, Quebec, Canada. It comprises a mixture of silver, phosphorus, copper, lead, tin, aluminium and zinc. The main object of the invention is to produce a solder having great tensile strength; one which will run easily; which may be used for aluminium as well as other metals; one which has a very low coefficient of expansion; one which is homogeneous, will not easily corrode and will be tough without being brittle. The various ingredients are claimed to give these properties to the solder. The inventor prefers to mix the metals in the following proportions: 22 grains of silver, 42 grains of phosphorus, 238 grains of copper, 1338 grains of lead, 5,051 grains of tin, 50 grains of aluminium, 939 grains of zinc. (989,573, April 18, 1911.)

Electric Furnaces.

Pinch Effect for Electric Furnace.—In his recent American Electrochemical Society paper (our May issue, page 277), Mr. CARL HERING, of Philadelphia, Pa., described in detail an electric furnace in which he makes use of the "pinch effect" for circulation of the charge. A patent has now been granted to him on this process. Since the principles were already described quite fully in our last issue, we give here only two of the 34 claims. The first claim reads as follows: "In an electric furnace, a hearth for containing a mass of molten material, columns or channels communicating with said hearth and adapted to be filled with molten material in communication with said molten mass to constitute the furnace resistor, and electrodes in end on communication with said columns or channels, the square of the current transmitted through said electrodes to the molten material in said columns or channels with relation to the cross-section of said columns or channels being great, whereby said mass of molten material is automatically stirred." The twelfth claim reads as follows: "In an electric furnace, the combination with a mass of molten material, of a molten mass in communication therewith serving as resistor, the ratio of the square of the current through said resistor to its cross-section being great, whereby circu-

lation of molten metal is produced, and an electrode in communication with said resistor having such dimensions that the C^2R heat developed in said electrode by the current traversing said resistor shall be equal to substantially twice the heat conduction loss through said electrode when no current is flowing." In the specification the laws of the pinch effect and of the heat losses through electrodes are given and rules are derived for calculating the dimensions of the essential parts of the furnace. Various modifications of the simplest arrangement, employing two vertical columns to communicate with the hearth, are described. By arranging the resistor columns not vertically but inclined to the vertical, less hydrostatic pressure is developed in the columns. With three columns the furnace is adapted to three-phase currents. A double furnace is described for the production of steel directly from ore; in one of the furnace chambers the iron ore is reduced, in the other chamber the refining of the steel is performed, both chambers being heated from the bottom, each by a set of resistor columns. Combinations of this resistance furnace with the use of arcs at the top are also described. (988,936, April 4, 1911.)

Electrode Arrangement for the Reduction of Radiation Loss.—Mr. E. C. Speidex, of the International Acheson Graphite Co., of Niagara Falls, has patented an arrangement of the electrodes above the bath for the purpose of reducing the direct radiation of heat from the surface of the molten bath. The electrodes are disposed in overlapping series, whether in parallel planes or in angular relation to each other, but in such a way as to protect, shield, or shadow the upper portion of the furnace structure, including the roof, from the effect of heat radiated from the zone of maximum temperature. (992,281, May 16, 1911.)

Combination Bessemer-Electric Steel Process.—A new combination process for refining Bessemer metal in the electric furnace has been patented by Mr. James H. Gray and assigned to the Illinois Steel Co. By the ordinary acid Bessemer process silicon and carbon are removed from molten pig iron; the metal is then dephosphorized in a ladle by means of a slag of lime and iron oxide (previously prepared in a cupola furnace); the metal is finally treated in an electric furnace for the removal of sulphur and oxygen. (987,715, March 28, 1911.)

Electrolytic Copper Refining.

Purification of Electrolyte.—In copper refining the solution becomes gradually impure and it is necessary to remove large quantities of the same periodically. The common practice is to crystallize out the bulk of the copper as blue vitriol (sometimes with the previous step of neutralizing the free acid of the electrolyte by passing over shot copper), and then precipitate the remaining copper on scrap iron. The residue is generally discarded. Messrs. C. H. Aldrich and J. K. Bryan, of Perth Amboy, N. J., propose to subject this residue to further purification. They concentrate the impure electrolyte (18 to 25° Be.) to 45° Be. in a lead-lined tank, heated by means of lead coils and concentrate it further in a lead-lined iron pan to 50° Be. The anhydrous salts are allowed to crystallize out until the bulk of the copper is removed. Thus far the process is the one usually pursued. But now instead of precipitating the copper, nickel and iron salts in one pan, the inventors let the mother liquor pass through a number of heated iron pans arranged in cascade. The mother liquor is thereby concentrated to 60 or 70° Be. and the salts are fractionally precipitated in the cascade of pans. The precipitate in the first pans is relatively rich in copper, that in the intermediate pans contains copper and nickel in about equal quantities, and that in the last pans is rich in nickel.

The resulting solution is allowed to settle in a tank from which it is then pumped into refining tanks arranged in cascade. This solution contains not only traces of copper, iron, nickel and zinc, but contains a material amount of arsenic and antimony in solution. In the electrolytic tanks a temperature of 170° to 180° F. is maintained, insoluble anodes are used and cathodes of lead, copper or iron. The electrolysis is continued until the bulk of the arsenic, together with traces of the copper and antimony, have been deposited on the cathodes. The arsenic deposit contains copper, antimony, etc., from which it can be easily separated and recovered by burning in a small furnace with suitable condensing flues. The residue is composed principally of copper oxide which is worked up in a blast furnace in the usual way. The purified liquor is allowed to settle and then returned to the copper refining tank. (991,685, May 9, 1911.)

Petroleum.

Separation of Oil and Water.—Four patents recently issued to Mr. Frederick G. Cottrell and associates, of Berkeley, Cal., are interesting for their application of the electric current to the separation of water from oil.

Crude petroleum as it comes from the wells is frequently accompanied by more or less water mixed with the oil in drops of macroscopic and microscopic size. The larger drops might settle in a natural manner, but the small ones will not so separate from the oil. Centrifugal action is wholly satisfactory and distillation is expensive. The inventors have conceived the idea of applying a high-potential alternating current to the mixture, causing the water particles to coalesce into large enough drops to settle out of the oil.

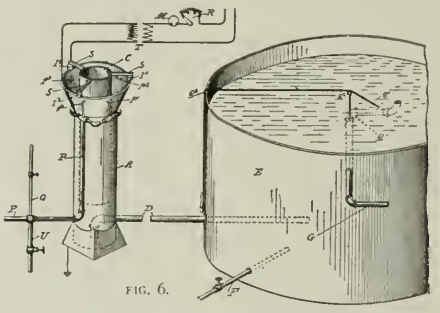


FIG. 6.

Fig. 6 gives a complete view of a diagrammatic apparatus for carrying out the invention. The vessel A contains the oil undergoing treatment and is itself one of the electrodes, being connected with one of the high-potential terminals of the transformer T. The other terminal is connected to the electrode C which is represented as an inner concentric shell in the vessel A, although its exact form is a subject of another patent. The two electrodes are insulated as shown in the diagram at I^1, I^2, I^3.

The oil and water mixture to be treated is received through the pipe P, and may be heated by steam delivered from Q, and diluted if necessary by dry oil through U. The mixture enters the top of the vessel A through jets p^1, p^2, p^3, p^4, and flows down between the electrodes, and finally out through a pipe D, which is of sufficient diameter to insure a quiet flow, free from a churning effect which would again emulsify the separating oil and water. On entering the tank E, the water, which has been caused to gather into drops of large enough size to settle naturally, drops to the bottom. Dry oil can then be drawn off through G and water and sludge is drawn off through F.

Care must be taken to have the active portion of the electrodes deep in the mixture to avoid the possibility of the separated water forming surface layers and causing short-circuiting. The heating of the oil prior to its treatment improves its physical condition and in the present instance this can best be done by steam, since there is no objection to the introduction of water in the oil. Many other details of the operation and construction of the parts will be apparent on a study of the original patents. (987,114-5-6-7, March 21, 1911.)

Automatic Gas Analysis for Boiler House and Chemical Works Control.

The following is a general description of the automatic gas analyzers made by the Precision Instrument Company, of Detroit, Mich., under license from Alexander Wright & Company, Ltd., engineers, of Westminster, England, and their two English directors, Messrs. John F. Simmance, M. I. M. E., and Jacques Abady, A. S. M. E.

These recording analyzers may be divided into two classes: Those for non-corrosive and those for corrosive gases. In the first class is the precision Simmance-Abady CO_2 recorder, so-called because it periodically analyzes samples of flue gas for the per cent of carbon dioxide which they contain and records

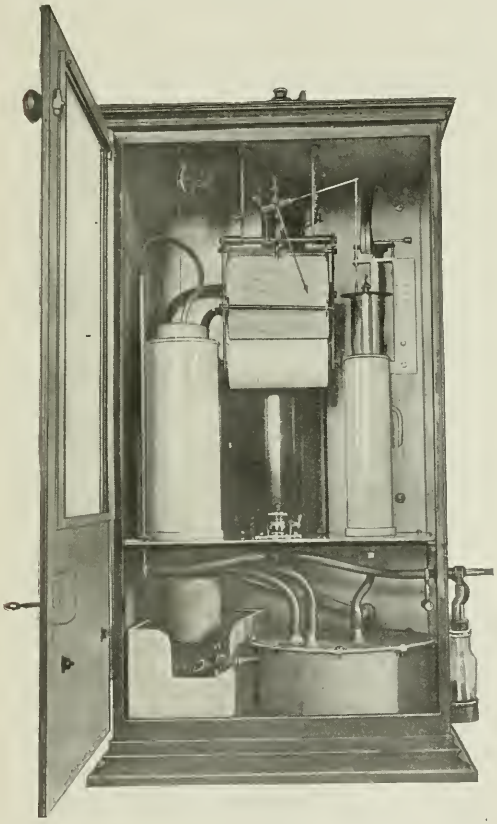


FIG. 1.—HIGH PERCENTAGE RECORDER FOR CHEMICAL WORKS.

this percentage on a chart, showing graphically and at all times of the day or night the completeness or incompleteness of combustion. The boiler house CO_2 recorder is perhaps the most thoroughly discussed of all automatic analyzers, for the reason that it is an established fact that a high percentage of CO_2 in the flue gas is one of the principal factors in economical combustion.

As the theoretical maximum of CO_2 in burned fuel gases is about 20 per cent, the charts for the five standard types of boiler-house recorders are graduated from zero to 20 per cent. Charts may be disc, drum or tape charts. These standard types may be used on furnaces fired with natural gas; they are also used for recording the percentage of CO_2 in producer gas, CWG, purified gas and gases in retort settings. Where the gas is under pressure a governor is fitted at the inlet, and a burner is also supplied for combustible gases.

Recorders are also built for ranges other than 20 per cent, as, for instance, 0 to 35 per cent, and 0 to 50 per cent for lime kiln control in beet sugar factories, etc.; also such high ranges as 0 to 100 per cent, 65 to 100 per cent and 80 to 100 per cent for special chemical machinery. The latter need a strong solution and require about ten minutes for a complete absorption of the CO_2 by the KOH, as compared with about three or four minutes with a weaker solution in the low per cent recorders. Furthermore, it is occasionally necessary to draw the gas sample against a high draught, and one recorder is now in operation in a large manufacturing plant where it is drawn against a maximum of 18½ in. water vacuum. A high percentage recorder for chemical works is shown in Fig. 1.

In the second class are the recording analyzers for corrosive gases, such as SO_2, HCl and chlorine gas, and which are used in paper mills, acid works, etc. They are built to record various per cents as are the CO_2 recorders, and operate on the same

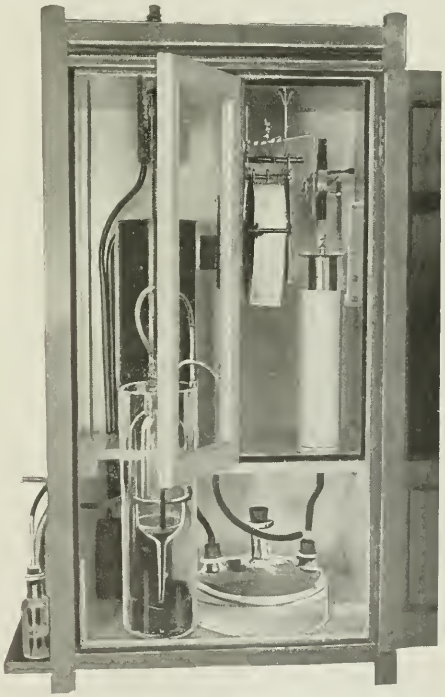


FIG. 2.—RECORDER FOR CORROSIVE GASES, SUCH AS SO_2, HCL, CHLORINE, ETC.

general principle. The main difference in the construction is that the parts which handle the gases, including extractor bell and tank and solution vessel, are made, not of metal, but of glass, to withstand the corrosive action of the gas. Also on account of the fumes from the gas the indicating and recording mechanism is contained in an air-tight compartment. A view of the recorder for corrosive gases is shown in Fig. 2.

Generally speaking, the recorder may be called an automatic chemist. Samples of gas are drawn into a waterjacketed burette called the extractor bell *d*, Fig. 3, where the sample is cut off and the gas reduced to atmospheric pressure. The recorder is thus independent of varying draft conditions. The bell is water sealed and open at the bottom, and gas is drawn

in by means of the vacuum created in the bell when the latter is raised. After the complete sample is drawn into the bell, cock *h* automatically closes and the bell starts to sink; pressure is built up in the bell, which forces the gas through the caustic solution in tank *m*, where the carbon dioxide is absorbed. The balance of the gas passes into the recorder bell *j*, which rises to a point on the scale indicating the volume of gas that has been absorbed. A pen then marks on the chart the same per cent of absorption as is indicated on the scale, after which the bell vents through cock *h*.

The motive power which operates the extractor bell is a float *b* rising and falling in the siphon tank *a*. A constant stream of water flows through cock *x* at the top of the recorder and into tank *a*. When float *b* rises to the top of the tank, valve *e* is tripped, allowing to come into the tank a flush of water which immediately siphons out of the tank through *g*, allowing float to sink and raising extractor bell.

This same stream of water which is the motive power actuates an injector below the inlet cock, and this injector brings

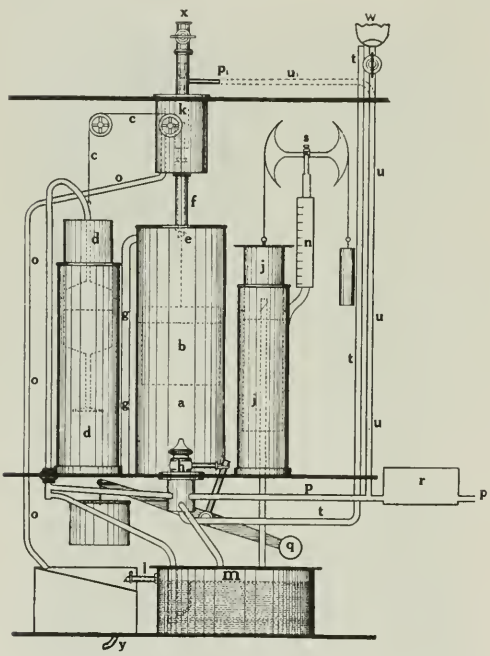


FIG. 3.—DIAGRAM OF CO_2 RECORDER.

along a constant stream of gas through auxiliary connection p_1 so that there is no dead gas in the sampling pipe.

In boiler-house service there is practically no "lag," the operations of the recorder being about three minutes behind the actual operations in the furnace. The cycle of operations is completed in about three minutes in the boiler-house recorder, whereas for gas containing higher percentages of CO_2 the cycle may take as much as ten minutes, owing to the longer time taken for absorption.

For recorders handling non-corrosive gases, such as flue gas, all the working parts are made of metal and are enclosed in a metallic case, so that it is suitable for use in exposed situations.

To the gas inlet nozzle are attached two tell-tale bottles, one of which shows that the stream of gas is flowing constantly, and the other indicates when the sampling pipe is plugged with soot or other material. When used on a battery of boilers a pair of oil cups is located at each boiler, these oil cups taking the place of cocks or valves; to take records from any particular boiler in the battery a dust filter is shifted along by hand from one boiler to the next.

The recorders have been frequently tested by the best authorities, including chemists of some of the largest plants, and invariably check up to within one-half of 1 per cent or less of the most exhaustive hand analyses, and this under varying conditions of draft.

A New Electrolytic Cell for Hypochlorite Solutions.

By R. H. Pearsall.

Chlorine is to-day used in the large majority of bleaching processes and is also coming more and more into use for sterilizing water. Its value for bleaching is well known, and it may be pointed out here that it is so powerful a sterilizing agent that as small a quantity as one part in one million is enough to entirely destroy the bacilli of typhoid and diphtheria in one minute, even in bad water.

Chlorine is generally used in the form of chloride of lime, or bleaching powder, as this is cheap and portable. But this has the disadvantage that it leaves a residue of lime which is a cause of decay in fibrous materials which have been bleached. The powder is highly poisonous and, moreover, does not retain its valuable properties long after exposure to the air with the consequence that where smaller quantities are needed, as for water sterilization, it is difficult to maintain stocks of powder in good condition and at the highest efficiency.

This involves larger doses for sterilization purposes in order to be certain of results and irregularities in the quality of the powder result in the water occasionally tasting or smelling of chlorine.

An improved method which entirely avoids this is to use a solution of hypochlorite which as made to-day will retain practically its full strength for long periods, and is much more powerful in its action than bleaching powder. It has the further advantage that it forms no deposit in the pipes, valves, etc., as does the bleaching powder solution, so that much greater regularity and reliability of action is secured.

As it is most economically made in weak solutions containing from ½ per cent to 2 per cent of available chlorine, it is preferable to make the solution on the spot as it is required, a process which has become very simple and economical by virtue of recent improvements in electrolysis which are described in this article.

We may describe the principles of the formation of hypochlorites by electrolytic means. If a current is sent through a solution of sodium chloride in water the chloride is split up into the component parts sodium and chlorine. The chlorine is liberated at the anode and the sodium at the cathode, forming by reaction with the water sodium hydroxide and hydrogen.

In the manufacture of chloride of lime and caustic soda the products at the electrodes are kept separate, the chlorine being drawn off to lime beds and the caustic soda being withdrawn at the other electrode. In the manufacture of hypochlorite solutions, on the other hand, the liberated chlorine and sodium hydroxide have to recombine to form the hypochlorite NaOCl.

A widely accepted explanation of this reaction (due to Mr. Bertram Blount) is

$$2NaOH + 2Cl = NaOCl + NaCl + H_2O.$$

A different view is, however, held by the inventor of the new form of cell for this purpose, Mr. W. Pollard Digby, of London, England, namely, that for low strengths the reaction is

$$2NaOH + 2Cl = 2NaOCl + H_2.$$

His reasons for favoring this most simple hypothesis are two-fold. Firstly, the large number of authenticated cases in

which the electrochemical efficiency of production has not been 50 per cent or under as would agree with the first hypothesis but upwards of 80 per cent or 90 per cent, and secondly, when the reaction takes place under special control there is a gaseous evolution on the mixture of the anodic and cathodic products.

In early forms of cell two electrodes were immersed in the electrolyte and the current switched on. Trouble was met from the cathodic hydrogen which with heavy current densities rose in a constant stream from the electrode and caused discomfort to the workers. In this class of apparatus the reunion of the anodic and cathodic products is a haphazard affair depending wholly upon the circulation of the main body of the electrolyte in which the recombination takes place.

It is also found that the electrochemical efficiency of the cell depends upon the richness of the solution in hypochlorite. That is to say that as the proportion of hypochlorite rises the efficiency falls. The consequence is that in order to avoid undue waste of electric energy it has been necessary to draw off the electrolyte when it has reached only a low hypochlorite content and has still a large salt content. In other words, it has been necessary to waste salt in order to save current, and this in large quantities.

For instance, at Maidenhead, England, when the hypochlorite solution produced contained about 3.5 grams of available chlorine per liter, the amount of sodium chloride present was about 6.5 per cent. That is to say, each gram of available chlorine produced carried with it over 18 grams of salt. In other words, only one-twelfth of the chlorine initially present in the salt became available in the hypochlorite form.

This difficulty has been got over in the new cell referred to above, known as the Digby electrolyzer.

In this cell the manner in which the anode and cathode products recombine is controlled so that the body of the electrolyte is not altered in composition and the electrochemical efficiency is maintained while drawing off a hypochlorite solution with a very small saline content.

The principle of the cell may be explained by considering a cell with two carbon electrodes separated by two porous diaphragms—in this case of asbestos. The diaphragms divide the cell into three portions, the two end ones being connected together by a glass tube and separated from the middle space.

The electrodes are placed in the end portions which are designed to fit them with but small clearance, while the main body of electrolyte is in the middle portion. When the cell is working caustic soda is formed at the cathode end and the consequently denser solution passes by gravity through the glass tube to the anode compartment where chlorine is being liberated. Thus the caustic soda is brought directly to be acted upon by nascent chlorine.

The main body of electrolyte in the middle portion of the cell is a saturated salt solution for better conductivity. The solution which replenishes the cathode compartment may be pure water or a dilute alkali solution, as sodium carbonate.

The ions migrate under the influence of the current, the chlorine to the anode and the sodium to the cathode and the caustic solution there formed passes round to the chlorine without taking any appreciable amount of salt with it to be wasted, while at the same time the body of the electrolyte remains a salt solution pure and simple, and the electrochemical efficiency is maintained.

In practice these cells are made with three or five plates, arranged in series, and channels connect the different elements of the whole cell so that the anode reaction takes place on the side of the middle plates and the cathode reaction on the other.

The most economical current density to employ depends on the relative cost of electric energy and brine in the district in question, and the cells are worked at this density, while the strength of the resulting solution is controlled entirely by regulating the rate of inflow of water to the cathode end. The main electrolyte is kept a saturated solution with excess of salt, and more salt is added as this excess is absorbed.

In previous cells it has been found necessary to cool the electrolyte artificially in order to maintain stability in the reactions, and Mr. Digby, in his patent, refers to arrangements for cooling the caustic solution as it passes on its way to the anode. But in practice it has been found unnecessary to use artificial cooling in this type of cell, the anode and cathode portions being directly connected by short ducts.

The Digby electrolyzer is remarkably simple and inexpensive in its construction. The ends are formed of slate slabs and the partitions of asbestos sheet. The sheets are separated by slate distance pieces and the whole bolted together while the carbon electrodes are fitted into their proper elements.

In some cases the slate has been replaced by wood suitably treated with tar to further reducing the cost of the cell. There are no moving parts and the whole action is perfectly simple, with practically no possibility of getting out of order. Should dirty brine be used for economy the cell may be periodically scoured out with a jet of water without disturbing any mechanical connection in any way.

Cells with two elements in series are made to work on a pressure of 10 volts and to give an output of half a pound of available chlorine per 24 hours when taking 7 amperes. Such a cell complete costs about $200.

A Boiler Plant Waste Meter

It has been thoroughly demonstrated that waste of heat in boiler plants may be greatly reduced by more careful attention to what is going on in the furnace. As with the improvements which have been obtained in the engine-room by the use of the indicator, the thermometer, the steam gage, and in the use of electrical measuring instruments, so in the boiler plant the measurement of coal and water on scales or by meters, and the measurement of waste by a CO_2 recorder result in considerably decreasing the cost of power production.

On the average it may be stated that only 60 per cent of the heat in the coal enters the boiler; 40 per cent is wasted. Of this 40 per cent the total waste is distributed about as follows: Loss in radiation, 5 per cent; loss in soot and smoke, 1 per cent; loss represented by actual loss of coal through the grates, 2 per cent; loss due to combustion of the carbon to carbon monoxide instead of dioxide, 2 per cent; heat carried away by the chimney gases due to their temperature, 30 per cent.

The amount of coal waste due to loss of heat up the stack is quite fully realized, as is the fact that considerable reduction may be obtained in the extent of the chimney loss by measurement of carbon dioxide. The relation between per cent carbon dioxide and heat lost may be shown diagramatically by the chart of Fig. 1. The areas in each case are proportional to the amount of heat wasted, and as these areas are reduced by decreasing the excess air, the per cent of CO_2 is increased.

The CO_2 recorder was thus devised to automatically inform the engineer of the amount of waste occurring. The CO_2 measurement gave an indication of how much air was being used per pound of coal, the most important item in determining the amount of loss in the waste gases going up the chimney. The other important item, the temperature, is taken by pyrometers, mercury thermometers or other means.

Caustic potash absorbs carbon dioxide. This is the fundamental principle of all instruments for making a measurement of CO_2. In the older types of automatic recording instruments bells or jars for measuring the gases before and after the CO_2 was absorbed were used. The difference in volume of the gas before and after the CO_2 was abstracted gave a measurement of carbon dioxide in the gases. Valuable as they are, such instruments have, however, certain limitations.

If, instead of measuring the per cent carbon dioxide by measuring the actual volume of a given amount of gas before and after the carbon dioxide has been abstracted from it, the *pressure* of the gas is measured, then the change in pressure of the gas due to the reduction in its volume when the CO_2 is

absorbed will give a ready and reliable method of measurement.

The diagram of Fig. 2 shows how this principle is employed in the Uehling waste meter for measuring both percentage of carbon dioxide and temperature. Gas is drawn through the apparatus continuously by a small steam aspirator. Between the two apertures—that is, in the chamber marked *C*—the carbon dioxide is absorbed (in the case of the carbon-dioxide instrument) and thus the volume of the gas between the apertures is reduced by an amount depending upon the amount of CO_2 present. Any reduction in the volume of the gas between the two apertures increases the vacuum. These changes are proportional to the amount of CO_2 absorbed and thus the suction or partial vacuum forms a basis for indicating and recording variations in carbon dioxide in flue gases.

The indicating instruments are simply columns properly calibrated and are supplied at the machine proper and also as shown in Fig. 3 for the front of the boiler. The indicators at the boiler front are found to be most useful in that reference can be made continuously to them, whereas the recording charts are more valuable as permanent records and records from which to obtain average results with different kinds of coal, different grate area, different drafts, etc. The recording instruments may be placed anywhere and any number of them may be used as all they do is to record the partial vacuum or suction.

As the same principle and similar apparatus is used to measure temperature it is evident that a very effective combined CO_2 machine and temperature machine or waste meter is built on this principle. In measuring temperature, instead of the change in vacuum being produced by absorbing part of the gas between the apertures, change in vacuum is secured by reason of the difference in temperature between the two apertures. One aperture is in a nickel tube placed in the flue and the other is maintained at a constant temperature by exhaust steam.

The appearance of a combined instrument of this type may be seen in Fig. 4, a photograph of a Uehling waste meter installed in the plant of Robert Gair Company, Brooklyn, N. Y. Here the two indicating columns for per cent carbon dioxide and for temperature may be seen at the front of the instrument proper, while on the board are the recording gages for temperature and carbon dioxide. These are at the left and right. In the boiler-room proper is placed the CO_2 indicator.

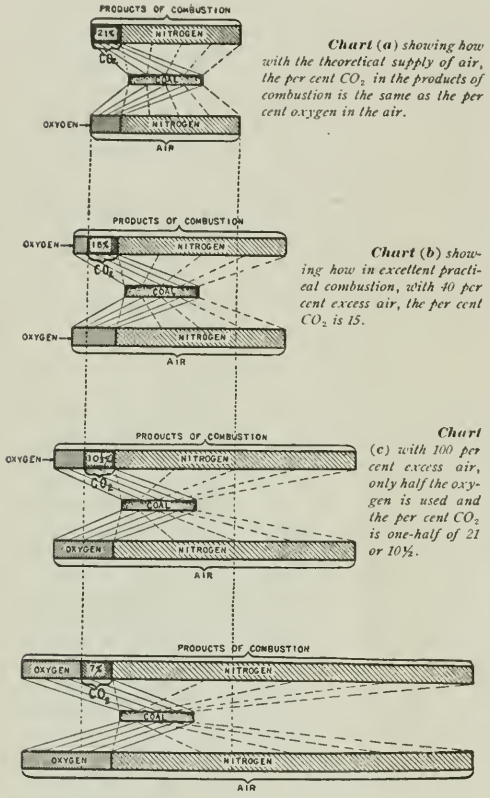


***Chart (a)** showing how with the theoretical supply of air, the per cent CO_2 in the products of combustion is the same as the per cent oxygen in the air.*

***Chart (b)** showing how in excellent practical combustion, with 40 per cent excess air, the per cent CO_2 is 15.*

***Chart (c)** with 100 per cent excess air, only half the oxygen is used and the per cent CO_2 is one-half of 21 or 10½.*

***Chart (d)** with 200 per cent excess air, only one-third the oxygen supplied is needed and the per cent of CO_2 is one-third of 21 or 7 per cent.*

FIG. 1.—CHARTS SHOWING THE CONDITIONS FOR DIFFERENT PERCENTAGES OF CARBON DIOXIDE.

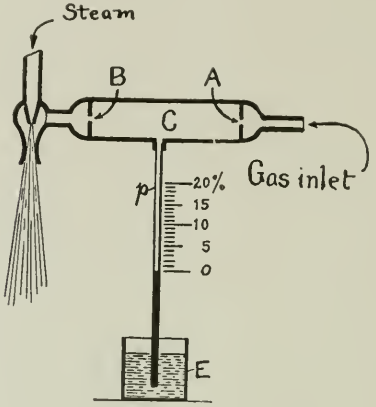


FIG. 2.—PRINCIPLE OF WASTE METER.

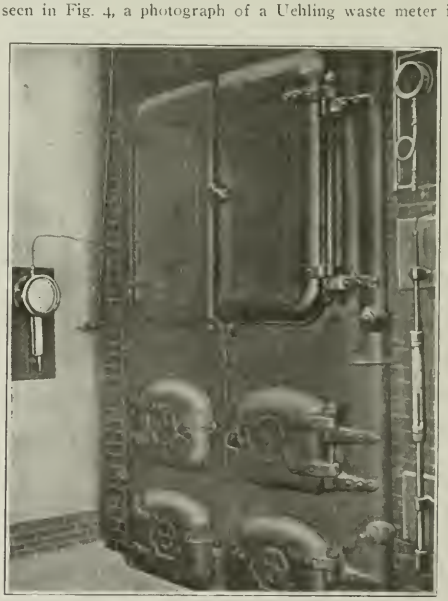


FIG. 3.—BOILER WITH CO_2 INDICATOR AT RIGHT AND DRAFT RECORDER AT LEFT.

Carbon Tetrachloride as Fire-Extinguisher.

The cheap electrolytic preparation of chlorine from brine has greatly reduced the cost of making carbon tetrachloride, a highly-valued solvent for fats and oils, so that it can even be used as a fire-extinguisher, as noticed in London *Engineering* of April 28.

Various compounds of carbon with chlorine and with the other halogens, fluorine, bromine, and iodine are known, and they are regarded as derived from methane (the fire-damp of the miner), CH_4, in which the hydrogen atoms are replaced by halogens. When one or three atoms of hydrogen are replaced, the resulting compounds are anæsthetics producing sleep (especially chloroform, $CHCl_3$) or antiseptics (iodoform CHI_3). Several of the other compounds are important in the dye industry; but, with the exception of the fluoride CF_4, none of them can be prepared by the direct union of the elements.

The tetrachloride is a colorless liquid, characterized by a pleasant smell, recalling that of Rhine wine; it freezes at — 25° Cent. and boils at 78° Cent., but the vapors are practically non-inflammable and non-explosive and stifle combustion, and the carbon tetrachloride thus occupies an exceptional position.

Various processes for the preparation of the tetrachloride have been patented in recent years, especially after Edward R. Taylor succeeded in making as much carbon bisulphide from coal and sulphur in his simple electric furnace as the United States requires. According to J. A. Robinson, who drew attention to carbon tetrachloride in the *Quarterly* of the National (American) Fire Association in January last, the tetrachloride is made by passing chlorine gas over heated carbon bisulphide. The condensed products contain carbon tetrachloride and sulphur dichloride; the latter is decomposed by the injection of caustic soda, and the tetrachloride is purified by distillation. It is put on the market in 100-gallon drums; the density of the

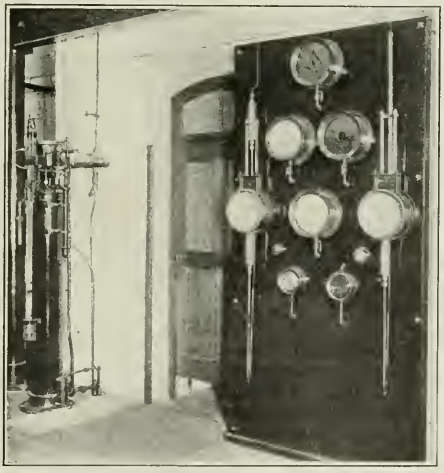


FIG. 4.—INSTALLATION OF WASTE METER IN PLANT OF ROBERT GAIR COMPANY IN BROOKLYN.

compound is 1.604, and the pound costs about 10 cents (5d.).

Fire-extinguishers filled with tetrachloride have been used by the Interborough Rapid Transit Company of New York on their cars. It has been pointed out that carbon tetrachloride, being a non-inflammable and non-corrosive substance, would be preferable to water and sand for extinguishing undesired arcs in electric installations. While no electrical engineer would probably resort to water in such a case, yet the tetrachloride may be useful.

In the oil and fat industry mixtures of carbon tetrachloride and naphtha are used, and it is interesting to note that, according to E. A. Barrier, the addition of tetrachloride would make the naphtha safe against fire and explosion. Considerable percentages of the tetrachloride would, however, be wanted. A naphtha distilling at about 55° Cent. should contain 30 per cent of carbon tetrachloride. A motor gasoline (70° Cent.) should be mixed with 50 per cent of tetrachloride to be free from explosion hazards, and with 60 per cent to be free from fire hazard; but in all these cases an additional 5 per cent of tetrachloride would technically be considered advisable.

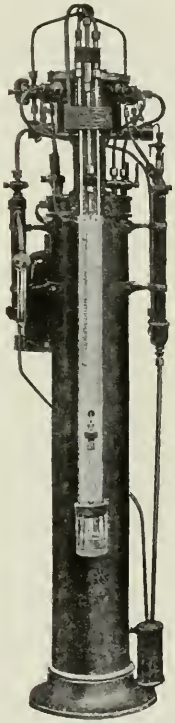


FIG. 5.—UEHLING WASTE METER.

The continued contact between hot carbon tetrachloride and water or water vapor should not cause direct dangers. The resulting products would be carbon dioxide and hydrochloric acid, not pleasant, yet not particularly dangerous gases. But it would be rash to predict that no other reactions might take place during a large fire; carbon oxychloride (sometimes called phosgene, because it is formed when carbon monoxide and chlorine are exposed to sunlight) might also be liberated, *e.g.*, in highly unpleasant vapors, which water would decompose into carbon dioxide and hydrochloric acid.

Tests and Analyses.

The Electrical Testing Laboratories, Eightieth Street and East End Avenue, New York City, which in the past confined their activities chiefly to testing work in the electrical and illuminating engineering fields, have recently extended their equipment so as to include coal testing, paper testing, tests of structural materials and thermometer tests.

Both chemical and calorimetric analyses of coal are made. An Atwater calorimeter and an Emerson calorimeter (both bomb types) are available. All thermometers used have been calibrated by the Bureau of Standards at Washington, as also has the standard combustible which is used to standardize the calorimeters.

For paper testing special apparatus has been installed, to determine the material from which the paper is made, various physical properties such as bursting strength, tensile strength, folding strength, etc., and certain facts of a chemical nature such as kind of size and filter, amount of ash, presence of acid, chlorine, etc.

For testing structural materials a 200,000-lb. testing machine has been installed for tension, compression and transverse testing of steel, iron, brick, stone, concrete, timber, etc. A torsion testing machine of 230,000 in.-lb. capacity has also been provided. The machine is capable of taking specimens from 6 in. to 20 ft. in length, and up to 3½ in. in diameter. This equipment together with the smaller machines already installed enables the Electrical Testing Laboratories to make any of the usual tests on materials used in industrial and structural enterprises, such as wire, cable, rope, chain, belting, iron, steel, alloys of various kinds, building materials, etc. A complete equipment for making and testing cement briquettes according to standard specifications has also been installed.

For the thermometer department special apparatus has been

built for the testing of thermometers of all kinds used for industrial purposes up to 500° C. and also for testing clinical thermometers. All apparatus has been built similar to the Government standard apparatus at Washington and the methods used are also the same. The standards employed by the Electrical Testing Laboratories are those of the Bureau of Standards in Washington, D. C.

NOTES

The Abbe Engineering Company, of 220 Broadway, New York City, has removed its office from the twelfth to the eleventh floor of this building, since their rapidly growing business in tube mills, pebble mills and ball mills required an enlargement of their quarters.

The American Grondal Company is the new name of the American Gröndal-Kjellin Company. This change of name has been made in connection with the broadening out of the scope of their work in concentrating and briquetting low-grade iron ores. The company will conduct complete commercial tests on any ore submitted in its testing plant, and, from the results of such tests, make a proposition for the proper method of treating the ore, design the plant, and if so desired, supervise the installation.

Vanadium.—Ever since the formation of the American Vanadium Company the entire product of vanadium alloys produced by this company has been marketed by the Vanadium Sales Company of America. The offices of both companies have been in the Frick Building, Pittsburgh, Pa., and their products, especially ferrovanadium, have been sold in every manufacturing country in the world. Recently the Universal Vanadium Company has been formed for the purpose of acting as selling agents for the American Vanadium Company in a more comprehensive and effectual manner, and it will act in a selling capacity in conjunction with the Vanadium Sales Company of America, both companies taking charge of the entire product of the American Vanadium Company.

The National Carbon Company, of Cleveland, Ohio, has just issued a publication entitled "Practical Operation of Arc Lamps." It is full of useful instructive information for the use of lamp owners and operators. A carefully prepared alphabetical index deserves special mention.

Earthenware Apparatus.—Hard, white vitreous earthenware and porous earthenware are finding increased use in chemical and electrochemical work. Messrs. John Maddock & Sons, of Trenton, N. J., who make a specialty of the manufacture of such earthenware apparatus (which should not be confounded with chemical stoneware), make these goods from either white porous clay or a white vitreous body, either glazed or unglazed, with a transparent glaze. They are making large quantities of white vitreous battery jars and also large quantities of round porous cups, there being practically no limit to their size. They also make quantities of special porous oblong boxes.

Spray Nozzles.—The Buffalo Forge Company, of Buffalo, N. Y., have sent us their pamphlet on Buffalo spray nozzles and strainers. As is well known, the spray nozzle gives an exceedingly fine spray with a minimum water pressure. Operating at pressures as low as 15 lb. the centrifugal action of the nozzle produces an atomizing effect which is further increased with the increase in pressure. The greatest atomizing effect is obtained at a pressure of about 40 lb. The successful results obtained with these nozzles are due to the fact that the water enters the circular chamber tangentially, receiving a whirling motion, and is not obstructed in any way until its exit through the opening in the disk. As it approaches this opening an increase of velocity is caused by the change in the shape of the passage until it bursts into an atomized spray, which does not depend for its fineness on the small size of the orifice. For this reason large water passages are used throughout, with consequent freedom from clogging. The nozzle is made in two parts. The cap can be replaced if the action of the liquid is such as to corrode the metal and destroy the effectiveness of the spray. While regularly furnished in brass, the Buffalo Forge Company makes these nozzles in large or small quantities of any material desired, like lead or brass. It is particularly interesting to note that monel metal nozzles have replaced others in producer-gas power plants, blast furnaces and chemical plants.

The Ashland Fire Brick Company, of Ashland, Ky., is sending out a series of very clever bulletins which impart interesting information in an easy conversational style. From one of them we learn that the flint clay from which their brick is made is permitted to weather in the yards for six months before it is used. Clay when it comes from the mine is very hard and brittle, but weathering grinds it to fine particles and many of the impurities are washed away. The result is a better and smoother brick.

Casting Alloys is the title of a pamphlet recently issued by the Lumen Bearing Company, of Buffalo, N. Y. The different bronzes made by this company are classified according to their uses and concisely described as to their properties. For chemical and metallurgical purposes their alloy No. 0 is particularly interesting. It is a copper-tin acid-resisting bronze, for pump bodies, valves and other parts coming in contact with fluids containing alkaline or acids. Its tensile strength is 50,000 lb.; specific gravity, 9.05; weight per cubic inch, 0.3258 lb.; the shrinkage in sand molds, ⅛ in. per foot.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in February, 1911, was 69. The total tonnage milled during that period was 1,749,834 tons of ore: 9627 stamps were in operation with an average duty of 7.5 tons per 24 hours. Tube mills in commission numbered 201. The yield for the month was 610,828 fine ounces gold. The number of tube mills in operation shows an increase of 30 per cent as compared with February, 1910.

Chas. C. Moore & Company, engineers, San Francisco, Cal., and the Pacific Coast managers of the Babcock & Wilcox Company, have recently prepared for distribution among the engineering fraternity a notebook specially designed for the convenience of those who have to keep many individual notes and memoranda. One of the books and extra inserts (the memoranda being detachable) can be obtained by addressing the company.

The Utah Society of Engineers held its annual dinner at the Commercial Club, Salt Lake City, on April 20, 1911. Addresses were made by John Dern, president of the American Mining Congress, and various other members of the society. The officers for the ensuing year were elected as follows: President, M. D. Grosh; first vice-president, C. F. Moore; second vice-president, E. H. Beckstrand; secretary, R. B. Ketchum; treasurer, A. S. Peters.

The General Engineering Company, Salt Lake City, Utah, has recently moved into new quarters, where it has one of the most complete ore-testing plants in the West. Tests by concentration, cyanidation, amalgamation, flotation and magnetic concentration can be made on lots of ore ranging in quantity from 50 lb. to a carload. The company also has put in a complete cement testing laboratory.

The Ohio Copper Company, Lark, Utah, is putting in 60 Deister No. 3 concentrating tables to handle the slime. This change is expected to put the Ohio mill in a better position to save fine mineral and will also materially increase the capacity.

In conveyor belt installations some of the features which require attention are the thickness, width, material, and cover of the belt, the spacing of the carriers, the speed of the belt, and the proper feeding of the materials to be conveyed. These features vary with almost every installation, and should be determined specially for each plant.

Personal

Mr. J. M. Callow, of the General Engineering Company, Salt Lake City, has been appointed consulting metallurgist for the Inspiration Copper Company.

Mr. Francis E. Drake, who for several years has been the managing director of the Société Anonyme Westinghouse of Paris and Havre and principal officer in several of the Continental Westinghouse interests, has resigned from the active management of these properties to occupy himself with certain private interests, and, after a period of rest and recuperation, will establish his office at No. 16, rue Halevy, Paris. Mr. Drake is well known, not only in electrical, but in metallurgical circles, since before he went to Europe he was the president of the Lanyon Zinc Company and it was under his presidency and direction that the first experiments in electric zinc smelting in America were carried out by his company. Mr. Drake is interested in several electric enterprises in Europe and will carry out, as engineer and contractor, certain tram and railway lines now in process of development. The efforts of Mr. Drake in behalf of electric traction, especially in the direction of single-phase development and for other aid rendered to French industries, recently received the recognition of the French government in the shape of promotion to the rank of Officer in the Legion of Honor. Mr. Drake is a member of the Engineers' Club and a life member of the American Chamber of Commerce in Paris; he is also a member of the American Electrochemical Society.

Dr. W. C. Ebaugh, professor of chemistry in the University of Utah, has been granted a year's leave of absence for 1911-12.

Mr. J. R. Finlay, formerly with the Goldfield Consolidated Mining Company, Goldfield, Nev., has opened offices at 52 William Street, New York, as a consulting engineer.

Mr. A. I. Goodell is acting as Northwestern representative for the International Smelting & Refining Company.

Mr. Philo D. Grommon has been appointed superintendent of the mines and mill of the Ophir Gold Mines Company, Ophir, Col.

Mr. H. R. Holbrook, formerly employed by the Balaklala Copper Company, at Coram, Cal., has accepted a position as chief chemist for the Mexican-American Smelting & Refining Company, at Guaymas, Sonora, Mexico.

Mr. Theodore P. Holt has been appointed assistant professor of mining and milling in the University of Utah, where he has been an instructor since his graduation.

Mr. J. E. McAllister, until recently general manager of the British Columbia Copper Company, has resigned and will enter consulting practice in New York.

Mr. A. H. Prior, who has been cyanide superintendent of the Rio Plata Mining Company, has been appointed general superintendent, vice H. W. Edmondson, resigned.

The wedding is announced, on June 1, of Mr. **Hugh Rodman,** of the Rodman Chemical Company, of Pittsburgh, with Miss **Mary Bynum Iarnagin.**

Mr. Benjamin B. Thayer, president of the Anaconda Copper Mining Company, recently made his annual trip of inspection to the mines and reduction works of the company in Montana.

OBITUARY

James B. Daniels, general manager of the milling department of the Portland Gold Mining Company, Colorado Springs, Col., died on April 3, 1911, following an operation for appendicitis performed four days previously. Mr. Daniels had been identified with the metallurgical industry in Colorado since 1879, when he came to Leadville and entered the old Harrison Reduction works, which was one of the early smelters in that city. From that time until 1903 he was connected with smelters at Denver and Pueblo. Later he left the lead-smelting industry and entered the employ of the Portland company, which is engaged in the treatment of Cripple Creek ores by the chlorination and cyanide processes. Mr. Daniels died at the age of fifty-one, and is survived by a widow. He was highly esteemed by the officials of his company, who regarded him as a large factor in the success of their operations. He was greatly interested in the new process which the company has in operation at Victor, and was instrumental in bringing it to a successful conclusion.

Samuel F. Emmons, eminent geologist and pioneer in the economic branch of that science, died in Washington, D. C., on March 28, 1911. His last illness was of but five days' duration, but for a year or two previous he had been in poor health. Mr. Emmons was probably the foremost geologist in his country, and was widely known throughout the West, where his most important work was done. He was one of the first to apply the science of geology to mining, and his work in that line has led to the most remarkable mining developments which have been known in the United States. Especially was his work on the mining and geology of Leadville a masterly production and his greatest work. This was undertaken for the United States Geological Survey, in which organization Mr. Emmons was a leader for many years. His first connection with government work was in 1867 when he accompanied the United States Geological Exploration of the fortieth parallel. He made many contributions to the published reports of this expedition, and has been a noted contributor to the various publications of the Survey. As a member of many scientific societies he was always active in their interests and received many honors at their hands.

Mrs. Ellen H. Richards, wife of Prof. Robert H. Richards, of the Massachusetts Institute of Technology, died at Boston March 30, 1911, after a week's illness. Mrs. Richards was instructor in sanitary chemistry at the institute, from which also she was a graduate in 1873. In 1875 she married Professor Richards, who is at the head of the department of mining and metallurgical engineering, and together they continued on the instruction staff of the institute until the time of her death. It is given to few women to hold as prominent a position in the technical world, and live a life of such useful service as fell to the lot of Mrs. Richards. Especially in the application of science and scientific methods to social and domestic economy she was very active and helpful. In her later years she acquired an international reputation, and was honored at home by receiving the doctor's degree from Smith College. She was a member of many scientific societies and the author of important works on chemistry.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

351,576, Oct. 26, 1886, Henry R. Cassel, of London, County of Middlesex, England, assignor to the Cassel Gold Extracting Company (Ltd.), of Glasgow, Scotland.

Gold, etc. Ores or alloys are powdered and charged into an anode compartment separated from the cathode compartment by a diaphragm of asbestos. Metal is dissolved in the anode compartment and precipitated at the cathode. The apparatus illustrated comprises a stationary tank containing a revolving drum, constituting the anode compartment. The circumferential wall of this drum is of asbestos cloth, having a removable portion to admit the charge. A series of carbon anode rods are arranged in a circle within it, extending hori-

zontally through the end walls of the drum. The drum is surrounded by a tubular sheet-metal cathode, in two halves. Illustrates a specific connection for carrying the current through the revolving drum shaft to the anode. The anode compartment contains the chloride solution, and the drum is preferably submerged in the electrolyte in the tank.

257,059, Feb. 15, 1887, Desmond Gerald Fitzgerald, of Akerman Road, Loughborough Road, Brixton, County of Surrey, England.

Gold. Employs anodes of lead peroxide for electrolytically generating chlorine. The anode may be molded of a plastic mixture of lead oxide and ammonium sulphate or chloride, e.g., being plastered onto the inner surface of the cell as a lining. Prefers, however, to build up the anode from slabs, plates, tiles, fragments or granules of a molded mixture of litharge and ammonium sulphate, or hydropotassic or hydrosodic sulphate, peroxidized by a hypochlorite solution and set in the plastic lead peroxide mixture. The terminal is a plate of platinum embedded in the upper part of the anode. The cathode may be a plate of copper. Illustrates a cell consisting of a shallow tank of slate or brick, with suitable lining, the anode being formed as a tray lying in one side of the tank, while the cathode is a copper tray coated with insulation at the other side. May use plates of lead peroxide as anodes, alternating with suitable cathodes, to produce hypochlorites or chlorates.

359,156, March 8, 1887, Jean Antoine Mathieu, of Detroit, Mich.

Treats lead ores, e.g., carbonates containing gold and silver, by crushing and grinding, submitting to the steam from boiling acetic acid from three to ten hours, and boiling in acetic acid from three to five hours, while stirring, to dissolve out the lead sulphate and the silver and gold sulphates and chlorides. The bath is then allowed to cool to slightly below the boiling point, the solid matter settling out and being separately treated. The gold and silver are deposited from the solution, using lead electrodes.

360,852, April 12, 1887, Henry Renner Cassel, of New York, N. Y., assignor to the Cassel Gold Extracting Company (Ltd.), of Glasgow, Scotland.

Treats refractory pyritic gold ores with electrolytic chlorine and oxygen. The apparatus is a rotary drum of wood, or metal lined with insulation. It contains a series of carbon-rod anodes extending from end to end or may be lined with carbon-plate anodes. The drum is mounted upon a hollow perforated shaft of copper or iron, closely surrounded by a tubular diaphragm of asbestos cloth. The shaft extends at its ends into external vertical tanks, which are connected by an outside pipe. The drum is filled to a point above the shaft with a sodium chloride solution, and is charged with the pulverized ores. Current is supplied and as the drum rotates the metals are dissolved and deposited within the shaft and upon a metal screw conveyor therein in the form of a black slime, which is conveyed into one of the external tanks. The solution circulates freely between the tanks and through the drum.

360,853, April 12, 1887, H. R. Cassel, assignor to the Cassel Gold Extracting Company (Ltd.), of Glasgow, Scotland.

Apparatus for treating gold ores with electrolytic chlorine and oxygen from a sodium chloride solution. The apparatus is a rotary drum of wood or metal lined with insulation. This drum contains a circular series of carbon-rod anodes, extending horizontally through the ends of the drum, or may be lined with carbon or graphite anode plates. The ends of the drum contain large openings, receiving flaring metal casings constituting extensions of the drum shafts. The openings contain asbestos-cloth diaphragms. The ends of the shafts pass through stuffing boxes and extend into and open within hollow standards serving as tanks. The metal casings, shafts and tanks constitute the cathode compartments. One casing has a gas-vent opening, normally closed by a spring-actuated valve. This valve is automatically opened by a cam as it rises above the liquid level to permit the gases to escape. In use the drum is filled to or above the level of the shaft with a sodium chloride solution and is charged with the pulverized ores. Current being supplied, the metal dissolves, the solution passes through the diaphragms, and the metal is deposited within the trunnions and tanks as a black slime.

BOOK REVIEWS.

Practical Alloying. A compendium of alloys and processes for brass founders, metal workers and engineers. By **John F. Buchanan.** Octavo, 205 pages, 41 illus. Price, $2.50. Cleveland, Ohio: The Penton Publishing Company.

The bulk of the matter in this book appeared in *The Foundry* and other trade magazines. A long chapter on Metal Refining (in which term the author says he means to include reducing the metals from their ores) is picturesque writing, but neither very instructive nor accurate. The next, on History and Peculiarities of Alloys, is similar, only more so; it contains several absurdities. After this bad beginning the author settles down to business, commences to talk about matters with which he is familiar and redeems himself in a dozen chapters containing a large amount of valuable first-hand information. Many good photographs, on plate paper, add to the attractiveness of the work. The proposition to designate alloys is worth mentioning: Brass (copper, 60; zinc, 40) would be expressed as Cu-Zn (60-40) thus separating the numerals from the symbols and avoiding all confusion with the ordinary chemical formulas, such as Cu_2Zn; plumbers' solder would be Pb-Sn (2-1); silicon bronze, Cu-Sn-Si (90-8-2), etc.

We recommend the book as being very practical and well written except where too flowery. Everyone having anything to do with alloys—makers, users or students—will find it worth buying.

* * *

A Course in Qualitative Chemical Analysis. By **Charles Baskerville,** Ph.D., and **L. J. Curtman,** Ph.D. 8vo, no illustrations, 200 pages. Price, $1.40 net. New York: The Macmillan Company.

Qualitative Chemical Analysis, from the Standpoint of Solubilities, Ionization and Mass Action. By **J. I. D. Hinds,** Ph.D., L.L.D. 8vo, no illustrations, 265 pages. Price, $2. Easton, Pa.: The Chemical Publishing Company.

Qualitative Chemical Analysis. A laboratory guide. By **W. W. Scott,** A. M. 8vo, 31 illustrations of apparatus, 1 spectrum plate, 165 pages. Price, $1.50 net. New York: D. VanNostrand Company.

The above books, almost alike in size and scope, are yet somewhat different in treatment.

Professor Baskerville and Dr. Curtman are connected with the College of the City of New York. Their book is clear, systematic and full of bright suggestions; it extends only to the common bases and acids; numerous "notes" will be very helpful to the students. The book is well suited for a short course in a college laboratory.

Dr. Hinds is professory at the University of Nashville. His is typographically the best of the three books. The great prominence given the terminology of the ionization theory in this book is, in the reviewer's opinion, a hindrance rather than a help to the beginner trying to learn the elementary facts of qualitative analysis. If a student reported that a white powder given him for analysis contained "lead ions" and "NO_3" ions he would be reporting according to Dr. Hind's scheme, but hardly in accordance with the facts.

Mr. Scott is chief chemist of the Baldwin Locomotive Works, and his book is the most businesslike of the three. The contents cover almost the same extent of ground as the other two, but in a more practical fashion than either, and without the "ionic" form of expression of the second. It is well and forcibly written.

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Research Clubs at Mills.

An opportunity for experiment and investigation exists at most metallurgical plants of any magnitude, which requires only a little enthusiasm and direction on the part of a leader to develop results that will be well worth the effort. We do not refer to the elaborate testing departments and research laboratories maintained by the larger companies, but more especially to less formal associations in which the esprit de corps is the basis of organization and provides the needed incentive for their work.

Conditions are favorable for such an organization at almost every plant of importance. In the first place, it may be assumed that every man employed is, or should be, interested not only in the routine work but in his own progress and education as well. As most mills are operated twenty-four hours in the day, it is possible to carry on some experimental processes continuously by having men on each shift give a little time and attention thereto. And last, but not of least importance, such an organization would be of special value to men working at mills which are more or less isolated from other centers of interest. The social feature of life at these places is generally recognized by companies, and clubs usually are provided for the men. It would not detract seriously from the relaxation which the club is designed to afford, if say, one evening a week were devoted to an informal discussion of professional work and if this interchange of ideas was further extended to planning experiment and research.

Both the men and the company would profit from such a plan. To the former it would be a stimulus, creating genuine personal interest in the work and bringing out latent ability. This would enlarge their view and offer opportunity for advancement. The benefit to the company would lie in improved work, cheapened cost of operation, elimination of unnecessary steps in the process, and a spirit of co-operation among its employees. Even more substantial results might follow, such as the successful treatment of an ore formerly refractory to the process employed.

As said before, a leader who can suggest lines of investigation is invaluable in such a plan, and the company must acquiesce in making the necessary expenditures. We recall several instances where this arrangement is working successfully. At Goldfield the Florence mill is equipped with a special laboratory where the men are encouraged to make experiments bearing on the cyanidation of gold ores. The Portland Metallurgical Society was formed of employees of the Portland Gold Mining Company, and was the direct means of evolving the process now used by the company at its Victor mill. Recently Mr. Dorr has organized his men at the Mogul mill in South Dakota into a research club from which good results are expected. Doubtless other instances of clubs organized on this plan could be found, but there is still room for many more.

Continuous Cyanidation.

Hitherto in these columns emphasis has been laid on the value of continuity in metallurgical processes. Compared with intermittent procedure, the continuous method gives greater service per unit of time and labor, adds to net profits and reduces the losses sustained in the repetition of an intermittent method. In two of the principal steps in the cyanide process, agitation and filtration, we have seen remarkable progress owing to the introduction of the continuous feature, and this idea has persisted in the minds of metallurgists so that they are endeavoring to secure continuous operation wherever possible.

In this connection we are glad to present to our readers a timely article on continuous cyanidation, by Mr. John E. Rothwell, who has been studying and developing this idea for several years. A study of the flow sheet accompanying the article will reveal the advantages claimed by the author for the method. Mathematical accuracy and definiteness are substituted for guesswork and rule-of-thumb procedure. The operator may know just what is going on in any part of the process at any time. By means of measuring devices, such as diaphragm pumps, the quantities of solution or pulp delivered from one point to another are determined exactly. There is a definite quantity of solution of definite value drawn from the system for precipitation of the dissolved values. And yet, with all the mathematical nicety which marks continuous operation, there is ample elasticity to provide for accidents or other contingencies in that part of the treatment which concerns extraction by cyanide.

The complete absence of filters in this method is a notable feature considered from the financial standpoint. The cost of installation of a continuous treatment method probably would be less, and certainly not more, than that of a plant using filters, while the cost of subsequent operation certainly would be less. Extraction also should be improved owing to the manner of applying the countercurrent of solution to the ore. It will be noticed that the weakest and lowest grade solution is applied to the ore when it has been almost exhausted of its precious metal content, and that the solutions can be strengthened by the addition of cyanide as they come in contact with richer ore. As pointed out by the author, this feature insures the highest efficiency from the solvent.

As a contribution to progress in cyanidation the idea of continuous treatment is very timely, and we invite metallurgists to a free discussion of its merits.

The Cyanamide Process in Gold Metallurgy.

Experimental work with the Clancy cyanamide process continues to absorb the attention of metallurgists in the West. The experiences are varied, and the indications are that the ordinarily capable experimenter is not equipped, either by education or experience, to account for all the results he obtains, or to obtain uniform results continuously. This does not reflect unfavorably on the merits of the Clancy process, nor does it derogate from the correctness of the underlying principles. It mainly calls attention to the fact that the chemistry of the organic compounds of gold is not as well known as it will be a few years hence, and that the conditions for successful operation may be confined within comparatively narrow limits.

It is, therefore, timely to caution against promiscuous experimentation, as well as against the error of drawing hasty conclusions from a few results. There are so many variables to be taken into consideration that it requires patient, continuous and intelligent experimenting to determine them singly and in combination. The difficulties are increased because the present accumulation of knowledge on the subject is small, and hence each investigator must work according to the best light he can obtain. It would be of great benefit to the fraternity if metallurgists would contribute their results; not putting them forward as from ripe experience, but as tentative statements of observation.

One of the difficulties encountered is the satisfactory chemical control of the process. We are informed that the titration of regenerated cyanide solutions offers some difficulty. Results are varied, and for reasons not always apparent. It is stated that some regenerated cyanide solutions will show a decided solvent action on gold and silver, but that the same solutions will not give a titration for cyanide according to the usual methods. What condition exists in these solutions? What is the solvent, and how is it to be determined as regards quality and quantity? What are the effects of various current densities, and what arrangement and size of electrodes will give the best results? Can regeneration of the cyanide solution be well accomplished in contact with the ore pulp, or is it better effected in clear solution after filtration?

These and many other questions have confronted the present investigators, and the answers are not yet made to their satisfaction. The free exchange of opinons and results will solve these points, just as similar problems were solved in the standard cyanide process which has been brought to such a high state of perfection.

Co-operation and Competition in the Iron and Steel Industry

The public was given further enlightenment last month as to the attitude of Mr. E. H. Gary, chairman of the United States Steel Corporation, with respect to the corporation's position and the campaign of price maintenance, technically known as "co-operation," which he has conducted so successfully for several years. Inasmuch as Mr. Gary's attitude has not changed of late, and as the pursuance of the campaign necessitated his expounding his views from time to time, there was nothing strictly for him to say on these subjects when he appeared before the Stanley committee of the House of Representatives, pursuing an investigation of the affairs of the United States Steel Corporation and its relations with the independent manufacturers. However, the occasion was afforded for Mr. Gary to re-state his views, fully and distinctly, and he made most of his opportunities.

The policy of "co-operation" is offered as a modern invention, a substitute for the old practice of "destructive competition." The work is conducted very simply, the steel manufacturers merely meeting from time to time and then and upon other occasions freely exchanging information as to their business affairs, there being in particular an arrangement

whereby, while no manufacturer is bound not to reduce his prices, he is expected to communicate news of his intention to the other manufacturers before he announces the reduced price to purchasers. The policy is simplicity itself, being essentially based on a frank and full exchange of information between the different manufacturers in the iron and steel industry.

Mr. Gary was candid, apparently, to the members of the committee, startlingly candid, when he expressed his strong wish that there should be a department of the government to which the management of the Steel Corporation could go when in doubt as to what the law allowed it to do, and, upon giving adequate information as to its position, receive counsel which if followed would relieve it from responsibility. We opine that Mr. Gary's attitude in this matter is due to his having stronger views as to the scope and bearing of the Sherman anti-trust law than are entertained in many quarters. There is no reason to infer that Mr. Gary courts government supervision, with regulation of prices, for any other reason than because of the potency of the Sherman law or the force of public opinion, which would be likely to make another law were it not in existence.

To draw a word picture illustrating the differences between the so-called "destructive competition" and this modern policy of "co-operation" may readily appeal to the senses, but it is not stating or developing a scientific law. What is this "destructive competition"? Is it not destroying that which is inefficient and making more durable that which is efficient? This in itself is more likely to be good than bad. It is true Mr. Gary suggested that without the formation of the Steel Corporation there was danger of a monopoly being created, for he stated as his firm belief that had the Carnegie Steel Company continued in its old position and with its old management it would have driven all other steel producers out of the field. We can hardly concur in that belief. Prior to the formation of the Steel Corporation, April 1, 1901, the Carnegie Steel Company was making 2,500,000 tons of steel, which at that time represented approximately one-fourth of the country's steel ingot production of 10,000,000 tons. The country's production last year was in excess of 25,000,000 tons. Certainly a war of destructive competition would have tended to increase, rather than decrease, the demand, and it is inconceivable that a single concern could carry on such a warfare and at the same time accumulate funds sufficient to multiply its productive capacity by ten.

Mr. Gary's testimony brought out what was emphasized at the time the Steel Corporation was formed that one of its objects was to forestall a large amount of new erection which was planned by the various interests consolidated, new erection far beyond what seemed to be in keeping with the country's requirements and which would probably have involved the most serious competition. Quite likely it was an economic gain to the country that such unnecessary duplication of plants should be avoided. In the light of the actual history of the iron and steel industries during the last years this economic gain, as a consequence of the formation of the Steel Corporation, can hardly be denied.

Even granting that the "destructive competition" is altogether bad, the history of the past few years does not necessarily indicate that the new principle which it is sought to put in its place is necessarily good. We observe in the first place that there was a period of three years, from late in 1904 until late in 1907, when demand was sustained and the increasing productive capacity of the iron and steel industry was continuously and fully engaged. During that period it was urged that steady prices meant steady demand, but in the retrospect it appears that the slightly advancing tendency which the market displayed during that period tended to encourage buying and in some cases induced the anticipation of wants. Usually there are some who wait, but these did not and the result was a twelvemonth in which demand was scarcely more than half what it had been in the preceding twelvemonth. Never before had there been so violent a decrease in demand and this despite the ever repeated preachment that steady prices meant steady demand.

A still more serious feature, however, is this, that the encouragement of buying in 1906 and 1907 created a false impression as to the ordinary iron requirements of the country; it set a standard which the trade expected to be lived up to nearly all the time. There was an inducement to new erection, and the policy of co-operation undoubtedly added a great incentive, for it involved, directly or indirectly, the maintenance of prices and the distribution of whatever demand there was along the different producers according to their capacity. Naturally the larger one's capacity the larger the share of the going business. Accordingly we find the industry's capacity, including a few furnaces not yet completed, at about 35,000,000 tons of pig iron per annum, practically 50 per cent in excess of the average demand in the three years of extreme prosperity, 1905-6-7, and still more in excess of the demand which has been experienced more recently in the past six or nine months.

Thus we cannot avoid the conclusion that if the policy of "co-operation" has eliminated the old fashioned "destructive competition" it has instituted a practice of unnecessary new erection. We are not prepared to give unqualified approval to either result.

It is said that the country must "grow up" to the new capacity. In point of tonnage perhaps it will. Certainly it will if the old law of increase in tonnage demand holds good, but we are not certain that it will. The requirements, in service to be performed, presumably will, but nothing is more certain than that much of the steel now being employed is not as suitable as might be for the purposes to which it is applied. Frequently the service can be more effectively performed by lighter but better steel, and we may find that, while the service performed by the new steel made doubles in the future as often as in the past, the tonnage required to perform the increased service will not increase in proportion. If that is so much of the new construction now indulged in, anticipating future requirements, is misplaced, whereas by waiting until the demand appears the investment could be made much more intelligently and with much greater effectiveness.

And yet, after all that may be said on the shortcomings of the policy of co-operation, the fact remains that this policy with its fixation of prices is in line with the tendency of the times toward scientific socialism. And as such it is not only logical but eminently practical.

Indianapolis Meeting of the American Chemical Society.

The Indiana Section of the American Chemical Society is making special efforts to make the summer convention unusually successful. This will begin on Tuesday, June 27, and last throughout the remainder of the week. The council will meet on the afternoon of Tuesday, June 27, and after a business meeting will be taken for an automobile ride to the Indianapolis Speedway.

On Wednesday the general session will begin and three general addresses will be given. Prof. Charles Baskerville, of the College of the City of New York, will speak on "The Chemistry of Anaesthetics." Prof. Edward Kremers, of the University of Wisconsin, will follow with a paper on "The Quinhydrone Hypothesis of Plant Pigmentation." The third address, by Prof. W. Lash Miller, of the University of Toronto, will be upon "The Chemical Philosophy of the High-School Textbooks."

These papers will be followed by a complimentary luncheon at the German House, and after luncheon meetings of the Agricultural and Food, Industrial, Organic, and Physical and Inorganic Divisions and the Biological Section will be held. In the evening a complimentary smoker will be held in the gardens of the German House.

On Thursday morning there will be meetings of the Agricultural and Food, Industrial, Organic, Pharmaceutical, and Physical and Inorganic Divisions, one of the special features being the meeting of the Pharmaceutical Division in the new Science Building of the Eli Lilly & Company, followed by a luncheon there to the pharmaceutical chemists. At the same time a complimentary luncheon will be given to the remaining chemists at the Indianapolis Brewing Company, followed by an inspection of their plant. In the afternoon all of the chemists will visit the plant of the Eli Lilly & Company.

On Thursday evening there will be a public lecture by Mr. A. D. Little at the German House on the earning power of chemistry. This lecture is one of the features of the meeting and is intended to interest chemists, business men and the general public alike. Following the lecture there will be a concert in the gardens of the German House.

On Friday there will be an early visit to the Indianapolis Water Works followed by meetings of the Fertilizer, Industrial, Physical and Inorganic Divisions and the Sections of Biological and of India Rubber Chemistry. At 12:30 these will be followed by a complimentary luncheon at the automobile factory of Nordyke & Marmon, followed by an inspection of their factory. Later in the afternoon visits will be made to the Encaustic Tile Company, to the Fairmount Glass Works and to E. C. Atkins & Company. In the evening the usual banquet of the summer meeting will be held.

On Saturday there will be visits to the Kingan & Company, Ltd., to the Polk Sanitary Milk Company and to the Diamond Chain Company; in the afternoon to the Van Camp Packing Company.

A special endeavor is also being made to see that ladies accompanying members have an enjoyable summer outing including automobile rides, receptions, theater parties and luncheons.

Over 150 papers have already been received for the divisional meetings.

Dr. Charles L. Parsons, Durham, N. H., is the secretary of the Society.

The Western Metallurgical Field.

The Miami Mill.

The Miami copper mill in Arizona is one of the large plants that recently entered the list of producers. The general line of ore treatment is not dissimilar to that adopted in the other large copper mills treating disseminated, low-grade copper ores. The following outline was given at the May meeting of the New York Section of the Mining and Metallurgical Society of America by Mr. J. Parke Channing:

The ore is crushed in gyratory crushers to 2-in. cubes and screened through cast-steel trommels having ¾-in. holes. The oversize is reduced to ½-in. cubes by 54-in. x 24-in. rolls, and is added to the undersize of the trommel. This is elevated and distributed to six 1000-ton bins, each representing a unit in the mill. The ore is automatically weighed and sampled as it leaves these bins, and then flows into a shaking launder with screen bottom, where water is added. The oversize, larger than ⅛ in., is then reduced in Burch high-speed rolls and added to the launder undersize. The mixture is then divided, one-half going to each of the two sections of the unit.

The treatment in a section follows: Over two Callow screens of a mesh of 0.024 in., oversize to a 6-ft. Chile mill, at present provided with 0.029-in. screen. The two products join and go to a drag classifier, which consists of an endless rubber belt with blades on the bottom. The coarse material from the drag classifier goes to a 10-spigot Richards hindered settling hydraulic classifier, the discharge from each spigot going to a Deister No. 2 sand table. Each table makes concentrate middling and tailing products. The middlings and certain tailings are ground in a 6-ft. Hardinge mill and classified into sand and slime, the sand being treated on two Deister sand tables, and the slime on a Deister slime table.

The overflow from the drag classifier joins the overflow from the end of the Richards classifier and goes to a large conical dewatering tank. The underflow from this is treated on Deister slime tables which make concentrates, middlings and tailings. The middlings are retreated on Deister tables. The overflow from the large settling tank flows to eight smaller conical tanks, the underflow of which is treated on Deister tables, making concentrates, middlings and tailings. The middlings join the coarser slime middlings and are treated on the two Deister slimers mentioned above.

The water recovery is accomplished as follows: The mill tailing goes to large wooden conical tanks, underflow to waste launder and overflow to 80 concrete settling tanks. Overflow water returns to the mill, underflow joins the sand tailings and goes to tailings pond. This pond is provided with a dam made of tailings, back of which the water containing slime is settled and pumped to the mill.

The mill feed has averaged about 2.6 per cent copper, which is higher by nearly 1 per cent than the ore treated by some of the other large copper mills, notably the Utah Copper Company's mills. Ratio of concentration is 20:1, and the extraction has been gradually climbing from 70 per cent to 78 per cent, and it is expected that the promised 80 per cent will be reached. The concentrate runs about 40 per cent copper, the mineral being nearly pure chalcocite with but a small amount of pyrite.

The mill consists of six sections, of which three are equipped with two 6-ft. Chile mills each. The fourth section is provided with three Burch high-speed fine rolls, and they will shortly be used for fine crushing for comparison with Chile mills. The Burch roll is provided with a fleeting movement by which one roll is moved laterally in front of the other in the direction of the axis of the roll so as to prevent grooving.

The mill was originally designed so that the six sections would have a capacity of 2000 tons per day, but it seems evident that this will reach 3000 tons per day. The power consumption is estimated by Mr. Channing to be about 1000 kw, or 1400 hp, for the treatment of 2000 tons per day in four sections. This will include the crushing and pumping, each of these operations being done for about 8 or 10 hours alternately. The mill itself will take about 720 kw, leaving 280 kw for crushing one-half of the day and for pumping the other half, so as to keep the load steadily at about 1000 kw.

Chlorination Abandoned.

It is generally understood that the chlorination plant of the Portland Gold Mining Company at Colorado Springs will be

remodeled for use as a cyanide plant. The company treats Cripple Creek ore, and is using a combination of the chlorination and cyanide processes. The raw ore is first ground and roasted, then treated by chlorination, the tailings being concentrated on tables. The table concentrates are shipped to smelters, and the table tailings classified into sand and slime. The sand is ground in tube mills in cyanide solution, and the gold solutions recovered by decantation and precipitated on zinc shavings. In addition to treating the chlorination tailings by cyanidation, the company has treated a considerable tonnage of its old accumulated tailings by the same method. The total tonnage daily treated by cyanide has been about 500 tons, equally divided between current and old tailings.

The decision to abandon the chlorination end of the treatment at the Portland will leave but one company still treating Cripple Creek ore by chlorination, viz., the United States Refining & Reduction Company. At this company's Standard Plant at Colorado Springs a combination of chlorination and cyanidation is used, on much the same order as at the Portland, with the exception that at the Standard chlorine is generated electrolytically instead of by the action of sulphuric acid on bleaching powder. Cyanide metallurgists may glean a bit of satisfaction from the announcement that chlorination is to be abandoned by one of the old companies, but it must be remembered that chlorination has had a long and honorable record in connection with Cripple Creek ores, and that cyanidation has only in recent years arrived at that state of perfection which would make it a favored rival. True, some have contended almost from the beginning that cyanide was a better agent than chlorine for treating these ores, but in the past there has been room for argument which cannot exist at the present time.

Colorado.

The complaint against the backward state of mining in Colorado, which has been heard for the last year or two, is gradually giving place to a more hopeful expression. It would be difficult to state concisely the reasons for the decline, but undoubtedly the wonderful interest in agriculture and irrigation, the restrictions against prospecting on the public domain and the high cost of equipping and handling low-grade properties have all had their effect. Now, however, the pendulum seems to be swinging back to mining, and there is a noticeable increase in interest. The commercial bodies are talking mines and mining, and public excursions are being conducted into the mining districts. Irrigation has had its sway and mining is again in a fair way to receive respectful attention from financial sources.

By way of publicly encouraging the prospector to open new fields the Denver Chamber of Commerce has raised a fund which is available to prospectors who wish to go out on a "grub stake" agreement. The idea is that a prospector will be outfitted and kept in the field during the season, and that any discoveries shall become the joint property of the Chamber and the prospector. The Chamber retains the major interest, and has the work in charge of a committee of its members. This committee consists of Messrs. James Doyle, Max Boehmer and Ernest L. Foster, all capable mining men and engineers who have had a long and active experience in mining in the State. Opinion is divided as to the practicability of the plan, but it is a worthy effort to revive interest in an industry which, though not comparable in its magnitude to that of 10 or 20 years ago, still holds a high rank in the principal industries of the State.

The Colorado School of Mines' thirty-seventh annual commencement occurred at Golden on May 26. Thirty-five young men were graduated and received degrees in mining and metallurgy. Among the theses presented as part requirement for graduation, two received prizes which are awarded annually. Mr. Roy F. Smith received the Brunton transit, and Messrs. E. J. Dittus and R. G. Bowman received the Stoiher prize of $150 for the best thesis in metallurgy. Their work was on the direct production of molybdenum steel in the electric furnace.

At the annual election of the faculty of the school there were some changes, though not as many as might have been expected in view of the persistent rumors to the effect that sweeping changes would be made. Prof. A. J. Hoskin and Assistant Prof. A. C. Smith, of the department of mining, and Assistant Prof. C. E. Smith, of the department of geology, resigned. Their places have not been filled.

The testing plant being erected at the school is under cover and will be completed before fall. The Legislature made an appropriation of $50,000 for this work and no delay will ensue if the sum proves sufficient.

The Iron and Steel Market.

The half year just ended has been an unsatisfactory one for the iron trade. The production of pig iron may be estimated at 11,750,000 tons, with a probable error of less than 250,000 tons. The three preceding half years showed heavier production, while the second half of 1905 and both halves of 1906 and 1907 showed heavier production. In other words, since the middle of 1905 production by half years has been heavier than in the half year just ended with the exception of 1908 and the first half of 1909.

This record is particularly unsatisfactory because there has been such a large increase in capacity in recent years. In the first half of 1905 there were made 11,163,175 tons of pig iron and in the second half 11,829,205 tons. These outputs represented substantially the total capacity available at the time, at any rate the capacity available with ordinary market prices, for prices were not unsatisfactory at the opening of that year and rose during the year. On the other hand, it was physically possible to make fully 5,000,000 tons more pig iron in the half year just closed than really was made, so that production was just 70 per cent of capacity.

Prices of open-market products were unsatisfactory in the half year. Pig iron, which at the opening of the year stood at a level affording only the most moderate profits to the most fortunate furnaces and represented an actual loss to a number of furnaces, particularly those in the Mahoning and Shenango valley, declined slightly, the average level being about 25 cents a ton lower on July 1 than on Jan. 1. Connellsville coke advanced and then declined, showing practically no net change, while scrap in most markets declined slightly.

The slight decline in the open-market commodities, considering that demand and production were so much below capacity, shows that the adjustment had already been made. Indeed, the year 1910 was one of steady and fairly rapid decline in prices in the open-market commodities.

In the finished steel trade there is a controlled market, and as a result a demand far below productive capacity has not produced a profitless market level. Apart from the profits involved in coke manufacture and ore production, the level has been such that there was a profit from purchased pig iron made into billets, and from purchased billets made into most finished steel products. Technically there were some advances in finished steel products early in the year, there being two of a dollar a ton each in wire products and one of 10 cents per box in tin plates. The wire advances, however, involved granting buyers large contracts at old prices, before the advances, while the advance in tin plates merely covered the advance in pig tin effected by the foreign syndicate, thus giving no benefit to the tin-plate mills. On the other hand, merchant steel bars dropped through the action of the Republic Iron & Steel Company on May 24 announcing a cut of $3 a ton in its price on steel bars, while as a result of a conference between all the important steel makers, except the Republic company, on May 29, reductions were made of $1 a ton in plates and shapes, $3

on blue-annealed sheets, $4 a ton on black and galvanized sheets and approximately $4 a ton on corrugated material. Technically, the average of all finished steel products dropped about a dollar a ton during the half year on account of these changes, but by reason of the peculiar conditions surrounding the wire and tin plate advances the effective decline was really more. Standard section rails are not included in this reckoning, as they stand in a class by themselves, the nominal price not having varied for ten years, while the quality has been materially improved.

Thus the finished steel trade suffered slightly reduced prices and materially reduced production, although its operations were still fairly profitable, and much more so than those of the merchant blast furnaces.

The trade encountered several severely adverse influences. On Feb. 22 the Interstate Commerce Commission handed down adverse decisions in the "advanced rate cases" when the general expectation had been that the railroads would be granted at least a material part of the advances in rates for which they had contended so long and earnestly, though perhaps not altogether wisely. These decisions cast a gloom upon the trade. The railroads at once curtailed purchases, and shortly afterwards a number of them suspended shipments on orders already placed.

The failure of the Sixty-first Congress to pass the Canadian reciprocity bill induced the President to call an extra session of the new Congress, which is Democratic in the House and not safely Republican in the Senate. The iron trade favors Canadian reciprocity, but became seriously disturbed over the prospect of tariff revision. May 15 the Supreme Court handed down its decision in the Standard Oil case. This was adverse to the "trust," but as the words "reasonable" and "undue" and the phrase "in the light of reason" were used in the decision there was a short-lived impression that a new and mild construction had been placed upon the Sherman law. The Tobacco decision a fortnight later, with more mature reflection upon the Standard Oil case, largely dispelled this impression, and the country became convinced that there may readily be found a number of business institutions in existence contrary to the Sherman law as interpreted.

The next adverse influence was the investigation of the United States Steel Corporation, intrusted to the Stanley committee by the House of Representatives.

In these adverse conditions the iron and steel trade has perhaps done fairly well in making the record it has in the half year. For the future, the second half of the year, the prospect is much brighter. No one can well maintain that the situation has not been quite thoroughly "liquidated." The buying trade has conducted itself with a deep impression of the adverse influences and is more likely to throw off the incubus than to yield further. It is the uniform experience that the influence of adverse developments lasts but a time, and the mental condition produced by a new condition wears away even though the condition itself may not be removed.

It is the general expectation in the trade that the second half of the year will show a much larger tonnage than the first half. There is no expectation of materially higher prices, although slight upturns in pig iron may occur in some districts. In finished steel the position is stable. While the Republic Iron & Steel Company withdrew formally from the councils of the steel manufacturers and took independent action in the matter of steel bar prices, it has not abandoned the general policy of co-operation. After the slight price cuts made in May most buyers adopted the view that other and deeper cuts would follow. This the steel makers insisted was not the case, and during the past month the expectation of buyers of lower prices has decreased. There has been a slight increase of buying of finished steel products, beginning about the second week in June, and with any further increase the stability of the finished steel price level may be regarded as reasonably assured.

Prices are quotable as follows:

Pig. iron, f.o.b. valley furnaces, 90 cents higher delivered Pittsburgh; Bessemer, $15; basic, $13 to $13.25; No. 2 foundry, $13.50 to $13.75; malleable, $13.25 to $13.50; gray forge, $13 to $13.25.

Unfinished steel, Bessemer and open-hearth billets, $21; sheet bars, $22, both f.o.b. maker's mill, Pittsburgh or Youngstown; wire rods, $29, Pittsburgh.

Finished steel products, f.o.b. Pittsburgh, except as noted:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, 1.35 cents for tank quality, ¼ in. and heavier.

Shapes, 1.35 cents for beams and channels, 3 in. to 15 in. inclusive, zees and angles 2 in. x 3 in. and larger.

Steel bars, 1.25 cents, base; iron bars, 1.35 cents, Pittsburgh; 1.20 cents to 1.25 cents Chicago.

Wire nails, $1.80, base, per keg; plain wire, 1.60 cents, base; painted barb wire, 1.80 cents; galvanized barb wire, 2.10 cents.

Black sheets, 28 gage, 2 cents; galvanized, 3 cents; blue annealed, 10 gage, 1.50 cents; painted corrugated roofing, 28 gage, $1.40 per square; galvanized, $2.55.

Tin plates, 100-lb. cokes, $3.70 per base box.

CORRESPONDENCE.

The Manufacture of "Air-Salpeter" a Well Paying Industry.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The fixation of atmospheric nitrogen for fertilizer purposes is growing rapidly in importance, and it is interesting to notice that the production of nitrates from the air for fertilizer purposes is indeed a lucrative undertaking.

Its renumerativeness is plainly shown in the report presented recently at a stockholders' meeting by the president of the Norske Hydroelektriska Aktie Sellskap, Norway. This company was formed several years ago to develop and operate the Birkeland and Eyde nitrogen fixation process. This process, which is well known, consisted in an arc spread out into the shape of a thin disk by means of a magnetic field.

Recently this process has been abandoned, and the Schönherr process, of the Badische Anilin Fabrik has been installed and is exclusively operated at Notodden, Norway, where the company's factory is located.

The Norske Hydroelektriska Aktie Sellskap is capitalized at about $2,000,000, of which $1,200,000 is preferred and the remaining $800,000 is common stock. During last year dividends amounting to 8 per cent on the preferred stock and 5 per cent on the common stock were paid.

The gross earnings during the year amounted to $1,200,000, of which $350,000 was net income. Half of this sum was put aside for use in various funds, ' $23,000 was distributed among employees, and the remainder, after the dividend was paid, was transferred to the credit of 1911, to take care of the running expenses for this year. There are certainly few new chemical industries that can show any better financial record.

The company could not begin to satisfy the demand for calcium nitrate, or "Norge Salpeter," as its product is known to the trade, and on this account they are developing, in conjunction with other financial interests, some 135,000 hp in different parts of Norway, all of these to be applied to the manufacture of nitrates from the atmosphere. Most of this power is already developed and will be taken into demand during this year for manufacturing purposes.

Extensive tests have been carried on during the past year with large furnaces taking 4000 hp, and these have been so successful that wherever possible this size will be used.

The Norwegian power conditions are somewhat different from those that most generally exist in this country, since power can be obtained there at a very low figure. This will naturally influence to some extent the financial showing.

Niagara Falls, South Ontario. John Woods Beckman.

The Problems of Tungsten Concentration.—I.

By H. C. Parmelee.

The mining and milling of tungsten ores has been a growing industry in Boulder County, Colorado, since 1900. In the early days of gold and silver mining in this district the tungsten minerals were unrecognized, and they received various names from the prospectors and miners. In general they were regarded as forms of iron, and were considered without value. In 1900, however, the first production of tungsten occurred, amounting to 40 tons of ore valued at $3,216. Since then the production of high-grade ore and concentrate has increased annually, except during a depression in 1907-8.

The output for 1909 amounted to 1100 tons of high-grade ore and concentrate, valued by the United States Geological Survey at $396,000. The total output in the United States for the same period was 1607 tons, valued at $559,500. From this it will be seen that the Boulder County production is by far the most important in this country, amounting in 1909 to about 70 per cent of the whole. In 1910 the output from Boulder was 1530 tons of high-grade ore and concentrate, valued by the State Bureau of Mines at $736,700. During the same period the output of all other metals in the county was valued at $161,589.

The tungsten minerals in the Boulder field are the three iron-manganese tungstates, ferberite, hübnerite and wolframite, and the calcium tungstate, scheelite. Ferberite predominates, but scheelite occurs in many mines, as is shown by the occurrence of lime in the analyses of ore and concentrate—there being no other source of lime worthy of consideration. These minerals are of high specific gravity, scheelite being 6 and the other three ranging from 7.1 to 7.5. They are generally soft and friable. The ferberite occurs in three rather well defined forms which, however, frequently grade into each other. These are: (1) Well-crystallized crusts and layers covering the surface of rock fragments and cementing them together; (2) massive granular ore occurring in more dense seams; and (3) the highly silicious ore in which ferberite occurs in fine grains scattered through the gangue.

In breaking ore in the mine the highest grade is sorted and shipped without preliminary milling. At present this grade must average 40 per cent or more tungstic acid. The lower grades, ranging from 3 per cent to 15 per cent tungstic acid, are concentrated by wet methods. Two grades of concentrate are recovered and shipped; the first contains 60 per cent or more tungstic acid, and the second ranges from 35 per cent to 40 per cent. The former is usually recovered as coarse concentrate and the latter as slime.

The stability of the tungsten industry depends, of course, on the uses for the metal; and as these are not extensive and the demand is limited and variable, there is always the possibility of intermittent operation of mines and mills. It is safe to say that were it not for the large companies operating in the Boulder district the industry would have a more variable career than it now enjoys. At the present writing there is a large tonnage of unused concentrates in the Eastern market, as well as at the mills, with no immediate demand in sight. The principal use of the metal is still confined to the manufacture of tungsten steel for high-speed tools. Minor quantities are used in dyeing textiles, glazing pottery, coloring glass and paper, making bronze powder and weighting silks. The manufacture of tungsten electric lamps creates a slight demand, but as 1 ton of high-grade concentrate will suffice for the manufacture of 15,000,000 to 18,000,000 lamps the quantity used is not important.

The reader who may be interested in other details of the Boulder County tungsten field, such as general and economic geology, mining, technology, ores, etc., is referred to the Colorado Geological Survey, which has published a complete report,[1] from which some of the foregoing information has been abstracted.

[1] "The Main Tungsten Area of Boulder County," R. D. George, State Geologist, Boulder, Col.

Location of Mills.

The town of Nederland is centrally located with respect to the tungsten mills of Boulder County. It is reached by stage, 18 miles from Boulder, or by railroad to Cardinal station and thence by stage for 2 miles. At Nederland is the mill of the Wolf Tongue Mining Company. At present this company is running its mill on a custom basis, sampling and paying for the tungstic acid content according to a graduated scale of prices. On ore ranging from 1 per cent to 30 per cent WO_3 the price paid per ton increases with each per cent tungstic acid contained, and above 30 per cent WO_3 the price per pound of tungstic acid also increases with each additional per cent. This system of outright purchase of ore has superseded the former custom of milling the ore at a certain price and returning the concentrate recovered and is generally more satisfactory. It insures better work on the part of the mill and should make a better return to the miner.

At Cardinal, 2 miles west of Nederland, is the Cardinal mill of the Primos Chemical Company. This plant for the past eighteen months has been retreating the tailings made during four years' previous run, when the methods of saving were not as efficient as at present. The Lakewood mill of the Primos Chemical Company is located at Lakewood, about 2½ miles northeast of Nederland. It is the newest and most attractive of the mills in the district. The company's own ore is treated at present, although in the past a great deal of custom ore has been treated.

Between Cardinal and Nederland is the mill of the Alton Mining & Milling Company, a small plant for company or custom work. On Beaver Creek, about 2 miles southeast of Nederland, are two small mills, one belonging to the Tungsten Mining & Milling Company, treating company ore, and the other owned by Smith & Ardourell, at present retreating the old tailings of the Wolf Tongue Mining Company. The Clarasdori mill on lower Middle Boulder Creek is idle. In addition to the regular milling there is an effort in a small way to roughly concentrate some of the old mine dumps, either by screening and saving the fine mineral or by hand jigging. Any material obtained in this way is sold to the mills.

Difficulties of Concentration.

The concentration of tungsten ore presents difficulties not encountered with other base minerals. This is particularly true with some of the methods now in vogue. It is believed, however, that all of these obstacles can be largely overcome by applying principles and methods that have proved successful in dressing other ores. Excessive sliming is one difficulty, the cause of which is well set forth in the report of the State geologist, previously mentioned.

"Ferberite is a rather soft mineral with one perfect cleavage, and generally one or more prominent partings. As a result the mineral is extremely friable even in the massive and massive-granular forms. Much of the ferberite was deposited as aggregates or loosely arranged crystals and crystal grains, forming crusts over the surfaces of rock fragments. One crust succeeded another until in many places the opening was filled. In other places the cavities remained open, but the walls were lined in the same manner. The crystal grains composing these crusts average not more than ⅛ in. in length and about 1/16 in. in diameter. In much of the ore where the crust is broken the crystal grains are easily separated from one another. When to this ready crumbling of the mass is added the extreme friability of the grains and crystals themselves it is easy to understand the excessive sliming. The finer parts of the slime form an almost impalpable mass which, when stirred in water, gives it an inky appearance, and the water remains turbid for ten days to two weeks. To save these slimes is one of the difficult problems with which the tungsten millman has to contend.

"Another serious problem is the successful treatment of the highly silicious ores. In almost all the tungsten mines there is a certain amount of highly silicious ore, consisting of minute

grains of ferberite in a matrix of chalcedonic quartz or hornstone. The percentage of ferberite varies widely. Outside of certain limited areas this type of ore is fortunately not very abundant and rarely amounts to 20 per cent of the product. Various methods of treatment have been tried, but none has given entirely satisfactory results. Even very fine crushing leaves a large part of the ferberite with particles of quartz attached. In concentrating, these grains consisting of quartz and ferberite will be disposed of according to their specific gravity. Those in which quartz is largely in excess will go with the tailings and ferberite will be lost, while those in which ferberite is abundant will go with the concentrates and help make a low-grade product."

Corroborative testimony on the difficulty of treating some of the ores is given by Mr. Victor G. Hills[2], who was connected with the Colorado Tungsten Corporation during 1905-7. This company treated custom ores, and at that time was doing the best work in the district. Careful records were kept and improvements were made wherever posssible. From a list of twenty-one tests on custom lots the following are given as showing the extremes and average in percentage saving:

Lot.	Mine.	Crude Ore, % WO_3	Concentration, Ratio.	Concentrates, % WO_3	Saving, %
4	D	8.07	8.69	62.17	88.90
3	C	6.07	181.81	40.37	5.03
15	E	9.51	8.93	62.59	75.11

In amount of tungstic acid in the crude ore Lot 3 compares favorably with average lots, but owing to its physical qualities it was a failure from a milling point of view. Mere assays would indicate that it was as good as any other ore, but milling tests proved its worthlessness.

The percentage of saving made in the Boulder County mills is largely a matter of conjecture in many instances. The smaller mills practise no regular control, and the larger concerns prefer to make no statement of their saving. Undoubtedly the efficiency is greater than it was four or five years ago, and as the best mill then operating made a customary saving of 75 per cent to 85 per cent, it is probable that the claims of 90 per cent saving may be realized in present practice. There is little loss in coarse tailing where that portion is allowed to go to waste, and the main loss is in the slime tailing. The statement is made that coarse tailing will assay as low as 0.1 per cent WO_3 and that slime tailings from the best mills contain as little as 0.30 per cent to 0.40 per cent WO_3.

The mills are of small tonnage capacity. Four of them can treat about 5 tons in twenty-four hours, one of them 24 tons and another 40 tons in the same period. With two exceptions they are built especially for tungsten work and are new. The two old mills are remodeled gold mills, and, while not as conveniently arranged as the newer plants, they are doing good work. A notable feature is the large table area required per ton of ore treated, which naturally reduces the tonnage capacity. In no instance is screen sizing being practised with a view to eliminating coarse material which might prove worthless. In view of the friable nature of the tungsten minerals it would seem that sizing should prove worthy of investigation, and certainly classification should be more widely practised.

The stamp is a time-honored crushing device, and finds favor in some of the tungsten mills. Some of the considerations just given, however, indicate that the stamp is out of place in crushing tungsten ore. The minerals are friable and of high specific gravity. When crushed in stamps such material tends to slime excessively, both by reason of natural properties and the fact that it remains too long in the mortar. When this preliminary step has been taken it follows that elaborate provision must be made for saving fine mineral.

Another fact that indicates the wrong use of the stamp is that much of the mineral occurs in a condition which would permit of its liberation by coarse crushing. But, as a matter of fact, a screen analysis of table concentrates made from stamped ore will show about 60 per cent passing a 150-mesh screen and almost 80 per cent finer than 100 mesh. Modern ore-dressing practice would indicate some such method as follows on the average coarse-grained ore: Crushing by jaw crusher and rolls; screen sizing and jigging; regrinding jig tailing, with subsequent classification and table concentration; regrinding table middling, followed by slime concentration on belt machines, and, finally, treatment on canvas if found necessary. Screening might be combined with the classification in such a way as to eliminate a portion of coarse stuff carrying no value.

There is but one mill in the district that is successfully using the jig, and the results are satisfactory to the operators. Other plants are considering the use of coarse concentration, and a second mill will shortly go into commission on this basis.

There are, therefore, two distinct methods of treatment in vogue in the Boulder tungsten field. The first involves comparatively fine crushing with concentration on tables, vanners and canvas plant—the latter being a very important feature and extensive in area. The second includes stage crushing, jigging, concentration on tables and vanners, with a canvas plant as a minor adjunct. The adherents of each method claim an excellent saving, which is not questioned, but the advocates of stage crushing and coarse concentration claim a less expensive plant and cheaper operation. A further claim in favor of coarse crushing is that a greater percentage of high-grade concentrate is recovered, and, since the value of the concentrate is in direct proportion to its grade, it would seem rational to recover as much mineral of high grade as possible and reduce the percentage of fine or low-grade concentrate.

But even with coarse crushing, and every effort made to reduce the quantity of slime, it is noticeable that considerable fine mineral is produced which may float away, even from the boxes receiving concentrates. One of the striking features of all the mills is the provision made for catching this fine floating mineral. Settling boxes of wood or cement are provided, and in one instance the water escaping from the settler passes through a cocoa matting filter. In this connection the idea suggests itself that the use of a coagulant might be beneficial. It is known that fine tungsten concentrate will not settle perfectly clear for several days. The simple expedient of adding a small quantity of lime water will show how quickly the solution will be clarified. On a laboratory scale the mineral thus treated will settle and leave a clear solution in a few minutes. Experiments on a large scale might show that the addition of lime solution at the time of fine grinding would be beneficial in coagulating the mineral and aiding in its recovery. Or, the addition of clear lime water to the concentrate settling boxes might prevent the loss of fine and floating mineral after it had been separated from the gangue. Other salts than lime, which are of proved value in coagulating colloid matter, might be experimented with to advantage.

Magnetic separation has been applied successfully to tungsten ores, but it is not in vogue in the Boulder field. The Boulder ores are remarkably free from sulphides of iron, lead and zinc, cassiterite, arsenopyrite, etc., which usually call for magnetic separation, and this probably accounts for the non-use of expensive magnetic treatment.

The Bingham & Garfield Railway, connecting the mines of the Utah Copper Company at Bingham with its mills at Garfield, Utah, is about 20 miles long. It is laid with 90-lb. rails, has a maximum grade of 2.5 per cent and a maximum curvature on two curves of 10 per cent. The total length of tunneling on the line is 4795 ft. in four tunnels of the following lengths: 682 ft., 754 ft., 1280 ft. and 2079 ft. There are three steel bridges of the following heights and lengths: One across Carr Fork 690 ft. long and maximum height of 190 ft., one across Markham Gulch 640 ft. long and 220 ft. high and the third across Dry Fork 670 ft. long and 185 ft. high. In addition to serving the company's own mines and millls it will haul ores from other mines at Bingham to the smelter at Garfield.

[2] *Proceedings*, Colorado Scientific Society, Vol. IX., pp. 135-158.

Desiccation of Air by Calcium Chloride.*

BY FELIX A. DAUBINE AND EUGEN V. ROY.

The authors have already described elsewhere the principles upon which this process is based, with special reference to the blowing of blast furnaces with dry air and their investigations have been conducted with the object of that application in view. On publicity being given to the process certain manufacturers approached the authors with the desire of putting it into practice. They yielded to these requests and undertook to make investigation of the appliances necessary for drying large volumes of air. An appliance of this nature has been installed at the Differdange Steelworks, Luxemburg, where it is now in normal working.

General Principles of the Process.

It is unnecessary to reproduce here the discussions, theories and calculations which were published in 1909 in the *Bulletin de la Société de l'Industrie Minérale* (Vol. 11, pp. 397-452, 479-541), and of which a summary has appeared in the *Journal of the Iron and Steel Institute* (1910, Vol. 1, p. 613). Nevertheless, it appears to be advisable to recapitulate briefly the general principles upon which the process is based.

It is inherent in the very nature of bodies that every substance the solutions which, when highly concentrated, possess very low vapor tensions, should be, for that very reason, a hygroscopic substance, and capable of serving for the extraction, up to a certain state of equilibrium, of the moisture contained in atmospheres possessing higher vapor tensions than those of the solutions in question. The word "solution" should, in this connection, be understood in its most general sense—a solution of a given strength of concentration may be solid or liquid, this being a matter of temperature—and it will be seen later that one of the characteristics of the process which the authors are about to describe is the maintenance of the solutions obtained in this solid "phase."

The more rapidly equilibrium is attained, the more suitable is the substance for playing its part as a desiccating agent. A highly hygroscopic body would, therefore, be one which, at a given degree of concentration, would possess very low vapor tensions, differing greatly from the vapor tensions of the air to be desiccated. This is precisely the case as regards chloride of calcium and certain of its hydrates, the very low vapor tensions of which permit of obtaining rapid desiccation. In this connection the properties of calcium chloride have been known for a very long period.

It should also be noted that every external influence which is capable of lowering the tensions of the hydrates favors, by this very fact, the desiccation; and as for any given hydrate these tensions vary with the temperature, and in the same proportion, it follows that the lowest temperature possible should always be selected. Further, as the vapor tensions increase in proportion as the molecules of water absorbed render the dilution of the hydrates greater, it becomes equally necessary always to maintain the solution at a sufficiently high degree of concentration.

Consideration of these principles, taken together with numerous experiments, has enabled the authors to determine the economic conditions of the evolution of calcium chloride during the course of its periods of hydration and regeneration. It has, to begin with, been found necessary to avoid making the concentration higher than $CaCl_2 + H_2O$ for the two following reasons: First, that the regeneration of hydrates of greater concentration than $CaCl_2 + 1\,H_2O$ requires a very large number of calories, so that, despite the very low vapor tensions of these hydrates, they are slow in coming into equilibrium for reasons which are not altogether apparent, but probably depend on a special physical condition. In other words, the $CaCl_2 +$ less than $1H_2O$ hydrates are "tardy" in their operation. Secondly, it has been found that during the absorption of water vapor a dilution of about $CaCl_2 + 8H_2O$ should not be exceeded. This hydrate becomes liquid at a temperature of 15°, whereas all hydrates of higher concentration remain solid at this temperature. Briefly, the details of the process are as follows:

A layer of broken calcium chloride, the smaller pieces at the bottom and the larger pieces at the top, rests on a sieve. Within the mass of the calcium chloride, and in its lower part, is submerged a spiral grating, consisting of pipes for the circulation of water. The volume of air to be dried is made to traverse the mass of calcium chloride from top to bottom, the draught being created by a fan. The absorption of the water by the calcium chloride leads to an evolution of heat, and this heat is carried away contemporaneously with its production by the water circulating within the spiral.

Without this indispensable precaution the temperature of the mass of calcium chloride would rise, the tensions of its hydrates would rise likewise, and the chloride mass would cease to act efficiently. The hydration of the calcium chloride should be arrested when the outside pellicle of the broken pieces commences to liquidate. If it were continued the mass of the calcium chloride would be completely melted, and it would be impossible to regenerate it on the spot.

Further, the fact of its being liquefied indicates that the calcium chloride has formed a hydrate whose dilution is already very considerable (more than eight molecules of water at 15°), and that, as a result, its very high vapor tensions render it unfit for effective desiccation.

At this point it becomes time to regenerate, and to do this it is only necessary to attain the temperatures between which that hydrate best known for this purpose is formed, namely, $CaCl_2 + 1H_2O$, which is produced between 175° and 235°. During the progress of this regeneration it is necessary gradually to raise the temperature in such a manner as constantly to maintain the hydrates in their solid phase.

Suppose, for example, that, after the passage of the air to be dried, the formula of the exterior hydrates is, at a given moment, $CaCl_2 + 4H_2O$, a hydrate which melts at about 40°. Owing to the circulation of the water, which has maintained the temperature of 15°, this hydrate has remained solid; if at the moment of regeneration the temperature be suddenly raised to above 40°, the pellicle corresponding to the above formula melts, breaks down, and is wasted.

It is necessary, therefore, to proceed progressively according to a law which has been recognized in practice and may be deduced in theory from the curve of the liquid and solid phases of the different hydrates of calcium chloride.*

It is necessary also in the course of this regeneration to be careful not to exceed the temperature of 235°, above which the "tardy" hydrate $CaCl_2 + 1H_2O$, of which mention has been made above, is formed.

It may be seen from the foregoing that it is possible to effect the regeneration of the calcium chloride with sources of heat of comparatively low grade, and that it is possible to employ with this object the waste fumes which occur plentifully in all metallurgical works.

The regeneration of the calcium chloride having been effected by this systematic warming, it is necessary in order to render it again fit for the complete absorption of water vapor to cool it completely. This cooling is quickly attained by a rapid circulation of water in the pipe system.

It goes without saying that if the air to be dried were directed on warm calcium chloride no desiccation would take place, because, with the chloride of calcium at such a temperature, the hydrates even if highly concentrated might possess very high vapor tensions. Hence the intermediate period of cooling after regeneration.

When the temperature has returned to that of the average

*A paper read before the Iron and Steel Institute on May 11; slightly abstracted.

Bulletin de la Société de l'Industrie Minérale, December, 1909, p. 497.

environment, the chloride of calcium has regained all its hygroscopic properties, and is capable of desiccating afresh the new volumes of air.

The above constitutes a qualitative summary of the different phenomena which takes place during the evolution of the desiccating substances. It may now be asked, from the commercial point of view, what figures are involved in regard to the relative volumes of air to be dried and of the drying body; to the rate of passage of this air through the body; to the efficiency of the desiccation obtained, and to the amounts of heat to be employed for the regeneration and the time required for the cooling. The replies to these different questions will show at once whether the process is one of commercial application or not. The figures are as follows:

A mass of calcium chloride of 240 kg spread out in a layer of 24 cm in depth, on a square meter of surface, will desiccate 300 cu. m of air, percolating it from above downward per hour for four hours under average conditions of 15 grams of moisture per cubic meter. Without the water circulation the temperature of the chloride of calcium mass would gradually rise to the vicinity of about 40°; the circulation of the water prevents the elevation of the temperature beyond a few degrees, such, for example, as 4 or 5.

In order to regenerate the calcium chloride from its hydrates, calculation shows that it is necessary to expend about 7,500 calories per kilogram of water deposited. In practice it suffices to circulate from below upward—that is, in the opposite direction to the air current—warm air or fumes devoid of dust at gradually increasing temperatures from about 30° to 200° during a period equal to half of the length of time of the passage of the air current—that is, for the example given above for two hours.

Naturally, the circulation of the water is arrested during the circulation of the fumes or of the warm air used for the regeneration. The cooling, owing to the water circulation, does not take any longer time than that of the regeneration. In order to be on the safe side, the authors have always allowed that, given three stages of the evolution, it is necessary to have a triplicate apparatus, so that at any given moment there should always be:

An apparatus at work (desiccation of the air);

An apparatus in regeneration (the circulation of the fumes or of warm air); and

An apparatus in process of cooling.

With regard to the percentages of desiccation obtained they have been deduced already in the calculation. The table below gives the vapor tensions of various compounds of calcium chloride, with water at a temperature of 15°, showing the lower limits which it is possible to attain:

Compounds.	Temperatures.	Pressures in Millimeters of Mercury.	Relative Tensions.
$CaCl_2 + 4$ to $6H_2O$.....	15.5	2.47	0.192
$CaCl_2 + 2$ to $4H_2O$.....	15.0	1.47	0.116
$CaCl_2 + 1$ to $2H_2O$.....	15.0	0.64	0.050

What is called relative tension in the fourth column is nothing less than the state of hygrometric equilibrium which would be attained in a case of repose with indefinite masses of chloride of calcium in presence of finite volumes of air.

Given that the instance under consideration involves a movement of volumes of air and of finite masses of the desiccating substance, these figures themselves can never be reached, although it is always possible to approximate to them. Indeed, experiment has shown that it is possible in a well-watched and carefully conducted operation to attain a degree of desiccation such that the hygrometric condition of the desiccated air will be for the average period of passage but 10 per cent to 15 per cent of its saturation at a temperature of 15°, which corresponds to tensions of water vapor of 1.2 grams to 1.9 grams per cubic meter of air. The foregoing figures have clearly demonstrated in the preliminary experiments that the commercial desiccation of air is possible by such a process. At the present moment it is an accomplished fact.

Earlier Process of the Desiccation of Air by Calcium Chloride.

Fryer Process.—The earliest process that can be mentioned is that of Fryer at Coleford, which made its appearance in 1880. It involved employing the chloride in the most simple form by making the air pass over broken pieces of calcium chloride, collecting the water which settled and regenerating partly in special boilers, and carrying it subsequently to complete igneous fusion.

The inconveniences of this process are at once apparent. If it be sought to liquefy the calcium chloride, there are at once obtained extremely high dilutions, with very high vapor tensions. The tension of the equilibrium itself rises, and finally there is left, in the air to be dried, more moisture than is desired.

This is, however, not the chief inconvenience. The Fryer process allowed for no cooling of the mass during the desiccation, so that the temperature would rise very rapidly, and the calcium chloride became gradually inefficacious.

Finally, the calcium chloride being regenerated in special appliances, a whole series of manipulations had to be provided for—the withdrawal of the liquids, concentration, pouring, solidification, cooling and breaking. The latter operations, taking place on the cold chloride of calcium, are particularly difficult, as it is necessary to take special precautions, and to operate out of contact with air. Indeed, chloride of calcium being a waste product and having in itself no value, it might be affirmed that a chloride thus regenerated would cost practically as much as calcium chloride prepared for the first time. It may be pointed out finally that Fryer, not being aware of the peculiarities of the "tardy" hydrate $CaCl_2 + O$ to $1H_2O$, carried regeneration as far as igneous fusion, which is a disadvantage from the point of view of the activity of the chloride, and requires the expenditure of a considerable amount of heat. The authors are not aware that this process was ever applied commercially.

Cremer Process.—This process dates from 1885, and is illustrated in Fig. 1, in which A is a desiccating chamber, in which are arranged the shelves upon which are placed the materials used for the desiccation, sulphuric acid, calcium chloride, or other analagous substances.

B is the air inlet; B^2 is the outlet tube. Inlet B, instead of being connected with the open air, may be connected with the discharge vent of a fan, in which case the air may be forced into B instead of being aspirated from B^2.

Regeneration can take place on the spot. For this purpose the shelves have a double bottom for the circulation of hot air, or of waste gases to heat the calcium chloride placed upon them.

This process certainly evinces considerable progress, inasmuch as it involves regeneration of the moistened calcium chloride on the spot. It is still, however, exceedingly incomplete. Cremer made the air pass above the layer of calcium chloride instead of making it traverse the layer. The contact is thus less intimate. There is formed in the upper portion an exceedingly moistened layer, the tensions of which gradually increase, as much because of the increasing dilution as because Cremer had not foreseen the necessity of cooling.

Cremer did not indicate either the temperatures of the necessary regeneration, nor the limits within which it was necessary to keep; he provided for the heating of the mass by double-bottomed chambers, which was bad, as the particles in contact absorb all the heat and have a tendency to melt, while the topmost particles are insufficiently heated. Regeneration, in order to be certain, as the authors have shown, requires to be gradual and to be easily regulated. This can be readily attained by the circulation of hot gases through the mass, as the authors have provided for.

Elsner Process.—This process is recent, and dates from 1906. Furnace gases are led on the chloride of calcium, undergoing regeneration in a lower drum. The chloride of calcium, which is assumed to have been regenerated, is raised by means of a mechanical device to an upper drum, in which the air to be dried circulates. Here again no cooling is provided, for although the method of regeneration adopted leads to the supposition that Mr. Elsner wishes to maintain the solid phase

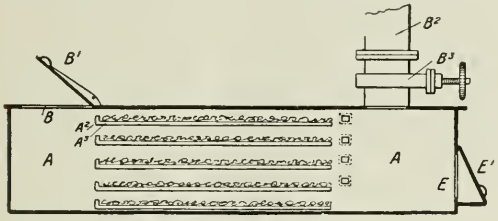


FIG. 1.—CREMER PROCESS OF DESICCATION OF AIR.

for his solutions, he has, nevertheless, neglected the safest method to attain that end—namely, the cooling during the hydration of the chloride.

To any one who has used chloride of calcium, and made it undergo many regenerations, it is a well-known fact that the lumps not only become regular sponges, which are more and more fitted to absorb moisture, but also that their cohesion becomes destroyed little by little, and that these lumps, which very readily resist the passage of air or of gases, pulverize at the slightest shock.

It is very probable that in the Elsner process it would become impossible to maintain the chloride of calcium in lumps, and that the particles would become smaller and smaller and more and more easily liquefiable.

Finally, Mr. Elsner charges his warm regenerated chloride directly into the topmost drum without first cooling it, which is absurd.

Harbord Process.—There are several Harbord processes, dating from 1907, 1908 and 1909. These processes are highly ingenious. Inert substances, such as heat, pumice-stone, all, generally speaking, porous bodies, are impregnated with a concentrated solution of calcium chloride. Regeneration likewise takes place on the spot. The principal object pursued by Mr. Harbord appears to be the retention of such chloride of calcium as might have become liquefied, and the prevention of its flow.

Granting that Mr. Harbord operates with solutions concentrated at high temperatures—above 200°, for instance—the particles retained by impregnation may well be equally active as the peculiar masses which intervene in the process described by the authors. Either one condition or the other must, however, obtain: either the volumes of air to be desiccated are large in relation to the small mass of the chloride of calcium impregnating the inert bodies, in which case there are quickly obtained liquid solutions of high vapor tension, and consequently but little effective, or else the masses of the chloride are, in relation to the volumes of air, in the proportions which the authors have themselves calculated, and the necessity arises for having appliances of enormous dimensions, the quantity of chloride being small in proportion to the quantity of inert matter.

Mr. Harbord has likewise made no provision for cooling, and this cooling would, besides, be difficult to secure, inasmuch as it would have to be carried out simultaneously on the inert bodies which serve to support the active agents. In the last resources, the presence of large masses of foreign matters would be the cause of a heavy expenditure of heat during regeneration, as it would be necessary to raise their temperature as well to the temperature of regeneration.

It would equally be a cause of embarrassment during the intermediary cooling down between regeneration and fresh hydration, as such bodies, which possess, generally speaking, a low calorific capacity, which makes them very slow in heating and causes the absorption of much heat, also require a very lengthy period during which to cool down, which would prevent the calcium chloride being put rapidly into use again.

Mr. Harbord has provided for the use of chloride of iron to prevent the formation of the oxy-chloride. This precaution is superfluous. With chloride of calcium, which had been in daily use during three months, and had undergone over a hundred successive regenerations, the insoluble residue was exactly the same as that of the same calcium chloride on its delivery from the factory.

Other patents are in existence. The bulk of those to which the authors have been able to obtain access resemble one or other of the systems referred to above.

Operation of Process at the Differdange Works.

The problem at the Differdange Works was to desiccate the whole of the air required for the blowing of the blast-furnace of 150 tons per twenty-four hours, in order to realize for this blast-furnace the economies described by Mr. Gayley in his experiments of 1904 and in those he had undertaken since.

The appliances were designed by the firm of Paul Wurth & Co., Luxemburg. Drawings showing two forms of apparatus are given in Figs. 2 to 8.

In the first design the blast is introduced by a central well and distributed in layers by means of the openings leading to different superposed reservoirs. On emerging from the latter it is collected in an annular chamber, where it is led to the place where it is to be utilized.

In a second form of appliance, on the contrary, the blast arrives by way of the annular chamber and is collected in the

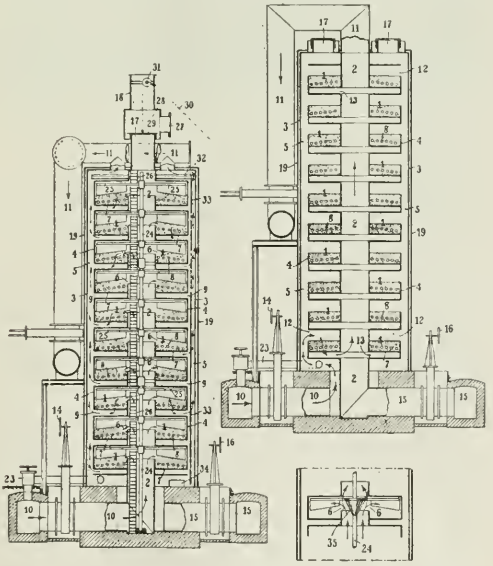


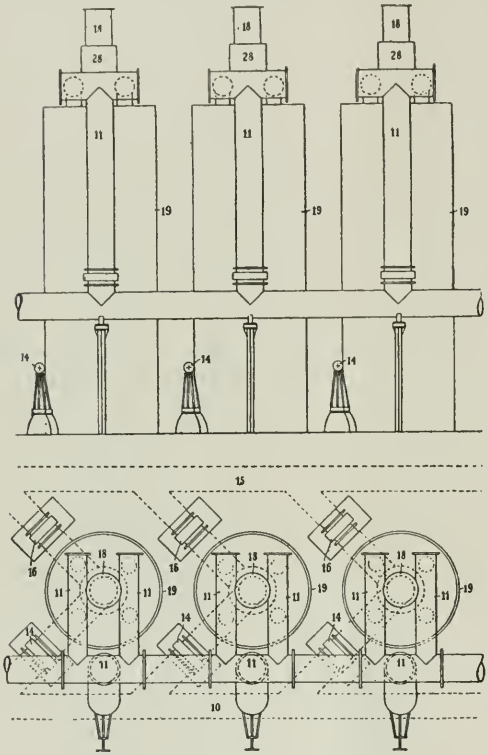
FIG. 2, 3, 8.—DETAILS OF PROCESS OF AIR DESICCATION.

central well after traversing the layers of chloride of calcium and is led thence to the place of utilization.

In order to be able to regenerate the chloride of calcium on the spot, the central well (or the annular chamber) is connected with a second pipe, admitting air or hot gases. The arrangement of the fans is such that it is possible, at will, either to pass the air to be dried, or the gases which are to serve for the regeneration, through the apparatus, or else to isolate it completely during the cooling down.

Figs. 2 to 8 show the two forms of design. Fig. 2 shows, diagrammatically and in longitudinal section, the first type of design, and Fig. 3 the second type. Figs. 4 and 5 show in elevation and in plan an installation composed of several drying apparatus; while Figs 6 and 7 show, in longitudinal section and in plan, the details of the spiral cooling arrangement. Fig. 8 shows details of the means for distributing the air current.

According to Figs. 2 and 3, the apparatus, which has a cylin-



FIGS. 4 AND 5.—DETAILS OF PROCESS OF AIR DESICCATION.

drical shape, is composed of a certain number of compartments identical with each other (1), a central shaft (2), a cover (3), which forms, with the external shell (4), a number of compartments, and the annular chamber (5).

Each compartment (1) has a circular inlet orifice (6) and a grid (7), on which the chloride of calcium rests. An outlet (9) is provided below the grid.

Towards the end of the desiccation the air is drawn by means of the fan across the main lead (10) into the central shaft (2), and distributed to all the compartments. It traverses the chloride of calcium from above downwards, depositing its moisture; it then escapes into the annular chamber and reaches the conduit pipe (11), which takes it to the place of utilization.

As is well known, the chloride of calcium warms during the desiccation, and this must be rectified by the cooling. It is with this object that water is made to circulate in the spirals provided in each compartment. These spirals are arranged to have a certain inclination to provide for their being rapidly emptied. The joints are effected in a special box securing a water-tight separation for the chloride of calcium, with a discharge vent in case the joint becomes defective. The chloride is thus completely protected against such an event occurring.

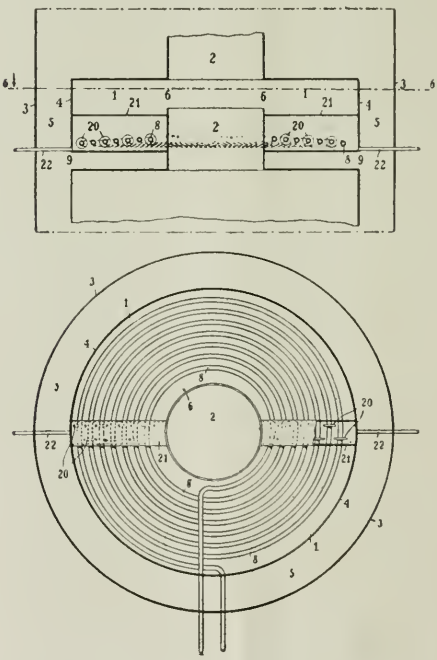
When the chloride of calcium has absorbed the requisite amount of moisture, the apparatus is cut off from the inlet and outlet of air by the slide valve (14), and communication is made by means of the slide valve (16) with the main (15) which conducts air or warm gases.

As the temperature should only be increased gradually, the fan aspirates cold air at the same time as it takes in the gases, to allow the temperature being regulated. By diminishing the amount of air admitted and increasing that of gas the temperature is gradually raised to 235°.

When the calcium chloride has been regenerated the apparatus is first cooled by means of a cold current of air entering the chamber (5) by the conduit pipe (23) and escaping by the outlet orifice (17) and then water is introduced in the spiral cooling coils until the temperature has been sufficiently lowered. The apparatus is then ready to desiccate the air-blast anew.

In the installation shown in Figs. 4 and 5 three apparatus have been assumed to be employed, of which one dries the blast, while in the second the chloride of calcium is being regenerated and the third is undergoing cooling. During the period of the passage of the blast, or at the end of the period of cooling, it is possible to obtain access to the appliance, either by the central shaft by means of the ladder shown in Fig. 2 or to the annular chamber by the ladder (33), running on rails at the top (32).

In the application of the process at Differdange, which is shown on Fig. 9, the conditions require the drying of 30,000



FIGS. 6 AND 7.—DETAILS OF PROCESS OF AIR DESICCATION.

cubic meters of air hourly. The apparatus, constructed in accordance with Figs. 2 to 8, possess the following features:

Total area presented to the passage of the blast, 100 square meters per apparatus.

Number of compartments, 10.

Depth of the layer of calcium chloride in each compartment, 24 centimeters.

Apparent density of the chloride of calcium, 1.0.

Weight of chloride of calcium contained in each apparatus, 24,000 kilograms.

Weight of chloride of calcium contained in all three appliances, 72,000 kilograms.

Cooling surface of the spirals in each apparatus, 170 square meters.

These appliances have been designed to work in the most unfavorable conditions—that is to say, to remove, during the summer months, 15 grams of moisture per cubic meter of air during a period of four hours.

At the time of writing this paper the appliances have been working normally for six weeks, but as the season is the end of winter and as the moisture in the air is not very large, it has been found unnecessary to make as many reversals as were contemplated.

Each apparatus receives the blast for 6 to 8 hours. The air which contains 6 to 8 grams of moisture before its passage only contains 1 to 1.5 grams per cubic meter on emerging from the apparatus, and this figure remains practically constant from the commencement to the conclusion of the period.

Regeneration requires four hours for its completion and is carried out by means of the waste smoke gases from boilers and from Cowper stoves. These gases, cleaned to the extent of 0.4 gram per cubic meter, pass directly through the mass of chloride. The temperature is regulated at 30° to commence with, and thereafter gradually raised in conformity with a certain ascertained law up to about 200°. In the summer the temperature will be carried to 275°.

Cooling takes three hours.

It is interesting to compare these results of drying the blast with those which are capable of being given by the freezing method used by Mr. Gayley.

The authors reproduce below a table of the vapor tensions of ice and the amounts of moisture which saturate a kilogram of dry air at temperatures below 0°:

Temperatures. Degrees.	Vapor Tensions in Millimeters of Mercury. Per Cent.	Weight of Water Vapor in Grams in Saturated Air per Kilogram of Dry Air. Per Cent.
0	4.60	3.80
—1	4.27	3.50
—2	3.95	3.20
—3	3.66	3.00
—4	3.39	2.80
—5	3.13	2.60
—6	2.89	2.40
—7	2.67	2.20
—8	2.46	2.00
—9	2.27	1.85
—10	2.09	1.70

From the above table may be seen to what temperature the air must be cooled to obtain results identical with those obtained at the Differdange Steel Works.

At 15° an amount of humidity of 1.5 gram per cubic meter corresponds with the content of 1.22 gram per kilogram of dry air. This figure, as may be seen, is well within the limits of the foregoing table. To obtain this result, calculation shows that it is necessary to cool the air effectively to —15°, which requires even considerably lower temperatures in the refrigerating tubes.

This shows the advantage which the employment of calcium chloride has over that of refrigerating machines. While with the latter system cooling to —5° leaves a minimum of 2.60 grams per kilogram of dry air, the use of the calcium chloride regenerated under the form $CaCl_2 + 1$ to $2H_2O$ and maintained in sufficient excess at 15° allows of a degree of moisture as low as 0.5 gram per kilogram of dry air being attained.

During the winter months, during which the refrigerating system is practically inoperative, particularly in Continental European localities, chloride of calcium allows of exceedingly low figures being attained—that is, distinctly less than 1 gram per cubic meter of air.

With regard to the blowing tests on dry air now being undertaken at the Differdange blast furnace it is impossible for the authors to give any details at present. The installation is too recent for the true value of the economies realized to be stated, but it is now quite established that the manufacture of dry air need present no difficulty whatever.

The installation has cost a little less than one-quarter of what would have been the cost of an installation for desiccation by means of refrigerating machines. One man for the day shift and one for the night shift are sufficient to handle the apparatus, which is of the most simple description. The expenses of working are thus greatly reduced.

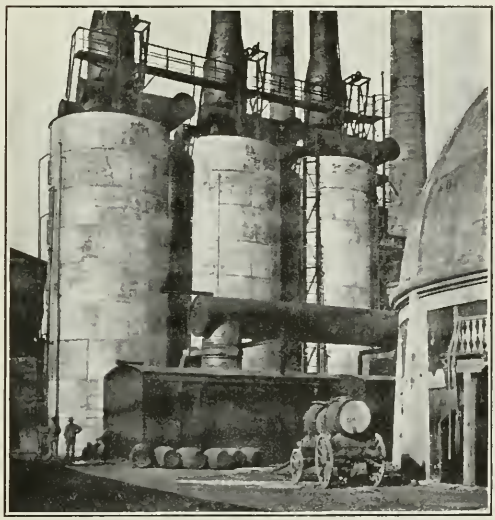


FIG. 9.—AIR DRYING BY CALCIUM CARBIDE AT THE BLAST FURNACE PLANT OF DIFFERDANGE STEEL WORKS IN LUXEMBURG.

The authors state, in conclusion, that similar apparatus are under investigation, not only for other metallurgical works, but also with a view to their application in other industries that require drying operations carried out at low temperatures, as, for example, for the desiccation of chemical or pharmaceutical products, india rubber, resins, gums, gelatin, albumen, glue, aniline colors and various organic bodies. It may thus be seen that there is a vast field open for the commercial production of dry air.

Aubone, France.

The Detection of Willemite by Ultra-Violet Light is used at Franklin Furnace by the New Jersey Zinc Company for the examination of tailings. It has been found that ultra-violet rays cause willemite to show a peculiar luminous glow, which enables one to examine in broad daylight tailings containing this mineral. The method used was described by Mr. George C. Stone, chairman of the New York Section of the Mining and Metallurgical Society of America, as follows: The apparatus consists of an arrangement to produce an arc between iron electrodes, which is rich in ultra-violet rays. Every barrowful of tailings is examined by the foreman, and if he believes that the material contains more than 2 per cent zinc it is returned for retreatment. The mill foreman becomes quite expert in examining material in this manner, but he is checked by assays. The discovery that willemite could be detected in this manner was made in 1896 and is due to Dr. Baskerville and Dr. Kunz.

The Melting Point in Its Relation to Alloying Capacity.

By Dr. Hans Goldschmidt.

The opinion is often expressed that if a metal has a high melting point it is difficult to produce with it an alloy of uniform composition. When scrutinized, however, this opinion is found to be unwarranted. When I first introduced the

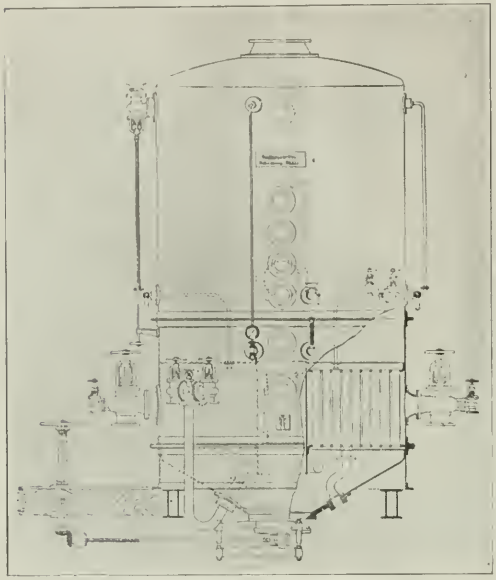


FIG. 36.—GERMAN VACUUM PAN WITH SHORT VERTICAL HEATING TUBES.

aluminothermic metals, such as chromium, manganese, ferrotitanium, etc., into commercial practice, I had to overcome considerable trouble, since metallurgists would tell me that the melting point of these carbon-free metals or alloys was higher than that of the usual brands then on the market which contained carbon, and that the higher melting point of my metals would interfere with their use.

Meanwhile, practice has long shown that carbon-free chromium, in spite of its melting point of 1550° C. (2822° Fahr.) alloys easily with molten steel. The same is true for carbon-free manganese with a melting point of 1245° C. (2273° Fahr.) and this manganese alloys especially easily with copper; in fact, more easily than manganese containing carbon.

In recent years titanium is often added to steel. Ferrotitanium made for this purpose by the aluminothermic process has a melting point of 1330° C. (2426° Fahr.) and alloys easily with liquid steel.

In the preceding paragraphs I have stated the melting points although they are not of practical but only of scientific interest. The following notes will make this clear:

An alloy is nothing but a solution. For instance, an alloy of zinc and copper is called a "solid solution." Zinc has a melting point of 419° C. (786° Fahr.), copper a melting point of 1098° C. (2008° Fahr.). Now let us try to alloy lead instead of zinc with copper; lead has a melting point of 330° C. (626° Fahr.), hence much lower than zinc, yet our efforts of producing an alloy of copper and lead will not be successful. It is simply a fact that lead does not dissolve in copper.

Among the most refractory metals known we have platinum with a melting point of 1775° C. (3227° Fahr.). Yet platinum alloys with tin and lead at a temperature of the melting points of these two metals. Many a young chemist has been sorry to make the experience that when by accident a small amount of these metals got into his beautiful new platinum crucible a hole was formed when it was heated. The fact is simply that the high melting point of platinum does not protect the crucible from the dissolving action of the metals mentioned.

Since, as stated before, an alloy of two metals is nothing else but a solution of one in the other, we are justified from a scientific standpoint to consider it as analogous to the solution of common salt in water. We know that common salt dissolves easily in water; the melting point of common salt, which is at 820° C. (1508° Fahr.) is without any significance for the process of solution.

From what has been said it is clear that for their practical purposes metallurgists have simply to take care that they employ metals or alloys which will dissolve. And it may become necessary to use intermediary alloys which dissolve more easily in the metallic bath. For instance, a high-percentage manganese-copper alloy is made from pure manganese, while instead of pure titanium ferrotitanium is used.

This carbon-free ferrotitanium, containing 25 per cent of titanium, is used now quite extensively as an addition to steel for making rails and rolls. This ferrotitanium made by the aluminothermic process dissolves very easily and uniformly in the molten steel. It has a small intentional content of aluminium, as this increases its solubility.

The experience of metallurgists has shown that as a rule carbon-free metals dissolve more easily than metals containing

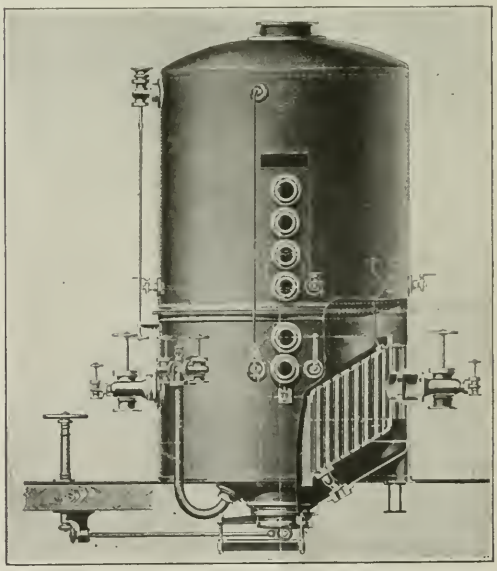


FIG. 37.—GERMAN VACUUM PAN, WITH INCLINED REMOVABLE HEATING SYSTEM AND VERTICAL TUBES.

carbon, since the latter contain carbides—that is, chemical compounds with carbon. It is true that the melting point of the carbides is lower, but as was shown before, this has nothing to do with the solubility, or what is the same, the alloying capacity. The situation is practically the same with ferrovanadium as it is with titanium. For titanium and vanadium it is advantageous to use an intermediary alloy, because only very small quantities of these metals are used.

When a metallurgist has the choice between metals made by different processes the main point of importance is to use starting materials of absolutely uniform composition. Only if he is sure that they will not vary in composition he can get good reliable results. This requirement is fulfilled for metals made by the aluminothermic process on account of their method of production, since they are not made in small quantities, drop by drop, as is the case in the arc or melting zone of electric furnaces, but are produced from a uniformly fused charge. Uniform products are obtained as a result of the uniform composition of the thermit mixture. On the other hand it is very difficult to maintain uniformly the melting zone in an electric furnace, and for this reason it is practically impossible to obtain an absolutely uniform composition in electric furnace alloys, such as ferrochromium, ferrotitanium and ferrotungsten, as especially the carbon content varies considerably.

Essen-Ruhr, Germany.

Evaporators and Vacuum Pans.

By B. Viola, M.E.,
Member American Society of Mechanical Engineers.
(*Continued from page* 311.)

Already in many German vacuum pans short heating pipes have been adopted, but if so arranged the belt or frame of the pipes are generally inclined and the heating tubes are in a

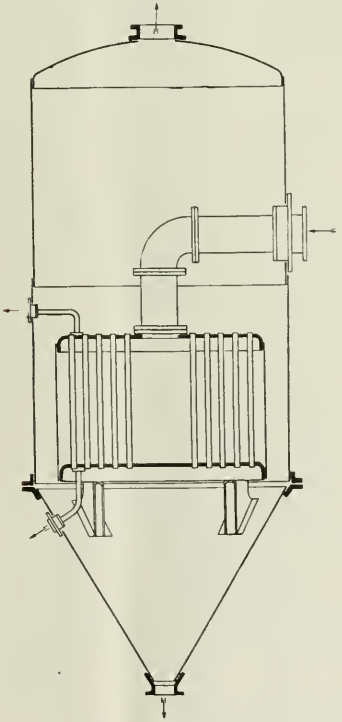


FIG. 37*a*.—PASSBURG VACUUM PAN, WITH REMOVABLE HEATING CHEST.

vertical position. These heating systems can be made steady or removable; the last ones being only hooked in and can be exchanged any time (Figs. 36 and 37). An American-made vacuum pan with removable heating chest can be seen in Fig. 37a of the Passburg system. This is another specialty, in so far as the heating steam does not enter on the side of the heating tubes, but on the top of the heating chest, which gives the steam a better opportunity to heat the pipe surfaces more equally and by that a better evaporation can be effected. The main point is, however, that the steam pipe carrying the steam into the steam chest should always be immersed in the liquor.

The most highly recommended shape for the inclined heat-

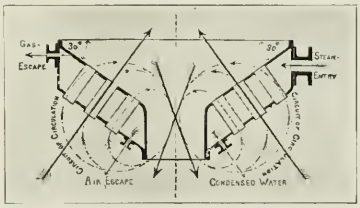


FIG. 38.—PRINCIPAL OF THE INCLINED HEATING TUBE SYSTEM.

ing tube system is shown in Figs. 38 and 39, and is the foundation of Stade's express vacuum pan. This most modern pan consists of one or more heating systems which are arranged in the following manner. The calandria consists of conical parallel plates, which are fixed like a double funnel and contain a central circulation pipe inside and a cylindrical ring outside. The heating tubes are fixed at such an angle that their central lines form a kind of parallel or central radiatiform (Fig. 38).

The steam enters at the upper part of the calandria, the condensed water escapes at the lowest point, while for air and gas separate pipes are arranged to insure highest efficiency of the system. The tubes are short and wide. They deliver even the most concentrated masses up to 5 per cent water to the top. The tubes being set inclined, the masses have to get out sideways and are forced to circulate sideways in or outside of the system.

There are only about three or four supports required for one system, consequently there are no complicated stays to hinder

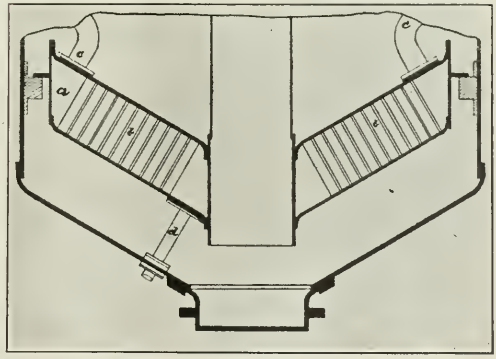


FIG. 39.—REMOVABLE INCLINED HEATING-TUBE SYSTEM.

the full circulation which is bound to take place in a methodical forced system. (Fig. 39.)

One of the chief points of the express vacuum for practical engineers is the easy access to all parts of the system. The tubes can be reached from outside and inside, so that in case of any leakage repairs can be effected in the most simple manner.

The fundamental principle on which the circulation system in general use now (in combination with short inclined express tubes) is based upon may be explained as follows:

The construction consists of a double or false shell fixed at certain intervals over the calandria (Fig. 40). The shells separate the large interior space of the pans in such a manner that a forced but natural circulation takes place from inside up (through the inclined express tube system) to outside down (the space between false shell and pan shell). All the superheat taken up through the tubes goes in form of vapor straight to the condenser and the so cooled masses possess again the necessary low temperature to get the highest transmission of heat when coming again into contact with the calandria tubes.

The natural law states that in two communicating tubes filled with liquids of different specific gravities the levels must be of different height. Now the express circulating system contains inside above the express heating system a mass-cuite mixed with the vapor generated. Outside, however, between the inner and the outer shells the mass-cuite remains free of any vapor as by action of the vacuum pump the vapor generating from the masses escapes at once to the surface and goes to the condenser. Consequently the specific gravity of the mass-cuite outside is higher than the specific gravity of the masses above the calandria. This causes the inside masses to overflow over the inner shells.

In addition to this circulation, however, the action of the heating tubes of the calandria has to be taken into account. These tubes serve as a powerful suction pump, drawing the mass-cuite from the bottom and delivering it through the express tubes to the inside shells system, helping in this way to form a strong circulation in the most simple way, doing away with all stirring and all movable parts inside the pan.

All kinds of steam can be used with the new express circulation pan, and a pressure of 7½ lbs. is sufficient even for large pans with high heads of mass-cuite. With steam of 10 lbs. pressure careful experiments with a large pan gave an evaporation of 15 lbs. water per square foot heating surface.

The materials used for the construction of vacuum pans are copper, iron, lead or lead-lined sheet iron. The heating coils or tubes are copper in most cases on account of its better heat-conducting power, or lead and lead-lined copper, according to the materials which has to be concentrated.

Pans made of copper are usually round, while those of iron are cylindrical or half cylindrical with flat bottoms.

Each vacuum pan, similar to the last effect of the evaporators, is provided with a foam catcher, juice catcher, and has an amply large diameter connection pipe with the condenser.

For the evaporation of certain acid liquors, lead or lead-lined sheet iron is

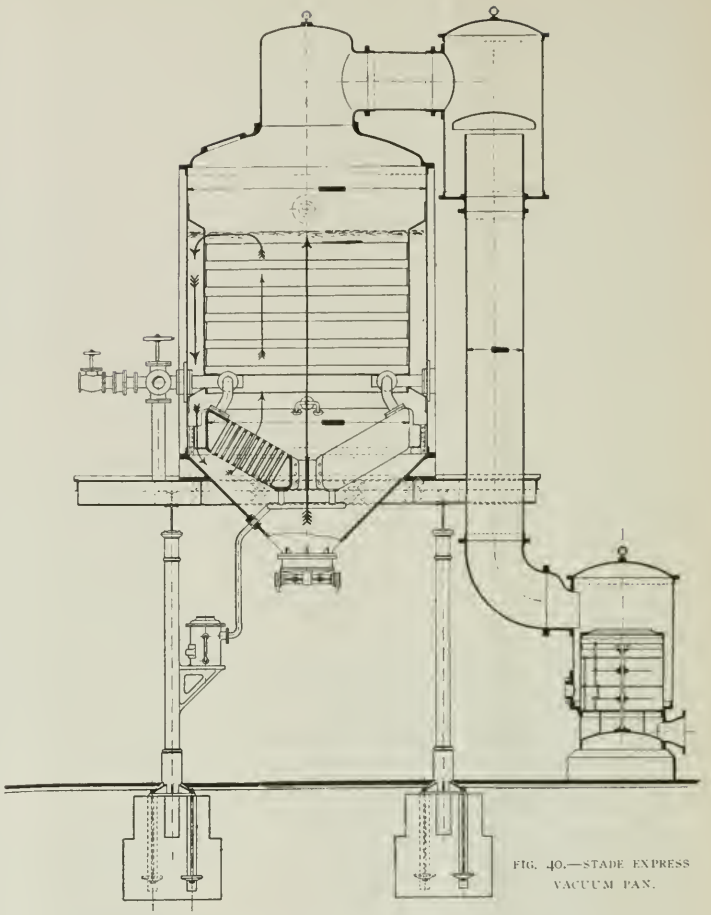
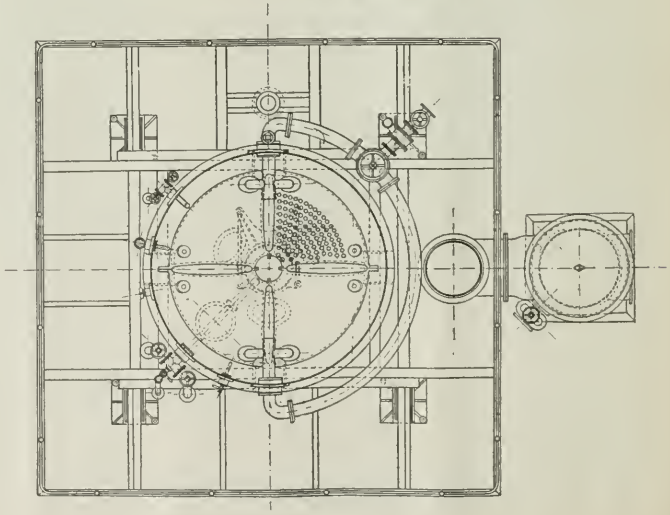


FIG. 40.—STADE EXPRESS VACUUM PAN.



the material mostly used for vacuum pans. If lead is used, then on account of the thickness, the whole pan has to be made of lead casting. Naturally the heating coils were made of lead also, in spite of their poor heating capacity, because no other material can stand certain acids. An entire lead vacuum pan with lead heating coils can be seen in Fig. 41.

Lead vacuum pans had neither foam nor juice catchers, and so it proved necessary to catch the acid particles, which were carried along with the vapors. For this purpose cylindrical cast-iron vessels or so-called overflows were connected with the vapor discharge of the vacuum pans, so that the acid particles could settle there and only the vapors passed on to the condenser. In order that the iron should not be destroyed by the acid particles some neutralizing material had to be used. A complete equipment of such a plant is shown in Fig. 42 vacuum pan, overflow and condenser. However, these overflows were destroyed very quickly and after a while were given up entirely and foam and juice catchers were used.

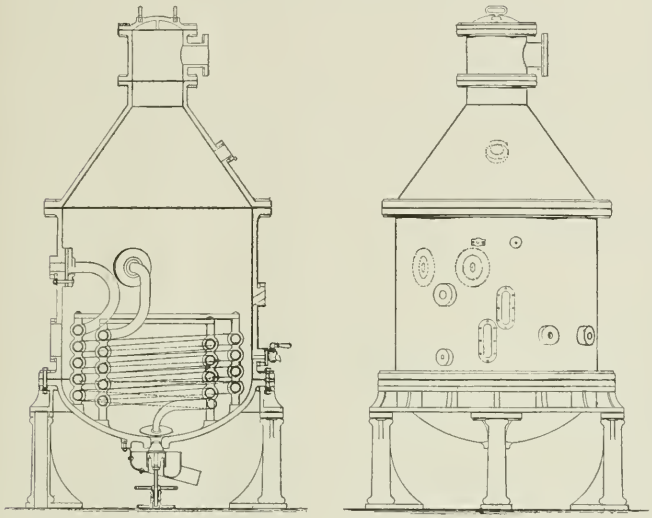


FIG. 41.—CAST-LEAD VACUUM PAN, WITH HARD-LEAD HEATING COILS.

Lead-lined sheet iron for vacuum pans and lead-lined copper coils for heating the acid liquids can be used with great success. Lead-lined sheet iron can be handled more easily than lead casting and be made much lighter in weight, which makes it preferable for the construction of vacuum pans. Lead-lined copper coils have a larger heating efficiency than lead; higher steam pressure can be used and little repair is required.

Fig. 43 shows the top of a lead-lined vacuum pan with foam catcher in the dome and a juice catcher in the vapor line, connected to the condenser.

Fig. 44 shows a complete disposition plant of a homogeneous lead-lined double vacuum pan, provided with single lead-lined copper heating coils and catch-all in the dome, through which the vapors pass to the condenser.

For the evaporation of acids the following constructions have to be mentioned also, which are built by the "Kuehnle-Kopp-Kausch" or homogeneous lead-lined system. Very often two separate evaporators are connected with one juice catcher and one condenser. Such an installation is represented in Fig. 45. These two evaporators are equipped with two homogeneous lead-lined copper coils, about 100 ft. long each and 2¾-3 in. diameter, having an evaporating capacity of about 16 to 20 lbs. water per minute. Both evaporators are working together, but as shown any of them can be cut out occasionally.

A very interesting homogeneous lead-lined evaporator is Fig. 46, which in fact is a double effect. Between the first and second effect we see a catch-all, where all the drops carried over from the first pan by the vapors, are caught and the vapor passes to the second effect, where a higher vacuum is obtained. This second effect has a catch-all in the dome, so that as few drops as possible should be carried over into the overflow, which is inserted between the pan and the plainly seen counter-current condenser.

These heating coils have in the first effect 3⅜-in. diameter by 300 ft. length and a heating surface of about 323 sq. ft. A homogeneous lead-lined copper coil for about 85 lbs. working pressure is followed by a lead-coil in the second effect with 3⅝-in. diameter and 323 ft. length. About 17 gal. of sulphate-alum solution are evaporated per minute. The liquor is in the first effect boiled and the vapors from there are used for the second effect, where the cooking process is finished. In this case both effects have to work together.

Another arrangement for an acid vacuum-evaporating plant, something similar to Fig. 45 is shown in the two homogeneous lead-line vacuum pans of Fig. 47. However, each pan has its own overflow and condenser, but both condensers are connected with one water separator and vacuum pump. This arrangement makes it possible that each vacuum pan can work independently from the other. The necessary valves are inserted between the condensers and tail pipe, cooling-water inlet and in the connections of the condensers and water separator.

Each apparatus is provided with a homogeneous lead-lined copper coil and each has a heating capacity of about 162 sq. ft. and concentrates per minute 10 gal. liquor to 2¾ gal. of acid.

It is always preferred to use above-mentioned equipments, because if two vacuum pans are connected to one condenser bronze or lead valves have to be used to cut out one pan. However, bronze is destroyed too quickly by the influence of acids and lead valves also do not last long enough; they get untight in a very short time.

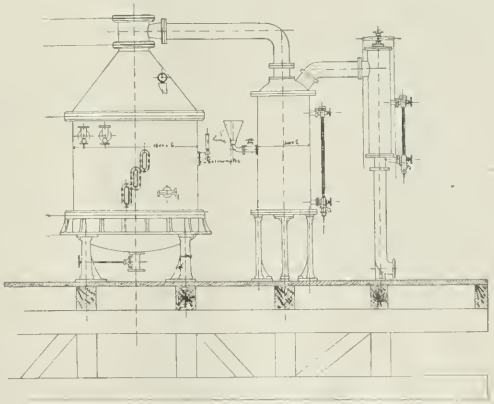


FIG. 42.—VACUUM PAN WITH OVERFLOW AND CONDENSER.

Fig. 48 is a homogeneous lead-lined vacuum evaporator with vertical heating pipes. The steam chamber encloses the outer surfaces of the pipes and the liquor circulates through them

exactly the same way as already mentioned in another part of this paper. To give the liquor a more lively movement and thereby enforce the evaporation, a propeller mixer is equipped in the center of the evaporator, which drives the cooler layers of the bottom always upward, making 150 rev. per minute. The setting of these mixers caused the change of the situation of the vapor outlet. This evaporator has about 280 sq. ft. heating surface and evaporates about 13 gal. of acid liquor per minute. It can be heated with steam from about 65 lbs. pressure.

A very simple construction of a homogeneous lead-lined evaporator with jacket is shown in Fig. 49, having 4 ft. 3 in. diameter, about 52 sq. ft. heating surface and when heated by live steam of about 80 lbs. pressure it evaporates around 4 gal. of liquor per minute. On the top in the center, the vapors are escaping to a condenser.

[*This serial is to be concluded in our next issue when vacuum dryers will be dealt with.*]

The Control of Thermal Operations and the Bureau of Standards. *

By George K. Burgess, D. Sc.

The name that is generally recognized as standing foremost in the determination of the thermal properties of materials is that of Regnault, but it is perhaps not as generally known that the experimental investigations in heat carried out by him over an interval of some thirty years, beginning in 1840, were initiated and supported by the French Government. The problem that Regnault first attacked was that of the properties of steam, a particularly opportune subject at that epoch, and the title page of his first memoir on the subject reads as follows:

"Account of the Experiments undertaken by order of the Minister of Public Works, and on the proposition of the Central Committee on Steam Engines to Determine the Principal Laws and the Numerical Data which enter into the Calculation of Steam Engines."

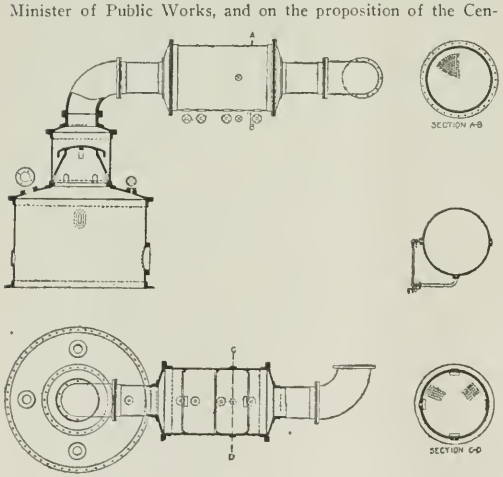


FIG. 43.—TOP OF VACUUM PAN WITH FOAM CATCHER AND JUICE CATCHER.

This was published in 1847, or seven years after the granting of the government subsidy. This delay was in part due to the necessity Regnault labored under of carrying on many researches, fundamental and subsidiary to his main problem, including, for example, the construction of both a temperature and pressure scale, as well as determinations of absolute densities and of specific and latent heats.

The example thus set by France in giving sustained governmental aid for the determination of thermal constants of fundamental importance has been appreciated and followed after many years, it is true, by other countries, several of which have established government laboratories for the carrying out of investigations, not only on the fundamental thermal properties of materials but also, in some instances, with activities covering

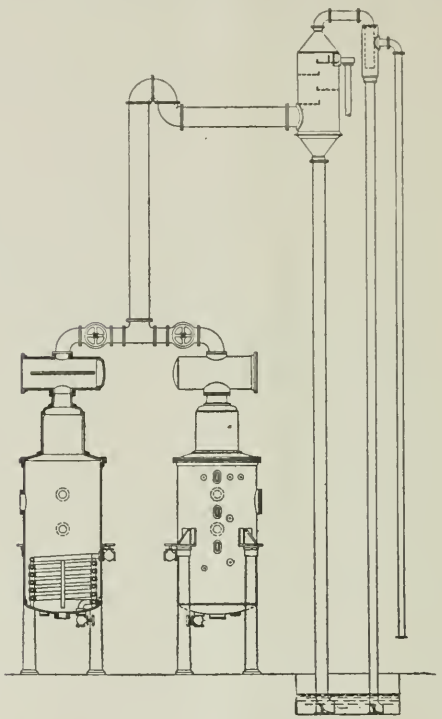


FIG. 44.—HOMOGENEOUS LEAD-LINED DOUBLE VACUUM PAN WITH SINGLE LEAD-LINED COPPER COIL.

the whole range of the physical sciences, including the testing of instruments and materials submitted by the public. Of this type are the German Imperial Physico-Technical Institution (P. T. Reichsanstalt), established in 1887, the National Physical Laboratory of Great Britain, organized in 1899, and the Bureau of Standards founded only ten years ago.

It would take us too far afield to endeavor to cover the whole subject of standardization and its relation to industry, and it will serve our purpose to limit ourselves to the consideration of a domain, that of heat, in which the development of standards has been less rapid and less exact than in many other lines. We shall sketch in some detail the part taken by the Bureau of Standards in the control of thermal operations, with the object of calling attention to an aspect of the question of economic efficiency that is of fundamental importance, namely, the rôle of common units and standards, their establishment, maintenance and availability.

It is a most painstaking matter to set up and maintain a temperature scale, in fact the textbooks state that temperatures cannot be measured and that we can only recognize or locate a given temperature in terms of some other phenomenon that we can recognize or measure. For such reasons, in great contrast to the certainty and permanence of our standards of

*A paper read at the Congress of Technology, April 11, 1911, in celebration of the fiftieth anniversary of the granting of the charter to the Massachusetts Institute of Technology.

length, mass, time, and more recently the electrical quantities, we have had and still have great discrepancies in our temperature scales and all the varied quantitative relations and thermal constants depending on them.

It is eminently fitting, therefore, and particularly necessary that there be a central authority for the establishment, maintenance and distribution of a single temperature scale and the

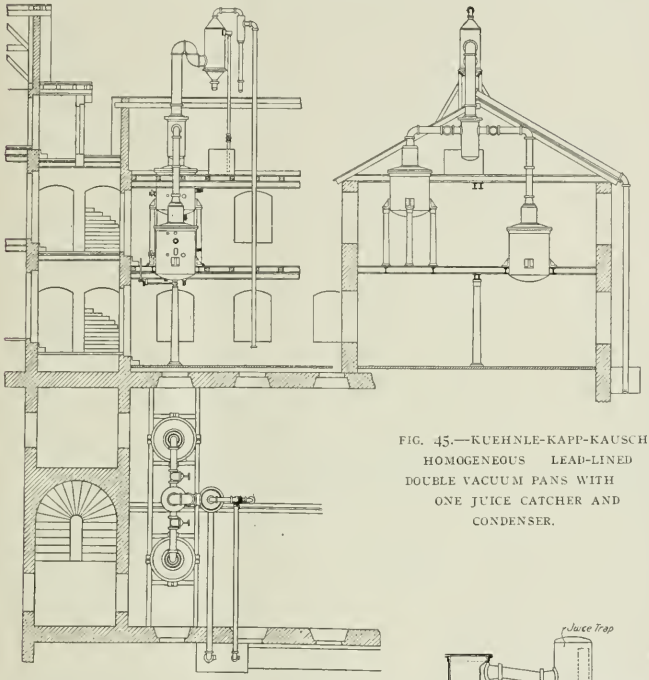


FIG. 45.—KUEHNLE-KAPP-KAUSCH HOMOGENEOUS LEAD-LINED DOUBLE VACUUM PANS WITH ONE JUICE CATCHER AND CONDENSER.

derived heat units. The International Bureau of Weights and Measures, established in 1871 at Sèvres near Paris revising the work of Regnault and others, performs this function in the first instance for the fundamental thermometric interval 0-100° C, while the higher and lower temperature ranges and derived quantities have been left mainly to the various national laboratories since their establishment, as they were previously left entirely to individual initiative. Among the most brilliant examples of the latter are those of Barus in the early eighties, when he was physicist of the Geological Survey, Rowland at the Johns Hopkins, and our own Holman.

Experience has shown, however, that it is quite impracticable for any individual working alone, and engaged in other pursuits as he must be, to maintain and disseminate a temperature scale. Indeed, progress in exact measurements in thermal constants and in all physical constants depending on temperature has often been greatly hampered in the past by the unavailability of any common temperature scale in terms of which results might be expressed. Each investigator was speaking a different temperature language which he could translate into no other. It required, for example, several elaborate, subsequent researches to express the results of Rowland's magnificent work on the mechanical equivalent of heat in terms of the hydrogen scale of the International Bureau. To-day it is neither necessary nor desirable for an independent investigator to try and set up his own temperature scale. Continuity and permanence of thermal standards can only be maintained to advantage by laboratories especially equipped for the purpose, the organization and working of which are not dependent on any individual and the equipment and staff of which are adequate to meet the demands of the public.

Since its creation in 1901, the Bureau of Standards has been actively engaged, in so far as its resources would permit while developing many other lines of work, in serving the United States in the capacity of an advisor on best values and available methods in the many lines of thermal measurements. This activity has necessitated the undertaking of a considerable number of experimental investigations, the carrying on of a very extended correspondence and the execution of many tests of instruments and materials. The bureau does not impose its authority in thermal standards on anyone, and any prestige in this field it may possess must rest on the reasonableness of its suggestions.

The temperature scale of the International Bureau has been reproduced here

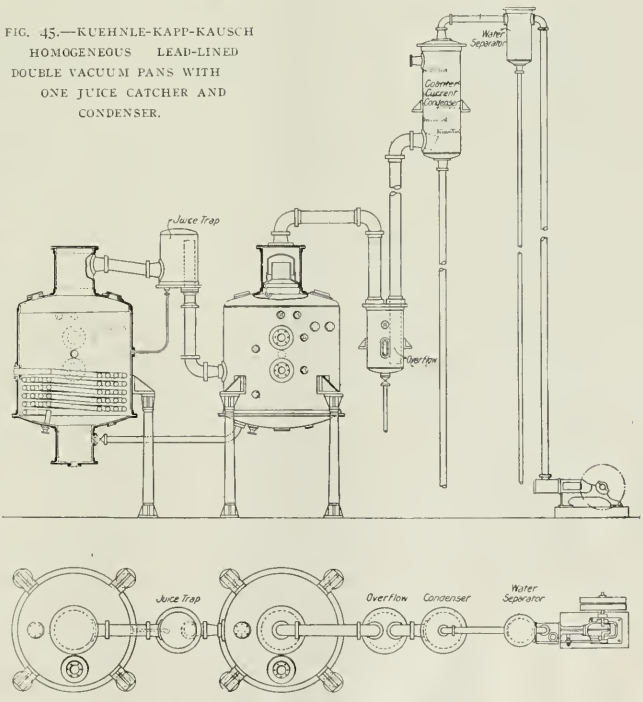


FIG. 46.—HOMOGENEOUS LEAD-LINED DOUBLE VACUUM PANS FOR SULPHATE ALUM SOLUTION.

to 0.°002 C in the interval 0 to 100° C, or to as close as temperatures are known within this interval; and for the remainder of the temperature range, for which there is no international authority, a temperature scale has been constructed which, except perhaps for the very highest temperatures, is in substantial agreement with those maintained

by the German and British laboratories. This temperature scale of the Bureau of Standards is reproducible to about 0.°03 at 500°, 1° at 1000° and 10° at 2000° C and has required for its establishment a great deal of experimental work, and advantage has of course been taken also of similar work elsewhere.

It may be of some interest to consider the relation of this bureau to some of the various bodies with which it comes in contact and mention in particular some of the thermal problems to the solution of which it has contributed.

The bureau is by law constituted the custodian of the standards and is authorized by the organic act creating it to exercise such functions as "the comparison of the standards used in scientific investigations, engineering, manufacturing, commerce and educational institutions with the standards adopted or recognized by the government; the construction, when necessary, of standards, their multiples and subdivisions; the testing and calibration of standard measuring apparatus; the solution of problems which arise in connection with standards; the determination of physical constants and the properties of materials when such data are of great importance to scientific or manufacturing interests and are not to be obtained with sufficient accuracy elsewhere."

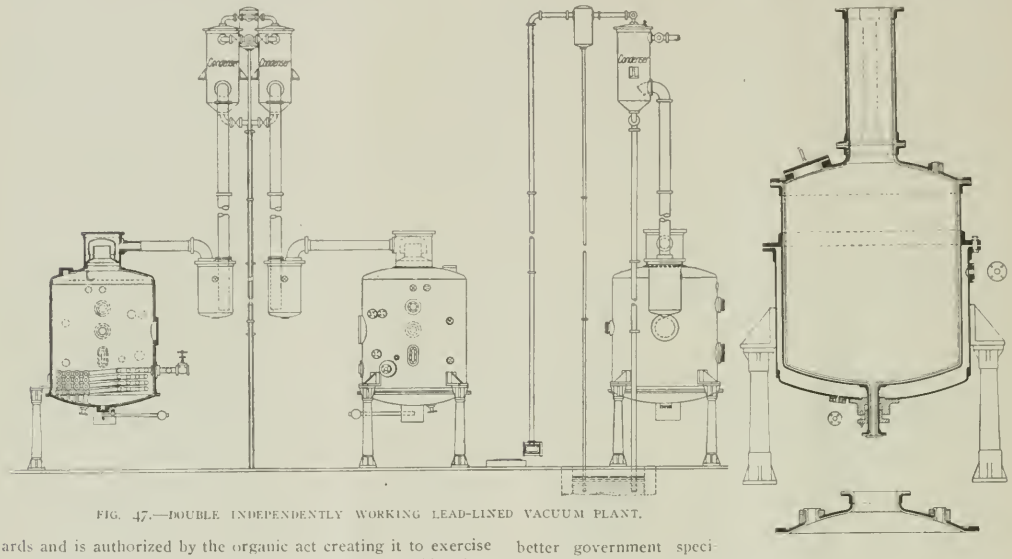


FIG. 47.—DOUBLE INDEPENDENTLY WORKING LEAD-LINED VACUUM PLANT.

Questions arising in the departments of the government requiring advice or decisions on the physical properties of materials are usually referred for solution to the bureau, and this demand for its services and advice is growing at a greatly accelerated rate for tests of various kinds of temperature measuring instruments, for the formulation of specifications, for the purchase of thermal apparatus and of materials on which thermal tests may be made to control their quality, and for the carrying out of specific experimental investigations on materials, processes or methods involved in the solution of some problem in which one or another department of the government is interested.

To mention a few examples among many of the control thus exercised, the purchase of clinical thermometers for two of the departments as well as of the various kinds of thermometers of several of the scientific bureaus is based on specifications drawn up by the bureau and its testing of the instruments before their acceptance, and the knowledge on the part of manufacturers that a bid may be rejected due to the findings of a disinterested, competent authority is not detrimental to the class of instrument submitted in competition for purchase in this way. Experience has proven this practice effective.

The purchase of refractory brick by the Panama Canal Commission is also based on such tests, as well as that of many other materials.

The decision as to whether contested materials are inflammable and, therefore, to be barred from carriage on passenger steamers, is left to the bureau, and such decision sometimes entails an unexpected amount of experimentation.

Investigations of lubricating and illuminating oils and of the types of apparatus used in the testing of them are being carried out with several objects in view, such as the drawing of better government specifications, the furnishing of data for better and more uniform testing methods both in this country and by international agreement. The tests of viscosity, for example, are now on a purely empirical basis, different countries use different instruments and in the United States there are several incommensurable instruments in common use. It is hoped to reduce all such measurements to a common basis.

FIG. 49.—HOMOGENEOUS LEAD-LINED EVAPORATOR WITH JACKET.

The bureau has been asked to take part in the work of several international and national societies or committees, usually with the object of establishing by experimentation the necessary conditions or specifications for the carrying out of some method of testing or standardizing materials, methods or instruments. We have not been able to meet all the demands of this kind in the field of thermal operations, but have limited ourselves to some of the problems that are in the most unsatisfactory condition. Besides the oil question, which is very troublesome, the subject of combustion and gas calorimetry has been attacked from the foundation, and although less than two years have been spent on this, we not infrequently receive praiseworthy letters of a slightly ironic cast asking for our final results on this or that phase of the problem. Accepting Regnault's standard, we still have five years to make good on this problem,

however. An outsider does not always realize that no one is more impatient to finish his problem than the worker himself, but where fundamental standards are concerned the worker is bound to be more conservative than his severest critics.

In this calorimetric work it has been necessary, for example, to design and install a complete plant for the production of pure oxygen, new calorimeters, resistance thermometers and accessory apparatus capable of giving results of the highest accuracy attainable at the present time, and the thermal constants must be determined with different sets of apparatus and checked by several methods.

This calorimetric investigation is very far-reaching and involves, for example, the heat values to be accepted as standard in this country for the constituent components of gas burned in every city and town and, therefore, the ultimate basis on which the price of gas as a fuel should be fixed. The heats of combustion which are being determined of those substances suitable for calibrating bomb calorimeters, in a similar way, will define the ultimate units on the basis of which coal and oil fuels should be bought and sold. Concrete calorimetric standards are realized in the form of certified samples of pure materials of known heat value, which are distributed to interested parties. The various types of calorimeter are also being compared, and incidentally to the whole investigation, improved methods of measurement and new instruments have been developed.

To the refrigerating industries we have furnished after elaborate experimentation the correct values of the specific heat of brine at low temperatures, a constant of very great importance to them. There is great need for further systemization of units in this field as well as the better determination of other constants fundamental to refrigeration. The question of these units will be gone over very carefully at the next International Congress of Refrigeration, which meets in this country next autumn.

The relation of the bureau to the manufacturers of precision instruments is very close, manifold, and in general most cordial. When it is remembered that previous to 1901 there was no generally recognized standardizing authority in this country for thermometers and that each manufacturer had his own scale tied up in one or more long cherished instruments, the breakage of which was a real calamity for him and his clients, the necessity for the establishment of a standards bureau is self-evident.

Besides the testing of thermometric instruments, and thereby furnishing all manufacturers with a single scale, uniform everywhere, the bureau, in certain cases, loans its own standards. It has also taken a very active part in the improvement of specifications and the methods and materials of manufacture as well as in testing methods. The resulting improvements in American thermometer manufacture have been such that in 1906 Gehr. Wiebe, the chief of the thermometer department of the Reichsanstalt, published the statement that the German thermometer manufacturers were complaining more and more of the loss of the American market.

The manufacturers of instruments for the measurement of high temperatures have also been aided greatly, and the condition in this field may be described as having been chaotic before the benefits of uniformity of calibration could be realized. Opportunity is also given for trying out and developing new types of instruments, and a great many suggestions have been given manufacturers which have been incorporated in their instruments. The standards of practically every pyrometer and thermometer manufacturer in the country have been tested at the bureau.

The past ten years has witnessed great strides in the development of the manufacture and use of pyrometers. Formerly the question usually asked by an engineer responsible for the control of some process involving change of product or output with varying temperature was "Do I need a pyrometer and is there a reliable one?" To-day he knows he needs one and asks which is the best for his purpose. The bureau carries on a very heavy correspondence with the users of such instruments, endeavoring to furnish them with the information best adapted to each case, although no pretense is made of replying to the often asked question, which is the best pyrometer?

In a similar way, advice has been furnished to universities

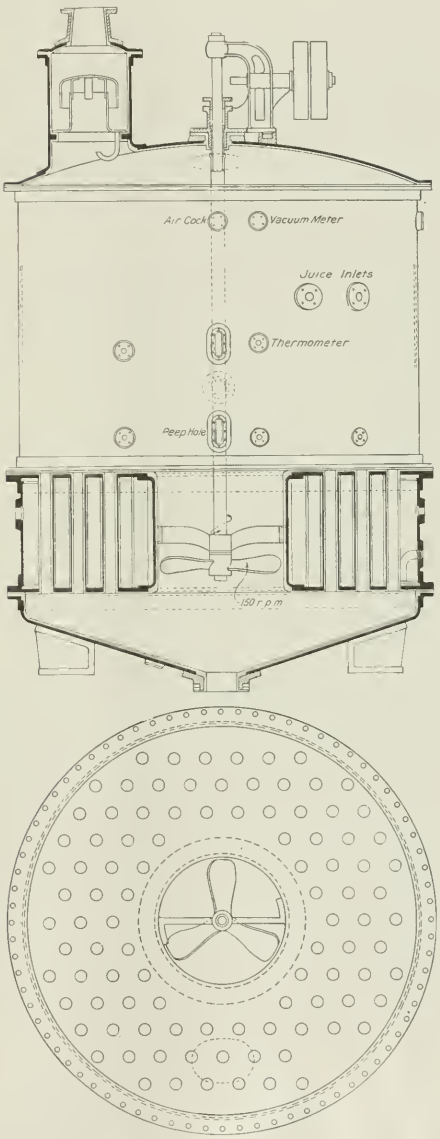


FIG. 48.—HOMOGENEOUS LEAD-LINED VACUUM EVAPORATOR WITH VERTICAL HEATING PIPES.

and technical schools regarding the formation of courses of instruction in heat measurements, the installation and design of new apparatus, and the testing with the highest possible accuracy of instruments to be used by them in numerous experimental investigations.

To still further aid in the dissemination of a uniform tem-

perature scale, preparations are being made for the distribution of samples of pure metals and salts of certified melting points, thus permitting any one to check his own thermometric or pyrometric apparatus in place. As a preliminary to this a careful survey of the reproducibility of the thermal behavior of such materials from several sources of supply has been made.

Among the other investigations under way or in immediate contemplation that have a bearing on the control of thermal operations may be mentioned studies of thermal properties of steels and refractories, including in both instances the purest obtainable materials and the commercial products; the properties of steam with respect to turbine design; the melting points of the elements and of some pure salts and alloys; gas thermometry; the temperatures of incandescent lamps; the corrections to be applied to optical and radiation pyrometers when sighting on steels, bronzes, clays, and so forth; and the comparison of the laws of radiation.

Some of these experimental problems have been undertaken at the suggestion of one or more interested individuals or corporations who often are in a position to furnish materials that may sometimes be difficult to obtain otherwise. Indeed, we are at times in the position of being offered a great many more opportunities of this kind than we can embrace.

Still another function the bureau has occasionally exercised, and one that will probably develop, is to act as referee or court of appeals in case of disputes involving standards, physical properties of materials and the like, and for the interpretation of contracts involving testing methods.

The testing activity of the heat division of the bureau for the past four years ending June 30, 1910, was as follows:

	1906-7	1907-8	1908-9	1909-10
Clinical thermometers.....	8,444	8,395	10,955	13,082
Various thermometers.....	577	798	1,791	1,508
Pyrometers	46	60	31	39
Calorimetric tests.....	....	....	41	58
Oils: Flash and viscosity.....	42	17	159	17
Miscellaneous	9	99	6	45

Several of these miscellaneous tests were of the nature of quite elaborate investigations and most of the pyrometers were manufacturers' standards. Nominal fees are charged for all tests except for the United States or State governments. About one-third of this testing is for the United States government. The bureau makes known the results of its investigations by means of its Bulletin, and its testing methods and specifications are described in Circulars of Information.

With the ever-increasing economic competition and the improvement of scientific methods and instruments as applied to the arts and sciences, the demand for higher accuracy of calibration is also constantly growing. Where, for instance, 0.°01 C was considered an ample precision a short while ago in combustion calorimetry, and 0.°001 C sufficed yesterday, the demand is now for a certainty of 0.°0001 C in temperature differences at 25° C. The meeting of such rigid requirements necessitates oftentimes the application of entirely new methods and instruments, and we are witnessing in consequence the passing of some familiar types, such as the mercury-in-glass thermometer, from the list of instruments of precision.

A most important step is yet to be taken in the unifying of the temperature scales and the values assigned to the various thermal constants in use in the several countries. To-day a pyrometer manufactured in France, calibrated in England and used in the United States comes in competition with a German-made and certified instrument and with one of American make and certification. They will each give a different temperature for a furnace of say 1500° C. We have the example of the unification of the standards of weights and measures as well as those of electricity and photometry, and it is to be hoped that ere long the standards of heat may also be rendered universal.

Washington, March 21, 1911.

The Methods of the United States Steel Corporation for the Technical Sampling and Analysis of Gases.

Copyright, 1911, by J. M. Camp, Chairman Chemists' Committee.

(*Concluded from page 306.*)

2. Blast Furnace Gas.—Before the analysis is commenced, the solutions in the pipettes are brought flush with the capillary tube, while the latter and burette are filled with water. The petcock on either end of the sample tube containing the gas sample (No. 2, analytical apparatus), is filled with water and attached to the rubber tubing, also full of water, leading from the shelf bottle (No. 1), while the other end of the sample tube is attached to the rubber tubing at the end of the main capillary and the sample tube is placed in its rack on the side of the apparatus. The leveling bottle is then placed on top of the apparatus, the pinchcock on the water supply, the lower, then the upper cock on the sample tube, and the three way cock on the capillary tube are opened; the water from the shelf bottle forces the gas into the burette against the pressure from the leveling bottle, which is lowered if necessary, the gas being always maintained under pressure.

About 25 cc of gas are thus forced into the burette, and then expelled through the stem of the three-way cock, all connections remaining as they are. This sample is rejected because it contains the air that was in the rubber connection between the capillary and sample tube. A second sample is forced in until it reaches the base of the burette and about one minute is allowed for the burette to drain and the temperatures to equalize; during this period the sample tube may be disconnected from the apparatus. With the leveling bottle on top of the apparatus, the tube leading from it is pinched between the thumb and finger and the three-way cock opened, the pressure of the gas being thus released. The remaining portion is slowly forced out by releasing the pressure until the meniscus just reaches the zero mark in the burette, then the three-way cock is closed. If the water in the leveling bottle is brought to the same level as that in the burette, the meniscus of the latter should be exactly at zero.

Determination of Carbon Dioxide. The leveling bottle is slowly raised after the cock in the first reagent pipette containing the solution of potassium hydrate has been turned so that the gas will bubble through the solution. If the leveling bottle should not be raised slowly the bubbles which first appear may be lost by being carried into the other compartment of the pipette. When all but about 5 cc of the gas has passed into the pipette the cock is turned to a position for drawing the gas back into the burette, keeping the leveling bottle raised, and the instant the reagent has been expelled from the side capillary of the pipette by the remaining gas the leveling bottle is lowered and the gas drawn back. Thus the gas is forced in and drawn out of the pipette at least three times, the reagent finally being drawn to its initial point in the stem of the capillary. The burette is allowed to drain for about one minute and the reading taken. The difference between this and the initial reading represents the percentage of carbon dioxide in the gas. If the percentage of carbon dioxide is high, to make certain that it has all been removed, the gas should again be passed into the solution of potassium hydrate and a reading taken as before.

Determination of Oxygen. The passing of the residue from the absorption of carbon dioxide into the fuming sulphuric acid for the absorption of the illuminants or olefiant gases is usually omitted with blast furnace gas, as these are either entirely absent or present only in such small quantities that they can scarcely be detected in the usual way. Therefore the residue is at once passed into the second pipette containing stick phosphorus and left there for about one minute to remove the small amount, if any, of oxygen found in blast furnace gas. The difference between the burette reading obtained and that after the absorption of carbon dioxide represent the percentage of oxygen present.

Determination of Carbon Monoxide, Hydrogen and Methane. One-half the above residue is used for the determination of these gases. The last reading when substracted from 100 shows the volume of the residual gases remaining in the burette. One-half this residue substracted from 100 shows the point on the burette the meniscus must reach to reserve one-half the residue. If it is desired to duplicate the explosion, one-half of the residue is reserved in the phosphorus pipette. If not, the leveling bottle is placed on top of the apparatus, the connecting rubber tubing pinched, and the residue to be rejected is slowly forced out through the stem of the three-way cock until the meniscus just reaches the calculated point. The three-way cock is closed and by the application of the leveling bottle the calculated reading should be obtained.

Approximately 45 cc of the oxygen and nitrogen mixture are now passed into the burette, through the permanent rubber tubing connection, and an exact reading is made. If the carbonic acid in the absence of oxygen is low in the sample it indicates a gas rich in combustibles, in such case a proportional increase is made in the amount added of the oxygen and nitrogen mixture. The leveling bottle is raised, the cock leading to the explosion pipette opened, and the gases are passed over and back two or more times to insure a thorough mixture; the gases are exploded, drawn back into the burette and after the lapse of one minute Contraction I is noted.

The residual gas is passed through the potassium hydrate pipette at least three times to remove the carbon dioxide. After the burette has drained a reading is made, which, less the preceding reading, gives Contraction II. The residual gas consists of nitrogen and oxygen, and for the purpose of calculation, the oxygen must be known. About 35 cc of hydrogen are added, an exact reading is taken, and the gases are passed back and forth into the explosion pipette until a thorough mixture is effected. The gases are exploded, drawn back into the burette, allowed to drain and cool and a reading again taken. This reading, less the former, represents the contraction due to the combustion of hydrogen and oxygen. This is known as Contraction III, and one-third of it represents the oxygen present prior to the last explosion.

In the analysis of all gases there should be an excess of hydrogen after Contraction III, hence two-thirds of Contraction III should be less than the hydrogen added; if it is not less oxygen still remains, this is removed by passing the residue into the phosphorus pipette until the oxygen is removed. The contraction is noted, multiplied by three and added to Contraction III; one-third of this total is the oxygen present prior to the last explosion.

The remaining gas consists of nitrogen and hydrogen, which do not enter into the subsequent calculations, as all the necessary data has been obtained for calculating the amounts of carbon monoxide, hydrogen, methane and nitrogen that existed in the original mixture. The reactions which take place in the combustions are as follows: $2CO+O_2=2CO_2$, or two volumes of carbon monoxide unite with one volume of oxygen, forming two volumes of carbon dioxide, there being a contraction of one volume. Therefore, the contraction and the oxygen used are each one-half the carbon monoxide, and the carbon dioxide formed has the same volume as the monoxide. $2H_2+O_2=2H_2O$, or two volumes of hydrogen unite with one volume of oxygen forming water vapor which condenses, the contraction being three volumes. The contraction is then three-halves the hydrogen, and the oxygen consumed is one-half the hydrogen. $CH_4+2O_2=CO_2+2H_2O$, or one volume of methane unites with two volumes of oxygen, forming one volume of carbon dioxide, the water condensing and the contraction being two volumes. The contraction and the oxygen used are each twice the methane and the carbon dioxide has the same volume as the methane.

From these facts the following equations are formed:

Contraction I=½ carbon monoxide+3/2 hydrogen+2 methane.

Carbon dioxide formed=carbon monoxide+methane.

Oxygen consumed=½ carbon monoxide+½ hydrogen+2 methane.

From these three equations a variety of formulæ may be derived for calculating the components of the original mixture. The simplest are:

Oxygen consumed = oxygen added — ⅓ Contraction III.

Hydrogen=Contraction I—oxygen consumed.

$$\text{Carbon monoxide}=\frac{\text{Con. I}+(4\times\text{Con. II})+\text{Con. III}}{3}-\text{oxygen added.}$$

Methane = Contraction II — carbon monoxide present.

As the analysis by explosion was performed on half portion the results must be doubled for expression of percentage. Nitrogen in all cases is obtained by difference, i. e., it is equal to 100 minus the sum of the percentage of the other constituents of the gas.

Example—Data.

Burette Readings.	
0.0	
Potassium hydrate.....	11.3% carbon dioxide.
11.3	
Stick Phosphorus.....	0.0 Oxygen.
11.3	
	88.7 Residue.
	44.3 ½ Residue.
	55.7 Water level.
Add oxygen and nitrogen mixture	46.6 mixture added, 44.4% oxygen.
	20.7 oxygen added.
9.1	
Explosion	9.9 Contraction I.
19.0	
Potassium hydrate.....	12.9 Contraction II.
31.9	
Add hydrogen.....	31.4 hydrogen added.
0.5	
Explosion	39.0 Contraction III.
39.5	13.0⅓ Contraction III, or oxygen in excess.

Calculations.

20.7 oxygen added.
13.0 oxygen unconsumed, or ⅓ Contraction III.

7.7 oxygen consumed in explosion I.
9.9 Contraction I.

2.2 hydrogen present.

9.9 Contraction I.
51.6 4 × Contraction II.
39.0 Contraction III.

100.5 total.

33.5 ⅓ total.
20.7 oxygen added.

12.8 carbon monoxide present.
12.9 Contraction II.

0.1 methane present.

Explosion ratio: I, 2.99: II, 1.55

Analysis.	Per Cent.
Carbon dioxide.....	11.3
Oxygen	0.0
Carbon monoxide	25.6
Methane	0.2
Hydrogen	4.4
Nitrogen	58.5
	100.0

Occasionally it is found that the gas when mixed with hydrogen for the second explosion will not explode, due to the fact that, the gas sample being abnormally high in combustibles,

TABLE OF FACTORS FOR CORRECTING THE VOLUMES OF GAS TO STANDARD CONDITIONS OF 62° F AND 30" MERCURY.

Temperature in Degrees Fahrenheit	Barometer in Inches of Mercury																			
	28.6	28.6	28.7	28.8	28.9	29.0	29.1	29.2	29.3	29.4	29.5	29.6	29.7	29.8	29.9	30.0	30.1	30.2	30.3	30.4	30.6
40	992	995	999	1002	1006	1009	1013	1016	1020	1023	1027	1030	1034	1037	1041	1044	1047	1051	1054	1058	1061
41	990	993	997	1000	1004	1007	1011	1014	1018	1021	1025	1028	1032	1035	1039	1042	1045	1049	1052	1056	1060
42	988	991	995	998	1002	1005	1009	1012	1016	1019	1022	1026	1030	1033	1036	1040	1043	1047	1050	1054	1057
43	986	989	993	996	1000	1003	1007	1010	1013	1017	1020	1024	1027	1031	1034	1038	1041	1045	1048	1052	1055
44	983	988	991	994	998	1001	1005	1008	1012	1015	1019	1022	1026	1029	1032	1036	1039	1043	1046	1050	1053
45	982	986	989	992	996	999	1003	1006	1010	1013	1017	1020	1023	1027	1030	1034	1037	1041	1044	1048	1051
46	980	984	987	990	994	997	1001	1004	1008	1011	1015	1018	1021	1025	1028	1032	1035	1039	1042	1045	1049
47	978	982	985	989	992	996	999	1002	1006	1009	1013	1016	1020	1023	1026	1030	1033	1037	1040	1044	1047
48	976	980	983	986	990	993	997	1000	1004	1007	1010	1014	1017	1021	1024	1028	1031	1034	1038	1041	1045
49	974	978	981	985	988	992	995	998	1002	1005	1009	1012	1015	1019	1022	1026	1029	1033	1036	1039	1043
50	972	976	979	983	986	989	993	996	1000	1003	1007	1010	1013	1017	1020	1024	1027	1030	1034	1037	1041
51	970	974	977	981	984	987	991	994	998	1001	1004	1008	1011	1015	1018	1022	1025	1028	1032	1035	1039
52	969	972	976	979	982	986	989	992	996	999	1003	1006	1010	1013	1016	1020	1023	1026	1030	1033	1037
53	967	970	973	977	980	984	987	990	994	997	1001	1004	1007	1011	1014	1018	1021	1024	1028	1031	1035
54	965	968	972	975	978	982	985	988	992	995	999	1002	1005	1009	1012	1016	1019	1022	1026	1029	1032
55	963	966	970	973	977	980	983	987	990	993	997	1000	1004	1007	1010	1014	1017	1020	1024	1027	1031
56	961	964	968	971	975	978	981	985	988	991	995	998	1001	1005	1008	1012	1015	1018	1022	1025	1028
57	959	963	966	969	973	976	980	983	986	990	993	996	1000	1003	1006	1010	1013	1016	1020	1023	1027
58	957	961	964	967	971	974	977	981	984	988	991	994	998	1001	1004	1008	1011	1014	1018	1021	1025
59	956	959	962	966	969	972	976	979	982	986	989	993	996	999	1003	1006	1009	1013	1016	1019	1023
60	954	957	960	964	967	970	974	977	980	984	987	990	994	997	1000	1004	1007	1010	1014	1017	1021
61	952	955	959	962	965	969	972	975	979	982	985	989	992	995	999	1002	1005	1009	1012	1015	1019
62	950	953	957	960	963	967	970	973	977	980	983	987	990	993	997		1003	1007	1010	1013	1017
63	948	952	955	958	962	965	968	971	975	978	981	985	988	991	995	998	1001	1005	1008	1011	1015
64	946	950	953	956	960	963	966	970	973	976	980	983	986	990	993	996	1000	1003	1006	1009	1013
65	944	948	951	954	958	961	964	968	971	974	978	981	984	988	991	994	998	1001	1004	1007	1011
66	943	946	949	953	956	959	963	966	969	973	976	979	982	986	989	992	996	999	1002	1006	1009
67	941	944	948	951	954	958	961	964	967	971	974	977	981	984	987	991	994	997	1001	1004	1007
68	939	942	946	949	952	956	959	962	965	969	972	975	979	982	985	989	992	995	998	1002	1005
69	937	941	944	947	951	954	957	960	964	967	970	974	977	980	983	987	990	993	997	1000	1003
70	936	939	942	946	949	952	955	959	962	965	968	972	975	978	982	985	988	991	995	998	1001
71	934	937	940	944	947	950	954	957	960	963	967	970	973	977	980	983	986	990	993	996	999
72	932	936	939	942	945	949	952	955	958	962	965	968	971	975	978	981	985	988	991	994	998
73	930	933	937	940	943	947	950	953	956	960	963	966	969	973	976	979	982	986	989	992	996
74	929	932	935	938	942	945	948	951	955	958	961	964	968	971	974	977	981	984	987	990	994
75	927	930	933	937	940	943	946	950	953	956	959	963	966	969	972	976	979	982	985	989	992
76	925	928	932	935	938	941	945	948	951	954	958	961	964	967	971	974	977	980	984	987	990
77	923	927	930	933	936	940	943	946	949	953	956	959	962	966	969	972	975	978	982	985	988
78	922	925	928	931	935	938	941	944	948	951	954	957	960	964	967	970	973	977	980	983	986
79	920	923	926	929	933	936	939	943	946	949	952	955	959	962	965	968	972	975	978	981	985
80	918	921	925	928	931	934	938	941	944	947	950	954	957	960	963	967	970	973	976	979	983
81	917	920	923	926	929	933	936	939	942	945	949	952	955	958	962	965	968	971	974	978	981
82	915	918	921	924	928	931	934	937	941	944	947	950	953	957	960	963	966	969	973	976	979
83	913	916	920	923	926	929	932	936	939	942	945	948	952	955	958	961	964	968	971	974	977
84	911	915	918	921	924	927	931	934	937	940	943	947	950	953	956	959	963	966	969	972	975
85	910	913	916	919	922	926	929	932	935	938	942	945	948	951	954	958	961	964	967	970	974
86	908	911	915	918	921	924	927	930	934	937	940	943	946	950	953	956	959	962	965	969	972
87	907	910	912	916	919	922	926	929	931	935	938	942	945	948	951	954	957	961	964	967	970
88	905	908	911	914	918	921	924	927	930	933	937	940	943	946	949	952	956	959	962	965	968
89	903	906	909	913	916	919	922	925	929	932	935	938	941	944	948	951	954	957	960	963	967
90	901	905	908	911	914	917	920	924	927	930	933	936	939	943	946	949	952	955	958	962	965
91	900	903	906	909	913	916	919	922	925	928	932	935	938	941	944	947	951	954	957	960	963
92	898	901	905	908	911	914	917	920	924	927	930	933	936	939	942	946	949	952	955	958	961
93	897	900	903	906	909	912	915	919	922	925	928	931	934	937	941	944	947	950	953	956	960
94	895	898	901	904	907	911	914	917	920	923	926	929	933	936	939	942	945	948	951	955	958
95	893	897	900	903	906	909	912	915	919	922	925	928	931	934	937	940	944	947	950	953	956
96	892	895	898	901	904	907	911	914	917	920	923	926	929	932	936	939	942	945	948	951	954
97	890	893	896	900	903	906	909	912	915	918	921	925	928	931	934	937	940	943	946	949	953
98	889	892	895	898	901	904	907	910	914	917	920	923	926	929	932	935	939	942	945	948	951
99	887	890	893	896	899	902	906	909	912	915	918	921	924	927	930	934	937	940	943	946	949
100	885	889	892	895	898	901	904	907	910	913	917	920	923	926	929	932	935	938	941	945	948

an insufficient amount of oxygen remains after the first explosion. In such case the gas is drawn back into the burette and the oxygen and nitrogen mixture is added until the volume approaches 100 c. c.; the gases are again mixed, exploded and the analysis completed as usual, the oxygen added being found from the sum of the two additions. If, in the first explosion of any analysis, the ratio of inert to combining gases is not sufficiently high, and doubt exists as to the formation of oxides of nitrogen, the explosion and resulting determinations may be duplicated, using a larger amount of oxygen and nitrogen mixture and the portion of the residue which has been reserved in the phosphorus pipette, or the residue from a new portion obtained in the following manner. One hundred cubic centimeters are taken from the original sample and the carbon dioxide, oxygen, and illuminants if present, are removed in the usual manner. The remaining residue will not be the same as that obtained on the first sample on account of the reduction in volume of the carbonic acid, due to its solubility in the water in the sample tube, but a part of the residue is taken equal in volume to the gas used in the first explosion, and the analysis is continued as usual.

3. **Producer Gas.**—For the analysis of producer gas the method as described under Blast Furnace Gas is used with one or two modifications as subsequently described.

Determination of Carbon Dioxide and Illuminants. A sample of exactly 100 c. c. is measured and the carbon dioxide removed in the usual manner. Next the gas is bubbled twice through the fuming sulphuric acid pipette to remove

the unsaturated hydrocarbons, composed chiefly of ethylene and commonly called illuminants. Care must be exercised not to allow any water to enter the pipette. After the acid has been drawn to its initial point in the stem of the pipette, the gas is passed into the solution of potassium hydrate two or three times to remove the white acid fumes. A reading is taken and the decrease in volume noted indicates the percentage of illuminants present.

Determination of Oxygen. Next the oxygen is removed, and since the presence of illuminants entirely or partially inhibits the desired reaction with stick phosphorus, it is essential that they be removed before this is attempted.

Determination of Carbon Monoxide, Hydrogen and Methane. Producer gas is much richer in combustible gases than blast furnace gas; therefore the extent of dilution of the residue taken for the explosion should be such as to reduce to the minimum the temperature of the latter, and thereby the error due to formation of oxides of nitrogen. It is also desired that the maximum amount of the original residue practicable enter the explosion, in order that multiplying errors due to fractionation of the sample may be reduced to the minimum. The method to be used for the determination of these constituents is dependent upon the percentage of combustible gases in the sample. This percentage usually varies inversely with that of the carbon dioxide present.

Method "A" is used when less than seven per cent of carbon dioxide is present in the sample. One-half of the residue remaining after the oxygen content has been determined is rejected or reserved in the phosphorus pipette. The remaining half is passed into the explosion pipette, the stem of the latter being completely filled with water. Approximately 70 c. c. of oxygen and nitrogen mixture are added, the mixture in the intervening capillary being swept into the burette by the water remaining in the stem of the explosion pipette. Then the oxygen and nitrogen mixture is accurately measured, mixed with the above residue and exploded; the gases are returned to the burette, allowed to cool and a reading is taken. Contraction I is found by subtracting the residue left after the explosion from the sum of the residue taken for the explosion and the auxiliary gas added. Contraction II, the amount of carbonic acid formed during the explosion, is next determined in the regular manner and the residue is passed into the explosion pipette. Approximately 70 c. c. of hydrogen are added, as just described with the oxygen and nitrogen mixture, and accurately measured, mixed with the above residue and exploded, the resulting gases being returned to the burette and measured in the usual way. Contraction III is found in the same manner as was Contraction I.

The expected amount of combustibles may not be present in the gas portion taken for the explosion, and the residue after the first or second explosion may be greater than can be measured in the burette. In case this should occur, the residue is measured in two portions, the sum of which plus 0.1 cc, the capillary space, equals the total residue.

Example—Data.

Burette Readings.		
0.0		
Potassium hydrate		4.3 Carbon dioxide.
4.3		
Fuming sulphuric		.6 Illuminants.
4.9		
Stick phosphorus		0.0 Oxygen.
4.9		
		95.1 Residue.
47.5	½	Residue.
		52.5 Water level.

Pass residue into explosion pipette		
100.0		
Add oxygen and nitrogen mixture		70.4 Mixture added, 45% oxygen.
29.6		31.7 Oxygen added.
Explosion	0.8	99.2 Residue.
		47.5 + 70.4 — 99.2 = 18.7 Contraction I.
Potassium hydrate		14.2 Contraction II.
15.0		85.0 Residue.
Pass residue into explosion pipette		
100.0		
Add hydrogen		70.3 Hydrogen added.
29.7		
Explosion	2.7	97.3 Residue.
		85.0 + 70.3 — 97.3 = 58.0 Contraction III.

Calculations.

31.7 Oxygen added.
19.3 Oxygen in excess, or ⅓ Contraction III.

12.4 Oxygen consumed in explosion.
18.7 Contraction I.

6.3 Hydrogen present.
18.7 Contraction I.
56.8 4 × Contraction II.
58.0 Contraction III.

133.5 Total.

44.5 ⅓ Total.
31.7 Oxygen added.

12.8 Carbon monoxide present.
14.2

1.4 Methane present.

Analysis.	Per cent.
Carbon dioxide	4.3
Illuminants	0.6
Oxygen	0.0
Carbon monoxide	25.6
Methane	2.8
Hydrogen	12.6
Nitrogen	54.1
	100.0

Explosion Ratios: I, 2.58; II, 1.68.

Method "B" is used when the carbon dioxide of the sample is above 7 per cent. The method of analysis calculation is identical with those described under Blast Furnace Gas, with the exception that oxygen and nitrogen mixture is added to the portion of the residue reserved for the explosion until the volume approaches 100 cc. If the carbon dioxide value is between 8.0 and 9.0 per cent the amount of oxygen and nitrogen mixture to be added should be diminished by about 10 cc or difficulty may be encountered in removing the excess oxygen in the second explosion.

Example—Data.

Burette Readings.		
0.0		
Potassium hydrate		7.5 Carbon dioxide.
7.5		
Fuming Sulphuric		.0.1 Illuminants.
7.6		
Stick phosphorus		0.0 Oxygen.
7.6		
		92.4 Residue.
46.2	½	Residue.
		53.8 Water level.

Add oxygen and nitrogen mixture	52.5 Mixture added, 44.5% oxygen.
1.3	23.4 Oxygen added.
Explosion	16.2 Contraction I.
17.5	
Potassium hydrate	11.2 Contraction II.
28.7	
Add hydrogen	28.7 Hydrogen added.
0.0	
Explosion	39.0 Contraction III.
39.0	
	13.0 ⅓ Contraction III.

Calculations.

23.4 Oxygen added.
13.0 Oxygen in excess or ⅓ Contraction III.

10.4 Oxygen consumed in explosion.
16.2 Contraction I.

5.8 Hydrogen present.

16.2 Contraction I.
44.8 4 × Contraction II.
39.0 Contraction III.

100.0 Total.

33.3 ⅓ Total.
23.4 Oxygen added.

0.9 Carbon Monoxide present.
11.2 Contraction II.

1.3 Methane present.

Analysis.	Per cent.
Carbon dioxide.....	7.5
Illuminants	.1
Oxygen	.0
Carbon Monoxide	19.8
Methane	2.6
Hydrogen	11.6
Nitrogen	58.4
	100.0

Explosion Ratios: I, 2.60; II, 1.56.

4. **Flue Gas.**—The repeated determination of large quantities of oxygen by stick phosphorus or pyrogallol solution introduces undesirable features with the use of either reagent. In the former case, unless care is exercised, the heat of reaction causes fusion, with consequent disarrangement of the phosphorus sticks; with the use of the latter reagent there is a liability of the incomplete removal of all the oxygen. In such a case it would be absorbed in the cuprous chloride solution with the carbon monoxide, if this reagent were used for the purpose, thus giving erroneous values to the carbon monoxide found. To overcome these difficulties the following method has been devised whereby the oxygen and carbon monoxide are determined by explosion, and it has been proven superior to the absorption method, which is the method usually followed in the analysis of flue gases.

Determination of Carbon Dioxide. This constituent is determined on exactly 100 cc, measured in the usual way, as under blast furnace gas.

Determination of Oxygen and Carbon Monoxide. One-half the volume of the residual gases is computed and rejected. The oxygen present in the sample varies inversely with the carbon dioxide content, consequently the hydrogen to be added for its combustion is dependent upon this factor. If the sample contains less than 10 per cent of carbon dioxide, approximately 20 cc of hydrogen and 30 cc of oxygen and nitrogen mixture are added to the reserved portion of the residue. If the sample contains more than 10 per cent of carbon dioxide, 15 cc of hydrogen and 35 cc of oxygen and nitrogen mixture are added The gases are thoroughly mixed, exploded, and after the proper interval Contraction I is noted. They are then passed through the solution of potassium hydrate to remove the carbon dioxide that has been formed from the burning of the carbon monoxide that may have been present, and Contraction II is noted.

To the residue hydrogen is added until the volume of the mixture approaches 100 cc. The gases are thoroughly mixed and exploded, drawn back into the burrette, and after the proper interval a reading is taken and Contraction III is noted.

The data necessary for the calculation of the percentage of oxygen and carbon monoxide in the sample has thus been obtained. From the reactions given under the determination of carbon monoxide, hydrogen and methane in blast furnace gas it is seen that when one volume of oxygen unites with two volumes carbon monoxide there is a contraction of one volume and the formation of two volumes of carbon dioxide, which on absorption cause a further contraction of two volumes; the total contraction is, therefore, three volumes for one volume of oxygen, or the oxygen is one-third the total contraction. The carbon monoxide was also shown to have the same volume as the dioxide formed. In the combustion of hydrogen and oxygen it was shown that the oxygen is one-third the contraction. Then the oxygen entering the two explosions, or the total oxygen, is one-third the sum of the three contractions and the oxygen in the sample is found by subtracting the oxygen added from this total oxygen. These facts are expressed in the following equations:

$$\text{Carbon monoxide} = \text{Contraction II.}$$

$$\text{Oxygen in sample} = \frac{\text{Con. I} + \text{Con. II} + \text{Con. III}}{3} - \text{Oxygen added.}$$

Burette Readings.

0.0	
Sodium hydrate	9.2% Carbon Dioxide.
9.2	
	90.8 Residue.
	45.4 ½ Residue.
	54.6 Water level.
Add hydrogen	19.8 Hydrogen added.
34.8	
Add oxygen and nitrogen mixture.....	29.9 Oxygen and nitrogen mixture added, 44.4% oxygen.
	13.3 oxygen added.
4.9	
Explosion	29.7 Contraction I.
34.6	
Sodium hydrate	.0 Contraction II.
34.6	
Add hydrogen	34.4 Hydrogen added.
0.2	
Explosion	25.2 Contraction III.
25.4	

Calculations.

0.0 Contraction II, or Carbon monoxide present.

29.7 Contraction I.
0.0 Contraction II.
25.2 Contraction III.

54.9 Total.

18.3 ⅓ total, or total oxygen.
13.3 Oxygen added.

5.0 Oxygen present.

Analysis.	Per cent.
Carbon dioxide.....	9.2
Oxygen	10.0
Carbon monoxide.....	0.0
Nitrogen	80.8
	100.0

Explosive ratios. I, 2.20; II, 2.96.

The absence of methane and hydrogen in a flue gas may be proven by the application of this formula:

$$H_2+CH_4=\frac{2(\text{Con. I})-\text{Con. II}}{3}-H_2 \text{ added for the first explosion.}$$

This is usually found to be zero.

Thermal Values.—If a calorimeter is used for determining the thermal value of a gas, the Junkers instrument is recommended for this purpose. The results so obtained must be expressed in b.t.u. under the standard conditions of dry gas at 62° Fahr. and 30 in. mercury.

When a calorimeter is not available it is customary to calculate the calorific value of a gas from the analytical results. Considerable discrepancies exist among the values commonly given as heats of combustion for the different gases; but it is believed that the following figures, being means of published values of several well-known investigators, will give results which are sufficiently reliable for all technical purposes.

In the following table, showing the b.t.u. developed in the combustion of 1 cu. ft. of the different gases, measured at 62° Fahr. and 30 in. mercury, two values are given—the gross and the net, the difference between them being the latent heat of the water formed, which is not included in the latter. The products of combustion in both cases are assumed to be at 62° Fahr. For the purpose of comparison all Corporation results should be expressed in terms of net b.t.u.

	Gross.	Net.
Ethylene (illuminants)	1591	1491
Carbon monoxide	324	324
Methane	1012	912
Hydrogen	328	278

In the calculation of the thermal value of a gas the percentage of each combustible present is multiplied by its value in the above table, and the sum of the products will represent the b.t.u. evolved in the combustion of 1 cu. ft. of the gas.

Specific Gravity and Percentage by Weight.—When suitable apparatus is not available for determining the specific gravity of gas, it may be calculated from the volumetric analysis by means of the following table compiled by the Bureau of Standards, showing the specific gravity of various gases compared with air:

Air	1.0000
Carbon dioxide	1.5290
Ethylene (illuminants)	.9684
Oxygen	1.1054
Carbon monoxide	.9670
Methane	.5545
Hydrogen	.06951
Nitrogen	.9674
Water vapor	.6222

The calculated specific gravity of a gas is found by adding the products obtained by multiplying the percentage by volume of each constituent gas by its specific gravity. The respective products thus obtained divided by the specific gravity of the gas yield the percentages by weight of the constituent gases:

The specific gravity of moist gas is found by adding the product of the specific gravity and the percentage of water vapor accompanying dry gas to the specific gravity of the dry gas, and dividing the sum by one plus the percentage of water vapor. This latter value is found by multiplying the grains per cubic foot dry gas by .3012, the volume occupied by 1 grain water vapor, or by dividing the vapor pressure by the difference of the gas pressure and the vapor pressure.

Weight of a Cubic Foot of Gas.—The following table, calculated from the most recent values given by the Bureau of Standards, shows the weight in pounds of 1 cu. ft. of the constituent gases at 62° and 30-in. pressure. The weight of 1 cu. ft. of gas is the sum of the values found by multiplying the weight per cubic foot of each of the constituent gases by its percentage by volume.

Air	.07626
Carbon dioxide	.11660
Ethylene	.07386
Oxygen	.08430
Carbon monoxide	.07375
Methane	.04228
Hydrogen	.005302
Nitrogen	.07377
Water vapor	.04745

The weight of 1 cu. ft. of moist gas is found by adding the product of the weight of a cu. ft. of water vapor and the percentage of water vapor accompanying dry gas to the weight of dry gas, and dividing the sum by one plus the percentage of water vapor.

Correction for Temperature and Pressure.—All metered gas shall be corrected to dry gas at 62° Fahr. and 30 in. of mercury pressure as the standard conditions for comparison.

When T = Temperature of gas at meter plus 459°.

T_s= Temperature sought plus 459°.

P = Pressure found by barometer + manometer reading at meter — vapor pressure at temperature of gas, and

P_s= Pressure sought.

Then a factor may be found by calculation, or from the table on page 358, equal to $\frac{T_s P}{P_s T}$ and applied as follows: The volume at standard conditions equals the volume observed times the factor. The b.t.u. at standard conditions equal the b.t.u. observed divided by the factor.

Vanadium in Cast Iron.*

By George L. Norris.

The use of vanadium in gray iron castings has been limited when compared to its use in steel castings, although the past year has shown a marked increase and its use in certain classes of castings has become well established. We have in the case of cast iron a material the cost of producing which from the crude pig iron has been slight. The price at which iron castings in general are sold precludes the use of any alloy for affecting the quality of the material, except in the case of certain classes of castings where unusual strength or some special quality in the iron is desired. In such cases even a very material increase in the cost of the castings is readily accepted.

In the case of vanadium we are dealing with a metal of fairly high price and its use in the manufacture of cast iron naturally will be confined to the classes of work that can stand a material increase in cost in return for the special qualities obtained. The addition of 1/10 per cent of vanadium to cast iron at the present price of $5 per pound means an increase of ½ cent a pound for the iron in the ladle and necessitates an increase in selling price of ¾ cent to 1 cent a pound at least to cover losses due to scrap, gates, spruces, etc.

While vanadium has a strong affinity for oxygen and nitrogen and is partly used up by combining with these elements, the primary purpose of adding vanadium to cast iron is not the removal of oxygen and nitrogen. The removal of oxygen can be effected sufficiently well by cheaper means and as the injurious effect of small amounts of nitrogen on a refined metal, such as steel, is still an open question, the harmfulness of this element on a metallurgically crude metal like cast iron is open to still greater doubt.

Vanadium increases the strength of the cast iron to which it is added from 10 per cent to 25 per cent, depending, of course, upon the initial strength of the iron. This increase of strength is due directly to the intensifying effect of the vana-

*Abstract of a paper presented before the American Foundrymen's Association at the Pittsburgh Convention.

dium on the metals present in the cast-iron alloy, such as silicon and manganese and also by combination with the chemically combined carbon.

It exercises a very strong effect on the grain of the iron, causing a more even distribution of the graphite with consequent greater freedom from both hard and porous spots. This effect is very apparent, vanadium cast-iron fractures showing a finer and more uniform grain. In the case of chilled iron, vanadium produces a deeper, stronger and tougher chill, and one less liable to spall or flake. This chill, however, is not as hard apparently as the more brittle chill of ordinary chilling iron, and can be filed and machined more readily than the latter.

Vanadium also gives an element of toughness to cast iron which is very apparent in machining the castings. This toughening effect, together with the uniform fineness of grain or texture due to even distribution of graphite in small flakes, makes vanadium cast iron very superior for such purposes as gas and steam-engine cylinders, glass-bottle molds, gears, castings to withstand heat and internal pressures of fluids and gases without leakage.

Three or four years ago the New York Central lines equipped one of its locomotives with a pair of vanadium cast-iron cylinders. The performance of this pair of cylinders over a test period of about two years was so satisfactory that 500 or more locomotives on these lines are now equipped with vanadium cylinders. The test of the original pair demonstrated that over twice the normal mileage of ordinary cast-iron cylinders could be obtained from vanadium cylinders before reboring. The loss in casting these cylinders was extremely low, no cylinders being scrapped at all for the first 183 locomotives ordered.

The same sort of results have been reported by a builder of large gasoline engines for marine use. In his particular case the number of cylinders formerly lost when tested after machining was excessive, due to porousness of certain sections.

Piston rings and cylinder and valve bushings of vanadium cast iron have been thoroughly tried out and adopted by a number of leading railroads on account of the greatly increased length of service obtained. A very interesting application of vanadium has been for glass-bottle molds. These are chill castings and are usually covered with engraving. The iron has to be clean, close and uniform in grain, machineable and take a good polish. The first vanadium cast-iron bottle molds were made a little over a year ago and proved not only to have a longer life, but it was found possible to increase the speed of the bottle machines as the glass did not appear to stick to these molds to any extent. This can be readily accounted for by the uniformity of the grain and freedom from mottled spots.

One of the first applications of vanadium to cast iron was in chilled iron rolls for sheet mills. A number of these were made and gave excellent results, some of them showing 50 per cent greater tonnage with one-half of the dressing against the regular chilled roll. The question of cost, however, entered in here in an unusual manner. Fully 65 per cent to 70 per cent of the roll breakages are due to accidental causes and no amount of vanadium would insure, for instance, against a roller letting a pair of tongs pass through the rolls.

We have had opportunity from time to time to make comparative tests of the strength of cast iron with and without vanadium extending over a considerable period, and believe that these comparisons are more nearly representative than single tests, as they represent average conditions.

The following tests are the average for ten consecutive days' work on locomotive cylinders. The transverse tests were made on bars 1 in. square, 12 in. between supports; the bars were machined all over and consequently were absolutely comparable, which is not always the case with bars only brushed or cleaned.

	Transverse.	Tensile
Plain cast iron	2,130 lb.	24,225 lb.
Vanadium cast iron	2,318 "	28,728 "

This iron has stood 750-lb. water pressure for sections ¼ in. thick. The average chemical analysis of this iron is about as follows:

Combined carbon	0.50
Silicon	1.50
Manganese	0.45
Phosphorous	0.600
Sulphur	0.095

I endeavored to get together a series of tests on irons of various compositions to show the effect of vanadium both on the strength and on the chilling qualities, but owing to various delays I am not able to present this in complete form. The following tests were supposed to be from iron from the same ladle, before and after adding vanadium, but the chemical analysis shows that such could not have been the case. The difference in composition, however, so far as it influences the physical qualities of the iron are all in favor of the iron without vanadium. In addition to the bars for tensile test, chill blocks were cast to show the effect of vanadium on the depth of the chill.

CHEMICAL COMPOSITION.

	Without Vanadium.	With Vanadium.
Graphitic carbon	2.34	2.48
Combined carbon	0.79	0.71
Silicon	1.83	2.21
Manganese	0.43	0.424
Phosphorus	0.508	0.546
Sulphur	0.102	0.071
Vanadium		0.056

PHYSICAL TESTS.

	Without Vanadium.	With Vanadium.
Tensile strength	34,560	35,600
Depth of chill	No white chill	1/16 in. white chill

Size of chill test block 4½ in. square by 1½ in. thick.

The difference in the machining quality of the two irons was decidedly in favor of the vanadium iron. I need hardly mention that the cast iron treated with vanadium should be at least approximately of the proper composition for the particular class of work to which it is intended.

The usual amount of vanadium added to cast iron is 1 10 per cent for cupola melted iron, equivalent to about 5 ounces of ferro-vanadium for each 100 lb. Some foundries advocate the use of a higher percentage, up to 15 per cent, which is about as much as can be conveniently added in the ladle. As low a percentage as 0.05 per cent will produce distinctive effects.

Owing to the high melting point of vanadium and the limited reserve heat available in a ladle of iron it is advisable to use the alloy in a finely crushed condition, or even powdered, when the amount of iron treated is small, and to use an alloy of low melting point. The iron should be tapped out as hot as possible and a ladle used that has just been emptied in order to conserve as much heat as possible.

After the bottom of the ladle is covered with a few inches of iron the finely crushed or powdered ferro-vanadium is added by sprinkling on the stream as it flows down the spout to the ladle; in this way advantage is taken of all the available heat and also of the mixing effect of the stream as it strikes the iron in the ladle. After the vanadium is added the contents of the ladle should be well stirred and allowed to stand a few moments in order to insure thorough incorporation and complete reaction.

In the case of the higher-grade air-furnace iron, with its reserve of available furnace heat, the procedure is very simple. After the charge is melted and 15 to 20 minutes before tapping the ferrovanadium is added and the bath well stirred or

rabbled. The addition of 0.18 per cent to 0.20 per cent vanadium is recommended in this case, equivalent to 10 ounces to 11 ounces of 35 per cent ferrovanadium per 100 lb. of metal.

The American Vanadium Company has lately developed a ferrovanadium containing about 30 per cent to 35 per cent vanadium, 10 per cent to 15 per cent silicon, 5 per cent to 10 per cent manganese and 2 per cent to 5 per cent aluminium, which melts and dissolves readily in molten cast iron and has been giving very satisfactory results. In distinction from standard low silicon "Amervan" ferrovanadium this new alloy is called "Masvan."

Melting Non-Ferrous Metals in an Electric Furnace.

By CHAS. A. WEEKS, M. E.

In an address by Dr. J. W. Richards, of Lehigh University, at the convention of the American Brass Foundrymen's Association, held at Detroit, June, 1910, the possibilities of melting bronze and brass in an electric furnace were compared in a theoretical way with the present methods of melting these metals in oil-direct-fired and crucible coke or oil-heated furnaces.

For several years I have been investigating the use of an electric furnace for melting copper, monel metal, bronze, brass and zinc refining, in large units and on a commercial scale, comparing the costs of melting in my electric furnace with other methods of melting now in use, along the lines of Dr. Richards' address.

In order that this work be done in a large and original way, I designed and built an electric furnace with a crucible capacity of three tons of copper. My knowledge of non-ferrous metals melting caused me to take rather radical steps to prevent the copper and other volatilizable metal from being volatized as they would be in the general arc type of furnace. To prevent this burning of metal I designed a furnace which may be classified under the arc type, but in which the heat is transferred to the metal by radiation from the arc, together with some reflection and conduction from walls and roof of furnace. The furnace is entirely enclosed and the atmosphere is neutral, the furnace works very smoothly, heating any size of charge up to furnace capacity; it is built economically and with no lining complications, it is as simple as an open-hearth furnace.

The furnace used in making these runs was also provided with a condensing system for the treatment and recovery of zinc from zinc dross and zinc ores containing copper, iron, lead, etc. While unfortunately I was not able to treat ores at the time of running the furnace during July and August, 1910, yet at that time I made a very good zinc from impure zinc dross, and during the run made some 1600 lb. of zinc and no blue powder. Some zinc oxide was produced at vent of condenser, but this was wanted and expected. It is my belief that this is the largest amount of zinc produced in an electric furnace, without making blue powder, made at a commercial cost. The run was stopped at that time by reason of making more zinc than the condenser could handle.

Copper Melting.

A number of runs were made melting copper scrap in which charges of about 2000 lb. were melted. To melt and superheat to a temperature above 1600° C. from cold furnace required 600 kw-hours, and the metal was poured in 3½ hours. This temperature was considerably in excess of copper melting, using more power and longer time. The loss of copper with new lining was less than 1 per cent.

Melting another charge of 2000 lb. to temperature of 1200° C. required from cold furnace about 500 kw-hours, time three hours. Under operating conditions, that is, with a hot furnace and using charges of two tons or more, this figure would undoubtedly be cut down to 300 kw-hours or less per ton, metal heated to pouring temperature of 1200° C., with melting time about 2½ hours.

As no slag complications are involved, the furnace can be tapped into as many receptacles as wanted without power interruption; the copper can be poled at will and resulting tough pitch copper can be rested in the furnace indefinitely without becoming oxidized, since a blanket of charcoal may be kept floating on the metal to protect it from such slight amounts of air as do enter a practically tight furnace.

This furnace is entirely free from the very objectionable vaporization of copper as experienced with those furnaces in which the current arcs from the electrodes to the bath, with resultant danger to the operator, as noticed in the letter of Mr. C. A. Hansen in the February, 1911, number of this journal.

The above figures will apply to melting monel metal, the proper pouring temperature of which is around 1400° C. The melting of monel metal in an open-hearth or reverberatory furnace or in crucibles allows a very serious oxidation of the metal; as a general result the castings made are porous for high-pressure work. It is only necessary to say that melting this metal in a non-oxidizing furnace, such as my electric furnace, where the metal cannot be burnt or oxidized, the making ing large sound castings from this metal will be accomplished.

Bronze and Brass Melting.

In his address Dr. Richards gave 1 lb. of coke = 0.67 lb. of oil = 1.8 hp-hours or coke at $4 per ton = oil at $6 = power at $12.25 per hp-year. I have taken power at 1 cent per kw-hour and would say that while this cost seems low a well-known oil-engine concern will guarantee a cost of about ½ cent per hp-hour. Any large electric plant can sell power at this price at a profit for day load. In order to obtain some idea of the causes controlling oxidation and cost comparative runs on gas, coke and oil-fired furnaces show that oxidation varies approximately as the length of time required to melt; also in proportion to the square of the temperature. On this basis, comparing the electric furnace with direct oil-fired and coke-fired crucible furnaces, I have assumed for the former an oxidation of 0.7 per cent (actually 0.5 per cent should be taken).

Taking the amount of power necessary to melt one ton of bronze or brass at 150 kw-hours, I find a reasonable figure to be 190 kw-hours per ton in 1½ hours time.

For comparison with the above figures I will take figures given in a test of rival direct-heating oil furnaces. This test was made under the direct supervision of one of the best-known brass foundry superintendents, who is in charge of one of the largest foundries in the country.

This test was fair and the figures are presumably correct. The metal mixture was 66 parts of copper, zinc 34 parts, part scrap, gates, etc. Pouring temperature, 2135° F.; loss of metal through oxidation, 1 1/3 per cent; fuel, 36 gal. of oil per ton metal melted, at 3 cents per gal.; cost of lining, taken at 10 cents per ton.

Coke-fired furnace; coke $4 per ton; melting 8 lb. of metal to 1 lb. of coke, with oxidation 1 per cent; cost of crucibles $3 per ton metal melted, with copper at 13 cents per lb., zinc 5 cents per lb.

We thus get the following:

	Electric furnace.	Direct oil-fired furnace.	Coke-heated crucible furnace.
Cost of heat	$1.90	$1.08*	$0.50*
Oxidation	1.44	2.68	2.05
Linings	0.10	0.10	0.10
Crucible	0.00	0.00	3.00
	$3.44	$3.86	$5.65
Time of melting	1½ hours	2 to 2½ hours	2½ to 3 hours

While the electric furnace shows a lower cost and larger output, as above, melting in two-ton charges, the power cost

*Furnace builders' claim.

would be cut to 150 kw-hours or $1.50 per ton metal melted in 1½ hours.

Interest, depreciation or labor are not taken into consideration in above.

As is well known, copper, monel metal, bronze or brass cannot be melted in any type of fuel-fired furnace without oxidation, as the average crucible at red heat is metal-tight, but not gas-tight, gases readily finding their way through the pores of the crucible material. We also know that carbon dioxide gives up oxygen very readily to many metals, especially to zinc, being reduced to carbon monoxide, so that the fact of my electric furnace being sealed and non-oxidizing, makes it particularly valuable in melting non-ferrous metals.

The best furnace is the one that produces the best metal, from which the greater number of perfect castings can be made, in which the temperature is easily controlled, and in which a neutral atmosphere can be maintained for any length of time at required temperature without oxidizing and little loss of metal, in which metal can be treated or alloyed with large output without the production of deleterious fumes and with perfect safety to operator, I believe there can be no question as to the superiority of my arc-heat radiating furnace for the melting of non-ferrous metals in these respects. Compared with the direct oil-fired, coke or oil-heated crucible furnaces, it certainly makes better metal at a cost much less than was thought possible commercially. I here quote a paper of Dr. Richards and discussion as to actual present cost of melting yellow brass.

Efficiencies of Furnaces.

In a paper presented by Dr. J. W. Richards at the Pittsburgh meeting of the American Brass Founders' Association in May, 1911, the author explained what was meant by efficiency of industrial appliances, and cited examples of carelessness in describing the efficiencies of various products. Applied to furnaces, efficiency often means ratio of fuel to output.

Mr. Webster asked for information relative to the melting ratio of brass furnaces and stated that in using anthracite coal the usual ratio was from 2½ to 3 lb. of metal melted per pound of fuel. The question was then asked how this compared with regular practice, and Mr. W. M. Corse stated that with coke the average is 45 lb. per 100 lb. of metal = 2.25 lb. of metal per pound of coke.

Mr. H. P. MacKinnon, of the Hersey Manufacturing Company, South Boston, Mass., stated that he had melted 900,000 lb. of bronze for light castings, at an average consumption of 3¾ gal. of oil per 100 lb. of metal in a tilting furnace, and that approximately one-ninth of the metal was in form of borings. The cost of oil averaged 18 cents for 4½ gal., 75 gal. of oil melted 2000 lb. of metal at cost of 4 cents per gallon.

Mr. John A. Smith, of the American Air Brake Company, stated his fuel consumption was 3 gal. of oil per 100 lb. of metal in an open-flame furnace or 60 gal. per ton.

Fuel cost of melting 2000 lb. brass as above:

Coke, 900 lb., at $4 per ton	$1.80
Oil tilting crucible furnace, 75 gal., at 4 cents	3.00
Oil, open-flame furnace, 60 gal., at 4 cents	2.40
C. A. Weeks electric furnace, 1 cent per kw-hour, 190 or 150 kw-hours	$1.90 or $1.50

Temperature of yellow brass ready for pouring, 2000° F.

Refining of Zinc from Impure Zinc Dross in an Electric Furnace.

Some years ago several plants were erected, using methods and furnaces from my designs, for the work of recovering a pure zinc from impure zinc dross, purchased in open market, without selection.

The requirements of these plants were that the zinc recovered should be equal to the well-known horse-head brand of zinc in every way. The cost of refining to be low, production regular and furnaces of long life. I am glad to say that these conditions were met in every way, particularly as to furnace upkeep. The repair account in some five years has been less than 1 per cent of the first cost. As these furnaces have been under fire with heat high enough to volatize zinc, through retorts for a year at a time, they show fair construction. While the above is not under discussion I state it to show that I have some knowledge of this subject.

The greatest cost and uncertainty of the above process are the retorts, which are of small capacity, the largest size of retort holding 1000 lb. The cost of retorts per ton of dross charged is from $5 to $8 and in some cases where retorts are not properly treated the cost is greatly increased.

With a view of obtaining, if possible, a better product with larger charges, and in some cases to use a more impure dross and do away with retorts, I was led to design an electric furnace for this work, in which I could charge two to three tons, with an output of not less than 200 lb. of refined zinc per hour.

When I took up the construction of this furnace, which is of the arc type, from which heat is transferred to the charge by radiation from the arc, with some reflection and conduction from the furnace walls and roof, at that time (1905-6) the statement was made that all attempts to refine zinc metal from either zinc ores or zinc dross in presence of an electric arc had been failures, as the vapors were so superheated by the intense heat of the arc that it had been impossible to condense them; further, that when the zinc vapors did condense they produced blue powder. In this relation I would say that from the very first attempts I made to refine zinc dross, some ten years ago, I have never made during this entire period more than 5 lb. of blue powder in actual working.

As I was not particularly interested in blue powder and did want zinc metal, I was not interested in the failure of others. I built the furnace for this work, arranged to use a condenser, which I had having a capacity of 35 to 50 lb. metal per hour, intending to build a larger condenser later, with capacity of 300 lb. metal per hour.

After preliminary heating to dry out the lining and testing the working of furnace, which was found to operate very smoothly, we made several runs on zinc dross, and in the last run obtained over 1000 lb. of zinc metal. But owing to insufficient condensing capacity with an average output of 35 lb. per hour, and zinc being made so fast that metallic zinc jammed everything up, the run was stopped.

During the run no blue powder was made, liquid zinc coming off regularly for some 36 hours; during this time there was no adjustment or appreciable loss of electrodes.

To give some idea of the furnace working, I would say that with 80 kw input 35 lb. of zinc was obtained per hour; of this, by reason of the excessive overheating of the furnace, due to the small amount of zinc taken off, about 65 kw was lost by radiation and water cooling, leaving 15 kw available for distillation.

The cost of zinc production compares more than favorably with the present methods of zinc distillation, with the great advantage of treating dross on ores containing iron, copper, lead and other metals in large amount. While no attempt was made to obtain a great purity of zinc during these runs, nor using methods of refining as in my retort system, the zinc produced from the most impure zinc dross I have ever seen was more than good, as the following analysis shows:

Electric Furnace.

Charged Zinc Dross.		Refined Zinc Output.	
Lead	2.00%	Iron	.024%
Iron	6.50%	Lead	.666%
Zinc	85.50%	Cadmium	.105%
Undetermined	6.00%	Zinc by difference	99.205%

Retort Furnace.

Average Zinc Dross.		
Lead	.52%	Average zinc produced by my other process; zinc by difference 99.75 to 99.90.
Cadmium	.22%	
Iron	3.36%	
Zinc	95.90%	

From these analyses it will be seen that refining of zinc dross

into high-grade zinc in an electric furnace is a success. I believe that these runs made during July and August of 1910 produced the greatest amount of zinc metal made in an electric furnace without blue powder. As there was no change in the furnaces or condensers from their first design in 1905, there is no reason why these results could not have been obtained at that time.

Philadelphia, Pa.

The Phase Rule in Cyanidation.

BY H. B. LOWDEN.

It is not the purpose of this article to enter upon a scientific discussion of the phase rule as applied to the reactions occurring in the cyanide process, but rather to point in a general way to its bearing upon the conduct of operations in practical work. The object will be served by a consideration of the reactions between gold and an alkali cyanide if it is borne in mind that the influences at work in the treatment of silver ores by cyanidation are governed by the same laws.

Gold in ores when in contact with cyanide solution dissolves at a rate which decreases as the solution increases in precious metal content and becomes reduced in free cyanide. This decreasing rate of dissolution, analogous to the diminishing rate at which a salt dissolves in water as the point of saturation is approached, is well known, but if viewed in the light of the phase rule the reason for it is made clearer and the application of cyanide solutions to ores might in many cases be made more efficient.

Bearing in mind that dissolution will proceed until an equilibrium is reached between the undissolved metal, the metal in the solution and the free cyanide, and that as this equilibrium is approached the velocity of the reaction becomes reduced, it is easy to account for the relatively slight increase in extraction secured by prolonging treatment, as well as by the addition of fresh cyanide to solutions after being for some time in contact with the ores: the latter by reason of the fact that although the free cyanide is increased thereby, the alkali auricyanide remaining in the solution continues to exert an important influence on the velocity of dissolution. Were the alkali auricyanide removed from the field of the reaction, the velocity of dissolution would be increased by reason of the removal of that phase which, from its tendency to undergo the reverse reaction, exerts a detrimental effect by keeping the whole system near the point of equilibrium. An immeasurably small concentration of auric cyanide would suffice to bring the dissolution to a standstill were it not removed as rapidly as formed, by dissolution in the excess of alkali cyanide present with formation of the complex alkali auricyanide. Although this removal of auric cyanide is what renders the cyanide process possible, the fact should not be lost sight of that we have in the formation of alkali auricyanide only a system in which the permissible concentration is relatively large, and that while the tendency to the reverse reaction is, from a commercial standpoint, often negligible, there are nevertheless instances in which it should be given serious consideration.

A discussion of all the influences affecting these reactions, including those brought about as a result of working with such impure substances as ores, is unnecessary, and nothing would be gained by introducing complications which would only befog a clear view of the subject. The application of the phase rule points to the separation of ore from solution when the velocity of dissolution becomes low and the application of fresh solution in which the concentration of alkali auricyanide is small, and to the eventual attainment of a continuous cyanide process based on the principle of counter currents, in which the richest solution is in contact with the richest ore and the solution most free from alkali auricyanide is in contact with the ore after the dissolution of the greater portion of the values.

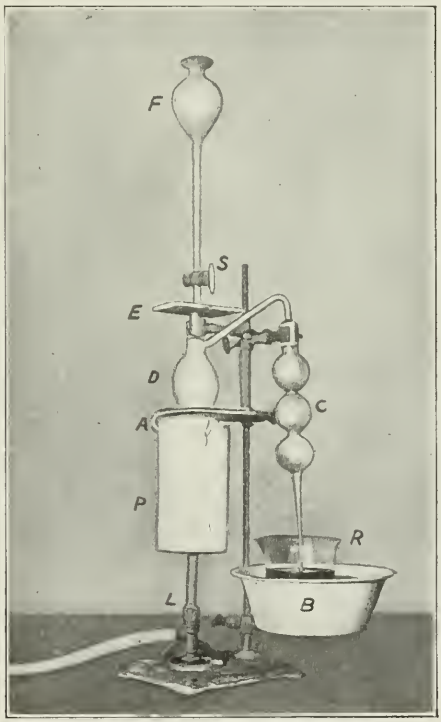
Denver, Col.

Determination of Arsenic and Antimony in Anode and Cathode Copper.

BY EDWARD F. KERN AND CHING YU WEN

In reviewing the literature on this subject, several methods were found and tried but none were sufficiently accurate for the separation and determination of arsenic and antimony in cathode copper of high purity. Some of these methods, which are in more general use, accomplish the separation of the arsenic and the antimony by distillation of the arsenic as arsenious chloride, which begins to distill at about 110° C.

These methods are in brief as follows: A sample of the copper is dissolved in concentrated nitric acid, after which about 1 gram of ferrous sulphate crystals is added, the nitrous fumes boiled off and the solution diluted. The iron, arsenic and antimony is precipitated with the addition of an excess



DISTILLATION APPARATUS.

of ammonia; thus, the arsenic and antimony is separated from the large quantity of copper, and are then separated from each other by distilling the arsenic as arsenious chloride.

These methods differ principally in the treatment of the iron precipitate, in the reducing agent used for the reduction of the arsenic, and in the procedure of the distillation.

Of these methods, the first that may be mentioned, was that which practically got rid of all the copper in the ferric hydroxide precipitate, first, by dissolving the iron precipitate with 50 c.c. of hot dilute solution of hydrochloric acid (1 acid to 5 water) through the filter, and precipitating it with concentrated NH_4OH while hot. The re-precipitated precipitate was dissolved with 20 c.c. of concentrated hydrochloric acid through the filter, and filter paper was washed with the same amount and kind of acid. This was then transferred into

the distilling flask which had previously contained 15 grams of ferrous sulphate, and distillation was made according to the method given by E. H. Miller,[1] which consisted of three intermittent distillations, using 50 c.c. concentrated hydrochloric acid each time.

The results obtained varied and were always low and showed that ferrous sulphate did not appear in this case to be a satisfactory reducing agent. Besides, the distillation, with ferrous sulphate as reducing agent, was difficult to control, because, as the solution became concentrated, it often happened that violent bumping was inevitable, sometimes so violent as to cause the distilling flask to crack

The combined reducing agents, composed of stannous chloride and ferrous sulphate, were next employed, and the object of using the former salt was to decrease the amount of the latter. The solution containing iron, arsenic and antimony was first reduced by adding, drop by drop, a saturated solution of stannous chloride in concentrated hydrochloric acid until it became colorless, and then transferred into the distilling flask containing 7 grams of ferrous sulphate crystals. The distillation was made as before. But this also proved unsatisfactory in that the results were low, varying from 15 per cent to even 50 per cent.

The unsatisfactory results in this case were, perhaps, due to the excess and strong action of stannous chloride, which caused over-reduction of the compounds to their metallic state and these reduced metals remained undissolved even in a concentrated solution of hydrochloric acid, as black particles were observed in the flask during and after the distillation. Another objection to the use of stannous chloride was that it complicated the determination of antimony. A. E. Knorr[2] thinks that stannous chloride is not a good reducing agent to use for the determination of arsenic, because of the volatility of chloride of tin.

Both sodium thiosulphate and ferrous sulphate were also tried, singly and combined. The iron precipitate with the filter papers, in this case, was treated with 20 c.c. of concentrated nitric acid in a 350 c.c. casserole covered with a watch-glass. It was digested and evaporated on a hot plate until it became a pasty mass. After addition of 3 grams of potassium bisulphate ($KHSO_4$) and 10 c.c. of concentrated sulphuric acid, it was carefully heated over a free flame until fume of sulphuric anhydride was strongly given off. The solid mass, after being cooled, was taken up with 30 c.c. of concentrated hydrochloric acid and distilled as before, using either 1 gram of sodium thiosulphate, or a mixture of 0.7 gram of sodium thiosulphate and 7 grams of ferrous sulphate crystals.

The low result obtained by this method might be due partly to the loss of solution by spattering during evaporation to dryness and partly to the interference of the sulphur dioxide which was liberated by the decomposition of the sodium thiosulphate. A saturated solution of sodium thiosulphate, instead of solid sodium thiosulphate, was also tried and the reduced solution in the distilling flask was boiled for a few minutes before addition of the hydrochlorous acid, in order to expel any sulphur dioxide that might be present. But this procedure also failed to give satisfactory results, they being low, and varying from 20 per cent to 60 per cent.

Thus it is seen that the methods mentioned above did not give satisfactory results, so other reducing agents were tried, with the outcome that the following modified method, which proved to give accurate results, was finally adopted. It is as follows

A sample of about 10 grams, in the case of anode copper, and from 30 grams to 10 grams, in the case of cathode copper, depending upon the amount of arsenic it contained, was weighed out and dissolved in a No. 6 beaker with concentrated nitric acid (45 c.c. for 10 grams of copper). The beaker was covered with a watch glass and warmed on a hot plate in order to hasten the dissolution of copper. When the solution was complete the beaker was removed from the hot plate and about 1 gram of ferrous sulphate added. The beaker was again heated to expel the red fume. The solution was then diluted to 400 c.c. or 500 c.c. and warmed. The iron, arsenic and antimony was precipitated with strong ammonium hydroxide. Sufficient excess was added to dissolve all the copper compounds.

The precipitate was allowed to settle, decanted, filtered while still warm and finally washed with a hot solution of ammonia (10 water to 1 ammonium hydroxide, sp. gr. 0.9). The precipitate, together with the filter paper was transferred into a No. 1 beaker covered with a watch glass, and digested gently on the hot plate with 20 c.c. of concentrated nitric acid, until practically all the red fume was driven off. (Care should be taken not to heat the solution vigorously lest some of it would be lost by spattering.)

After cooling the solution 10 c.c. of concentrated sulphuric acid was added and the beaker was re-heated on the hot plate till fume of sulphuric anhydride freely evolved. It was then allowed to cool and about 5 c.c. of a 10-per cent solution of hypophosphorous acid, or 6 c.c. of a 20-per cent solution of potassium hypophosphite was added (provided the solution did not contain more than 0.3 of a gram of arsenic).

The solution was again heated on the hot plate until all the excess of the hypophosporous acid, or potassium hypophosphite was destroyed; this was indicated by the evolution of strong fume of sulphuric anhydride. (In order to be sure of destroying the excess of the reducing agent let it fume for at least one-half an hour.)

The content in the beaker after cooling was transferred into the distilling flask (*D*, Fig. 1), the beaker was then rinsed twice with only 10 c.c. of water. The distilling flask was gently heated to boiling, with a smoky flame, in order to expel any sulphurous-acid gas that might be present in the solution. Bumping of the solution sometimes occurred and was prevented by imparting a rotary motion to the solution. It was then allowed to cool. After cooling any solution that was left in the beaker was rinsed into the distilling flask twice or three times with 30 c.c. of concentrated hydrochloric acid, making a total of not over 10 c.c. water and 40 c.c. concentrated HCl.

The apparatus, as is shown in Fig. 1, was now connected and adjusted: the receiver *R* (a No. 4 beaker), containing about 250 c.c. of water, was placed in a bath of cold water, *B*, and under the condenser, *C*, the tip of which was immersed into the water to a depth of about ⅛ in. Ten c.c. of concentrated hydrochloric acid was poured into the funnel, *F*, and allowed to run slowly into the distilling flask, *D*, through the stop-cock, *S*, in order to drive off the air in the lower part of the stem of the funnel, which would disturb the regularity of the acid feed. The stop cock was closed and the funnel filled with 80 c.c. of concentrated hydrochloric acid. Now heat was applied, first with a smoky flame, and when the solution began to boil was gradually increased until a proper flame was adjusted.

The hydrochloric acid in the funnel, *F*, was now allowed to run into the flask, drop by drop, at such a rate that the liquid volume in the flask might be during the entire distillation, maintained approximately constant. (The rate was, usually, three drops per two seconds.) When the acid in the funnel, *F*, was nearly exhausted, the distillation was complete. The receiver, *R*, containing the distillate was carefully removed and then the flame was turned off. The content in the distilling flask was left in place to cool and reserved for the determination of antimony.

The distillate was transferred into a No. 6 beaker and nearly

[1] "Quantitative Analysis for Mining Engineers," E. H. Miller, ed. 1904, p. 116.
[2] Private communication.

neutralized with a strong (30 per cent) solution of potassium hydroxide and the neutralization completed with a saturated solution of sodium bicarbonate, the addition of an excess of 70 to 80 c.c. being necessary. The neutralization was done by placing the beaker in a bath of cold water, in order to keep the solution cool while the neutralization was taking place. The neutralized solution was then titrated with a standardized solution of iodine (N 10.) Starch solution was used as indicator.

Precautions in the Distillation Method of Arsenic.

The analysis of arsenic by the above method requires careful manipulation. In order to do it successfully it is of much importance that the quantity of hypophosphorous acid, or potassium hypophosphite, added for the reduction should be limited to as small amount, as given in the above procedure; otherwise it reduces not only the compounds of arsenic and antimony to their metallic state, but also those of other metals, such as iron, copper, etc.

Over-reduction, according to A. E. Knorr[3], would fail to give satisfactory results, even though the metallic arsenic and antimony would afterwards re-dissolve in the concentrated hydrochloric acid solution. It is also important that complete destruction of the excess of hypophosphorous acid, or potassium hypophosphite, should be accomplished before the transference of the reduced content (solid salts and solution) into the distilling flask takes place. Any of either reducing agent remaining undestroyed will interfere with the determination and the result will in all cases be low.

The advantages of having a continuous acid-feed, during the entire distillation, are great and manifold. In the first place, the temperature may be properly regulated, secondly, the strength of the hydrochloric acid in the distilling flask may be maintained to prevent the reversible action, which may be shown by the following chemical equation:

$$As_2O_3 + 6HCl \leftrightarrow 2AsCl_3 + 3H_2O.$$

and, finally, a more rapid and effective distillation may result and, therefore, much time may be saved.

In order to secure a steady flow of the acid-feed, a grooved rectangular board of asbestos, *E*, is placed on the mouth of the distilling flask, so as to prevent the heating of the stop-cock, which would, otherwise, cause irregularity of the flow. Back-suction is the chief cause of failure. It is caused either by draught or by variation in the flame, which supplies insufficient heat. To prevent the former it becomes necessary to protect the flame from draught by suspending under the ring, *A*, an asbestos cylinder, *P*, 4½ in. high and of the same diameter of the ring. To prevent the latter, the flame should be carefully regulated from time to time. In this way back-suction is less liable to occur and success may be insured. If the distillate is once sucked into the flask, re-distillation proves absolutely fruitless, because arsenous chloride refuses to distill in such dilute solution.

The distillation apparatus was made by E. Machlett and Sons, glass-blowers, at 143 East 23rd Street, New York City. It is known as the Knorr Arsenic Distillation Apparatus.

Test of the Above Methods.

The accuracy of the above method was tested according to the following procedure: A weighed sample of about 0.1 gram of c.p. arsenous oxide was dissolved in 10 c.c. of sodium hydroxide. When the solution was complete 50 c.c. of concentrated nitric acid was added and then 20 c.c. of saturated bromine water in order to oxidize the arsenic to the higher state. The solution after an addition of about 1 gram of ferrous sulphate was heated to expel the excess of bromine, and after adding 15 grams cupric sulphate to the solution it was diluted to about 300 c.c. and heated to boiling. The iron and arsenic was precipitated with concentrated ammonium hydroxide and the red precipitate was treated in the same way as above described.

[3] Private communication.

In testing this method both potassium hypophosphite and hypophosphorous acid were employed as reducing agents. In the case in which potassium hypophosphite was used the amounts of arsenous oxide taken for analysis were 0.1004 gram and 0.1014 gram, which corresponded to 0.076 gram and 0.077 gram of arsenic respectively, and the analyses gave 0.070 gram and 0.072 gram of arsenic. In the case where hydrophosphorous acid was used the amounts of arsenous oxide were 0.1027 gram and 0.1014 gram which corresponded to 0.078 gram and 0.077 gram arsenic while the analyses gave 0.070 gram and 0.071 gram of arsenic. The values in both cases approximated the calculated values of arsenic.

Determination of Antimony.

For the determination of antimony Weller's[4] method was adopted. After the separation of arsenic by distillation the content in the distilling flask was transferred into a No. 5 beaker and diluted to about 400 c.c. The antimony, together with the copper, was precipitated in warm solution with a stream of hydrogen sulphide which continued to pass until the precipitate settled down and the solution became clear. The precipitate was separated first by decantation and then by filtration, and washed three times with hydrogen-sulphide water. It was placed with the filter paper in a No. 2 beaker and treated for an hour at room temperature with 40 c.c. of potassium sulphide (10 per cent) solution. The copper sulphide was filtered off, washed three or four times with hydrogen-sulphide water and discarded. To the filtrate now contained in a No. 5 beaker 50 c.c. of dilute sulphuric acid (1 acid to 4 water) was added to precipitate the antimony. The yellow precipitate was filtered off and washed with hydrogen-sulphide water three times.

The antimony sulphide, together with the filter paper, was placed in No. 2 beaker, treated with 40 c.c. of concentrated hydrochloric acid and oxidized to the pentad state by adding, little by little, about one gram of potassium chlorate, and the solution was heated to expel the free chlorine. (During the heating, should the solution show a dark coloration, more potassium chloride should be added.)

After oxidation and the complete solution of the antimony sulphide, the filter paper was filtered off and washed three or four times with a hot dilute solution by hydrochloric acid (1 part acid to 3 parts water). The filtrate was evaporated to 50 c.c. so as to make the solution to contain a constant quantity of hydrochloric acid. Twenty c.c. of concentrated hydrochloric acid was added and the solution diluted from 600 c.c. to 700 c.c. After the addition of 3 grams of potassium iodide crystals the solution was thoroughly stirred until the potassium iodide completely dissolved. It was then titrated at room temperature with a standardized (N 10) solution of sodium thiosulphate, starch solution being used as indicator.

As the liberation of all iodine does not take place instantaneously it is, therefore, necessary to titrate the solution slowly, that is, the thiosulphate solution should be run into the solution drop by drop until the blue color disappeared at least for one minute. Should, in any case, the blue color return immediately, more thiosulphate solution should be added. *Metallurgical Laboratory, School of Mines, Columbia University, New York City.*

[4] "Quantitative Analysis for Mining Engineers," Miller, 1904, p. 106.

Metallurgy at Cobalt.—In the sixth annual report of the Nipissing Mines Company it is stated that the company has erected a small plant at the mine to test on a working scale a process devised by Mr. Charles Butters for the treatment of high-grade cobalt ores. This is in accordance with the desire of the company to find some means of reducing to bullion the silver in the high-grade ores without the expense incidental to a regular smelting plant. The present experiment is said to be satisfactory as far as tested and bullion is being shipped.

The Electric Iron Reduction Plant at Trollhättan, Sweden—I.

At the meeting of the Jernkontoret (the Swedish Association of Iron Masters) at Stockholm on May 31 a report was submitted by the engineers in charge of the electric iron smelting plant at Trollhättan, Messrs. J. A. Leffler and E. Odelberg; this report being the first official publication in regard to the construction of the plant, its operation and the results obtained during the first six months of operation.

The following abstract of this report will give an account of the construction of the plant and will in part cover the operation. In our next issue the report of the operation will be concluded and the results of the operation will be given.

There is no necessity to give here an account of the former work done in Sweden on electric iron ore reduction by the Grönwall, Lindblad and Stalhane (A. B. Elektrometall) process, as this has been fully covered in former issues and volumes of this journal. A concise historical review of the process, for which the British, United States, Mexican and other rights have been acquired by Electro-Metals, Ltd., 9½ Union Court, Old Broad Street, London, was given in our March issue (this volume, page 163). In the following we give simply from Mr. Leffler's paper an account of the experience with this process at the Jernkontoret plant at Trollhättan.

The most important points to be investigated at this plant were the folllowing:

The proper height of the furnace and other dimensions most suitable for different purposes and different charges; the most suitable design and way of building; consumption of electrodes and the shape and arrangement of electrodes; energy consumption per ton of iron of certain composition; coal consumption per ton of iron of certain composition; circulation of gas, its significance for the economy and means to accomplish circulation in case of choking charges; manufacturing of iron of different composition; conduct of various charges and concentrates and the influence of same on the economy; value of escaping gases; efficiency of the furnace; maintenance cost at the furnace.

Naturally, certain of the above questions cannot be answered definitely after six months of operation, but the experiments are to be carried on and questions which are now vague will undoubtedly have their final solution within a not far distant future. The observations made so far will greatly serve the industry as they have been made under almost ideal conditions, with greatest care and accuracy, by aid of the best instruments, and under the supervision of a highly efficient staff of engineers.

Location of the Plant.

The plant is located at Trolhättan on grounds owned by the Swedish government by agreement with the Royal Water Power Commission and will be retained until Oct. 1, 1912.

The distance from Trollhättan railway station is about 1 mile, the works being connected with this station with a normal-gage railroad track. Two sidetracks are laid to the site, one being for transport of fuel to the fuel storage house and the other for transport of other materials to the works and of iron from the works.

The fuel is charcoal, which is conveyed from the railroad trucks into the storage house by two band conveyors of cold-rolled steel. From the storage house the charcoal is elevated to the furnace top in suspended baskets running on a wire rope.

The ore is barrowed in the usual manner from ore heaps to the crusher plant, from which it is elevated to the furnace top in a truck running on an inclined track.

A narrow-gage track has been constructed for transport of iron and slag to the iron storage and slag tip respectively.

The *coal-storage house* has the following dimensions: Length, 48 m; width, 21 m, and height to the eaves, 10 m. It has a capacity of about 7000 cu. m, which approximately corresponds to six months' consumption. The charcoal is shoveled from the railroad cars into wooden hoppers under which the band conveyors run, conveying the charcoal into the storage house. The band conveyors from the hoppers to the storage house have an inclination of 1:4.4 (13°) and run in wooden troughs, the sides of which are removable so that the charcoal can be unloaded by means of a scraper at any desired point.

The steel bands are 400 mm broad and 0.95 mm thick. Each conveyor is made of three lengths of such steel bands riveted together and has a total length of about 170 m. A 16-hp motor is provided for driving, but the actual power required for one conveyor is about 7 hp and about 10 hp for both conveyors working simultaneously. The speed is 0.45 m per second, and the average carrying capacity of each conveyor is about 20 cu. m per hour.

Crushing Plant.—The dimensions of the jaws of the crusher are 550 mm x 300 mm. The machine is placed on a concrete foundation, so that the working floor is about 2.75 m above the ground level. The barrows are dumped outside the house into an elevator truck, which brings the ore up to an incline along which it slides into the crusher. The crushed material falls into a sheet-iron hopper. In this hopper a sufficient quantity is collected to fill the elevating truck to the furnace top. The door at the bottom of this hopper is operated from the working floor, and the man in charge of the crusher also regulates the elevator to the furnace top. A 40-hp electric motor is provided for driving the crusher and its elevator.

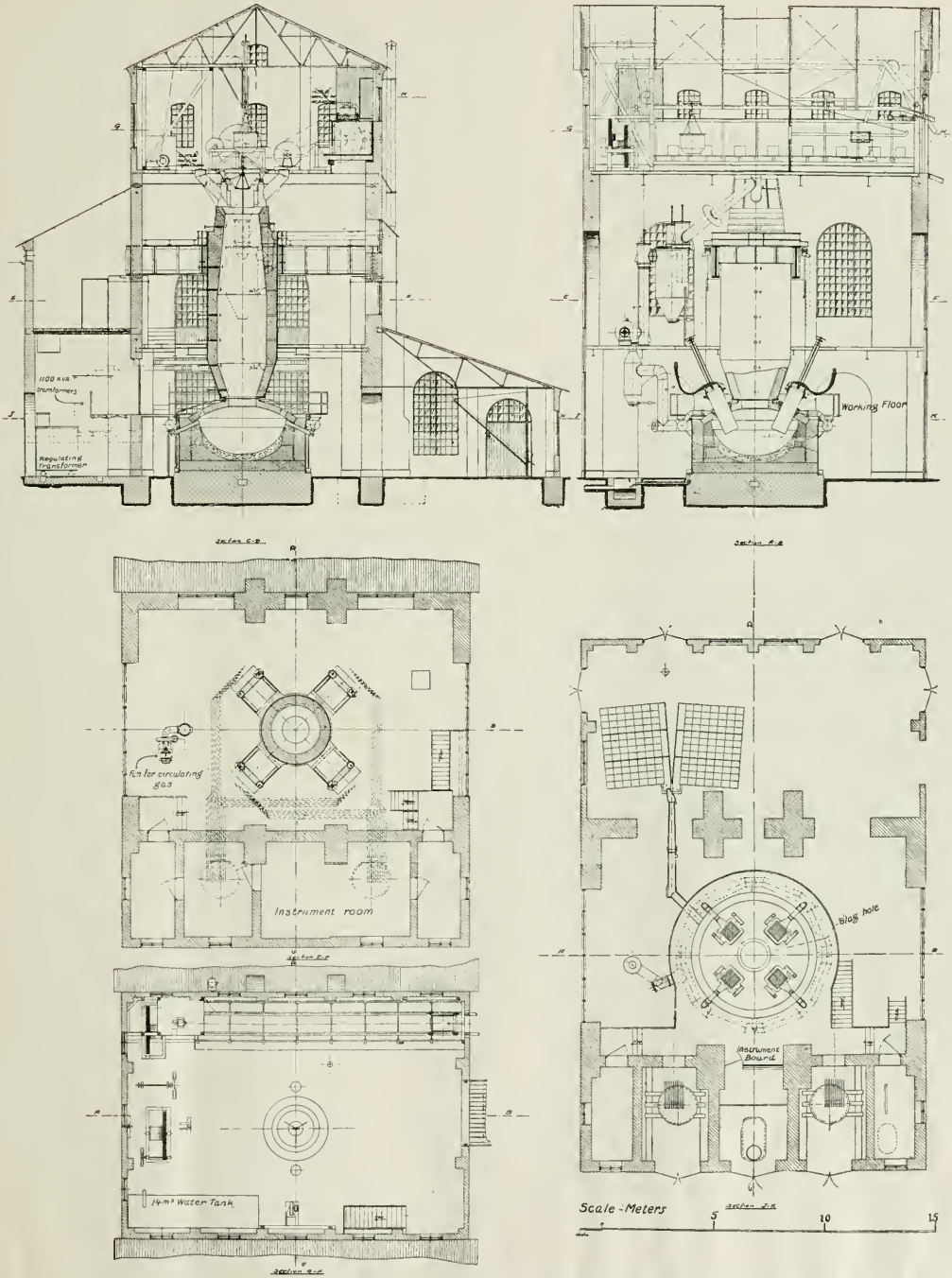
The *furnace house* has a floor space of 24 m x 15.5 m. The height from the ground floor to the furnace top is 13.7 m, and the total height to the eaves is 21 m. With regard to the electric installation, the high-tension apparatus and transformers are placed in a separate section of the building. From the two transformers for the furnace the low-tension conductors enter the furnace house proper through two openings in the wall.

The construction of the furnace is shown in Fig. 1. In this construction of furnace two separate portions are to be distinguished—that is, the shaft and the crucible or hearth. The shaft consists of a shell of steel plates 10 mm thick around the stack and 12 mm around the bosh. At the top it is riveted to an octagonal supporting ring of channel iron, on which the shaft is supported. The walls of the house on both sides of the furnace are built out at four places as suporting pillars. At a height of 9 m from the ground level they support two built-up beams, 1300 mm high, with a free length of 10.6 m. Around the furnace these are joined to form an octagon on which rests the channel-iron ring mentioned above.

The shaft is lined with Höganäs firebrick to a thickness of 360 mm in the bosh and 450 mm in the stack. At the top of the shaft is placed a Tholander charging bell and cone, which is raised and lowered by means of a capstan driven by a 2½-hp reversible three-phase motor. The cone is connected to the capstan by a 12-mm wire rope provided with a hook, which is removed between the charges so as to be out of the way of the coal baskets which pass close to the center line of the furnace.

The crucible rests on a concrete foundation. Like the shaft, it is surrounded by a sheet-iron shell 15 mm thick. At the top this shell is reinforced by a 200-mm x 18-mm iron band to take up the pressure of the arched roof. The lining of the hearth is of Höganäs and Stabbarp firebrick next to the shell and foundation with an inner lining of magnesite brick, while the dish-shaped bottom is formed of a mixture of magnesite and pitch rammed in so as to extend for some distance up along the walls. The roof is constructed of Ifö firebrick. The hearth and shaft are joined by means of a sand seal.

The four electrodes project through the roof in a slanting position at an angle of 65° to the horizontal. Each electrode is built up of four carbons of square total section of 660 mm x 660 mm. At the openings in the roof the electrodes are surrounded by cooling jackets of copper provided at the top with asbestos packing to prevent the leakage of gas. The connections to the electrodes are arranged at their upper ends and



ELECTRIC FURNACE PLANT FOR IRON ORE SMELTING AT TROLLHÄTTAN.

wedged between the electrodes and a holder of cast steel. This holder is supported on a frame movable up and down between two guides placed on each side of the electrode.

In this process gas is drawn by means of a fan from the gas outlets and blown into the crucible. Consequently the quantity of gas passing any given section of the shaft can be varied within certain limits by varying the speed of the fan.

This circulation of gas has two special objects:

a.—The gas blown into the crucible is there heated and gives off its heat to the charge in the shaft in passing up through it. Through this heating of the charge in the shaft a reduction of the ore by CO is facilitated, so that this gas is utilized to some extent for the process.

b.—The second purpose of the circulation of the gas is connected with the construction of the crucible, as the gas cools the roof and thus protects it from overheating. This cooling action of the gas is due in part to the absorption of heat, raising its temperature, and in part to the decomposition of CO_2 and H_2O in the gas in contact with the incandescent carbon in the crucible.

The arrangement of the gas pipes is shown in Fig. 1. From two gas outlets at the top of the shell the gas passes into a large dust-collecting chamber and from this chamber the circulating gas is drawn by the fan, which blows it into a circular pipe surrounding the furnace. From this pipe it finally enters the furnace through four tuyeres. The volume of gas corresponding to the gas formed escapes from the dust-collecting chamber through a pipe extending above the roof of the building and burns there.

In Fig. 1 four gas pipes are shown leading into the cooling jackets around the electrodes. These pipes were, however, soon disconnected, as it was found that the electrodes were attacked too strongly by the carbonic acid in the gas blown through them. The original intention of introducing the gas around the electrodes was to cool them as well as the surrounding brickwork.

The fan was designed for a speed of 2500 r.p.m. and a consumption of 8 hp and to deliver 70 cu. m of gas per minute against a back pressure of 325 mm of water, the temperature of the gas being assumed at 200° C. and its composition to be equal volumes of CO and CO_2.

The arrangement of tap hole, feeders and pig beds is also shown in Fig. 1. The iron is conducted in trenches of a total length of about 6 m to a small swinging spout in the foundry. There are six beds arranged in two rows of three beds each containing twenty-eight pigs. Each section is provided with its distributing feeder, to which the iron is conducted through the swinging spout. For tapping of the slag the crucible is provided with a slag hole from which the slag is tapped into slag bogies of ordinary construction. During a large part of the time the charge has, however, been high in iron and the quantity of slag correspondingly small. Under these conditions separate tappings of slag have not been made, but the slag which has accompanied the iron has been skimmed off and allowed to flow into a sand trench.

Among the arrangements at the furnace top the charging arrangement and provision for lifting the ore have already been mentioned.

The ore boxes are constructed of 2-in. boards. They are eight in number, and each has a capacity of 3.8 cu. m. Their total capacity is, therefore, about 70 tons of ore. A tipping wagon runs on a track above the ore boxes and into this wagon the elevator truck discharges the ore.

Two capstans are placed on the top platform, one for elevating the ore truck from the crusher and the second for bringing up the charcoal baskets. Both are driven by one 16-hp motor.

In one corner of the top floor is a water tank of 14 cu. m capacity. Water is pumped from the Stallbacka canal close to the works. The pumping plant contains two centrifugal pumps (one as a standby), each having a capacity of 1500 liters per minute. Of this quantity 625 liters per minute are for the requirements of the plant and the remainder is at the disposal of the adjoining works of the Trollhattan Elektrothermiska Aktiebolag.

The office building is located at the south corner of the site and also comprises the watchman's room, laboratory and storerooms. On the other side of the furnace house is a smaller building which contains a small repair shop and a shop for storing and fitting the electrodes. A grinding mill is also placed there.

The electric energy for the plant is obtained from the government's power station at Trollhattan and is three-phase alternating current of 25 cycles and 10,000 volts tension.

The two furnace transformers are each of 1100 kva with a guaranteed overload capacity of up to 1375 kva under one hour. They are coupled according to the well-known Scott method, giving secondary two-phase currents. Regulation is possible between 50 volts and 90 volts in eight stages and by different connections of the secondary coils, also from 100 volts to 180 volts. The voltage can be regulated during operation.

When the plans were first made for this installation there was some doubt whether such regulation should prove satisfactory and a special three-phase regulating transformer was installed to be coupled direct before the furnace transformers and thus giving these varying voltages. As soon as this plant commenced operations it was shown that the simple regulation direct on the furnace transformers was very satisfactory and the regulating transformer has never been used. This transformer has lately been exchanged for an 1100-kva single phase transformer designed to serve as reserve for both main transformers.

The low-tension conduits from the transformers are taken through the wall into the furnace-room. These conduits consist of six copper bars, 8 mm x 200 mm, for each electrode. These bars are supported on insulated clamps of brass. The energy is conducted to the four electrodes through 192 flexible cables (forty-eight per electrode). Each cable has an area of 185 sq. mm.

The instruments required for observing and regulating the working of the furnace and electrodes, together with all handles and hand-wheels for regulating the current, are placed on an instrument board on the working floor. In the so-called furnace office on the floor above are placed the self-recording kilowatt meter (showing the power consumption for the *furnace* only) and two kw-hour meters with a maximum indicator.

In the same room are placed two galvanometers for the eight thermoelectric pyrometers which measure the temperature in the furnace at the places indicated in Fig. 1.

At the side of each pyrometer hole on the furnace is a smaller opening which was used for observing the gas pressure and for collecting samples of gas.

Ordinary U gages with colored water are permanently connected to the furnace top, the discharge side of the fan and to a hole on the roof of the crucible. A "Mono" apparatus for a maximum of 60 per cent CO_2 records the percentage of CO_2 in the gas taken from the discharge side of the fan and the pressure is registered by a recording gage. The pressure due to the speed of the gas returning to the furnace is measured at an opening on the suction side of the fan, where the temperature of the gas is also determined by a mercury thermometer. The "Mono" apparatus, the gas calorimeter, the recording gas pressure gage, the speed gage and a gas-analysis apparatus of Sonden's type are placed in the room close to the furnace office, and are there easily accessible for the engineer in charge, who usually remains on the operating floor.

In electric pyrometers platinum-platinum-rhodium elements and silver-constant elements are used, the former in the lower and the latter in the upper part of the furnace. They are arranged to record the temperature every ten minutes. For

determining the temperature of the iron and slag on flowing from the hearth an optical pyrometer of Siemens & Halske's make is used. For the direct determination of the calorific value of the gas a gas calorimeter of Dr. H. Strache's, Vienna, construction is used. The quantity of gas passing through the gas pipes is determined by means of Krell's pneumometer and micromanometer. It has so far been impossible to use the recording instrument for determining the speed of the gas owing to the soot and dust carried over by it.

Cost of Construction.—In the estimate of Oct. 16, 1909, the total cost of the works was estimated at $60,700, and the association appropriated a sum of $69,000 for the installation.

The accounts show, however, that the actual cost amounted to $88,500. The excess in cost was caused by the whole construction being carried out in a more elaborate and substantial manner than was at first contemplated. The furnace house, for instance, was first intended to be constructed partly of wood, but has been built entirely of brick and steel construction, and the furnace shaft instead of being supported on a steel framework from the ground is supported on heavy beams resting on the brick walls.

Management of the Works.—The management of the plant is in the hands of a committee appointed by the association and consists of Major-General G. Geijer, Mr. Aug. Herlenius, chamberlain to H. M. the King, and Messrs. J. A. Brinell, chief engineer of "Jern-Kontoret," Axel Wahlberg and E. Odelberg, the last named acting as managing director for the committee.

At the works the management has been in the hands of Messrs. J. A. Leffler and E. Nyström, both engineers on the staff of "Jern-Kontoret," the former acting as general manager and the latter as superintendent.

The employees at the furnace consist of three men for charging, two for bringing up charcoal, four for crushing and bringing up ore, five furnacemen, three men for fitting electrodes, two for weighing iron and slag, one blacksmith, one blacksmith's helper, one electrician, five men for loading and unloading, one carpenter and one watchman and messenger.

The report will be concluded in our next issue, when the following subjects will be taken up: The principles of the reduction furnace; principles employed for calculating the form and dimensions of the shaft; charcoal, ore and limestone; electrodes and electrode consumption; the charges; the durability of the crucible and roof; iron and slag; composition and pressure of the furnace gas, and notes relating to working results.

Electric Furnaces for Molten Materials.

BY CARL HERING.

To design an apparatus as perfectly as possible one should first find what the desired ideals are and then endeavor to reach them as closely as practicable and in proportion to their importance.

The chief desirable ideal features of a commercial electric furnace for molten charges seem to be the following: Greatest possible economy in power; greatest possible rapidity of treatment, hence greatest tonnage per day for a given furnace; heating from the bottom instead of from the top; no higher temperature than that actually required; rapid and uniform distribution of the heat; rapid circulation to produce greatest possible homogeneity of product; largest possible surface of contact between the metal and the slag when the treatment is dependent upon the chemical action of the slag; heat transmitted from the metal to the slag rather than the reverse; greatest possible reduction of the troubles, cost and power losses due to the electrodes possibility of adding fuel heat; ease of control; reliability of operation; simplicity and cheapness of construction and operation.

The economy of power involves numerous features. The ideal method of heating is to generate the heat in the material itself, where it is wanted, as distinguished from generating it elsewhere and transmitting it to the material by radiation or conduction. There will then be no waste, as the efficiency of the utilization of the heat will be 100 per cent—that is, theoretically perfect. It is done in this way in the induction furnace, as also in all other resistance furnaces in which the material treated is itself the resistor. In the arc furnace whatever heat is generated in the arc itself and in the electrode can reach the material only by radiation and then only when the temperature is higher than that of the material.

To always pass the current through the whole charge means either a constant depth of bath, which excludes feeding with solid material, or else it means a provision for the regulation of the current through very wide limits, hence large and expensive machinery. For these reasons it seems preferable to pass the current through only a small portion of the charge, which portion is always constant and is independent of the depth of the bath. But this becomes practicable only when this small heated portion is continually and rapidly removed as soon as it is heated and is replaced by cooler material, otherwise it would soon be volatilized, which would break the circuit, injure most linings and possibly produce explosions. In other words, this is operative only when there is a circulation through the channels containing the portion of the charge which is being heated, and if such circulation is not very rapid, the furnace will necessarily either be a large slow-acting one, or the heated material will get too hot for the lining; circulation by gravity due to the heated material rising would be far too slow for a commercial furnace. The lack of means for producing a rapid circulation through such restricted resistor channels seems to have been the reason why this very desirable method had not come into more general use.

To further increase the heat economy the top surface of the charge and that exposed to the walls should be as small as possible for a given charge, as the losses of heat from an internally heated charge are due entirely to these surfaces and to the electrodes; they are less the smaller these surfaces are The ideal shape of the charge would therefore be a sphere; this, of course, is not practicable, but the shape should approximate to it as closely as possible, say, by making the charge the shape of a hemisphere if practicable instead of having the hearth large, flat and shallow.

These losses from the surfaces of the charge seem to be far greater than is generally supposed, hence in addition to reducing these surfaces as much as possible the wall thicknesses should be made as great as practicable, and of the best heat insulating material; bricks are cheaper than kilowatts, and they represent only a first cost, while the kilowatts lost are a continuous cost. There should also be as many joints between bricks as possible, transversely to the direction of the flow of heat, as joints seem to be excellent and cheap heat insulators and take up practically no space. The usual heat insulation of combustion furnace is very crude and wasteful and would be very inadequate for electrical furnaces in which those constant losses have a far greater money value.

Still further heat economies will be obtained the more rapid the treatment; these losses through the walls and electrodes evidently continue during the whole time of treatment, hence if its duration is reduced these important losses will be reduced in the same proportion. Hence it is very important that the heating should be done as quickly as possible and that everything possible should be done to hasten the various actions which bring about the melting and refining. If, for instance, it takes 50 kw merely to keep a ton of steel at the desired temperature during refining, which means that these losses amount to 50 kw per ton, then if the time necessary for treatment could be hastened so as to be only half as long, for instance, the energy required (in kw-*hours*, not in kilowatts) would evidently be only half as great. It is in this increase in rapidity of treatment that probably the greatest improvements in the reduction in power will be made in the future. It increases the

tonnage *per day* delivered by a given furnace. A small, rapidly acting furnace could produce the same tonnage per day as a large slow acting one, and the leakage losses of heat will be less per ton.

Attention need hardly be called to the fact that, as most hot materials rise it is far better, more rational and much more rapid to heat from the bottom instead of from the top whenever practicable. Heating a liquid by means of the arc, or by means of directly impinging gases, as in the open-hearth furnaces, must necessarily always be done from the top, hence the rapidity of the heating is then limited by the necessarily slow conduction of heat from the top to the bottom. Cases have occurred in which pieces of metal in the bottom of the charge were still unmelted, although the refining of the top of the charge had been completed. The ideal method is unquestionably to heat from the bottom. This becomes possible with some types of resistance furnaces.

Owing to these continuous losses, which are by no means small, the temperature ought not to be any higher than that required for the desired reactions, as these losses increase when the temperature rises. In the resistance furnace the temperature is generally within one's control, while with the arc it is the boiling point of the electrode material, hence generally far higher than required and the radiation is also to the sides and roof, where it is not only not wanted and is lost, but where it sometimes even acts destructively.

The more rapidly and uniformly the heat is distributed throughout the mass the more rapid will be the treatment and the less will be the leakage losses, as the temperature will be lower and the time of treatment less.

Homogeneity of the product is often of prime importance, particularly in steel refining. Complaints have been made that the metal at the bottom of a hearth of an arc furnace sometimes has a very different composition than that at the top. Moreover, an overtreatment of the metal at the top in order that after mixture with that at the bottom it will have the right composition, is a makeshift, and involves the element of chance to a great degree. Hence, circulation is very desirable.

Moreover, materials are already in use which when introduced into molten steel cause some particular refining action, and it is quite likely that more such materials will be forthcoming. When quickly and thoroughly mixed throughout the whole mass the time necessary for the reactions should be proportionately reduced, thereby decreasing the losses of heat, increasing the tonnage of a furnace per day (or reducing the size of a furnace for a required daily tonnage) and delivering a more uniform product.

The same applies to the alloying materials which are sometimes introduced into the molten metal; some of these tend to float and others tend to sink, hence circulation is very important.

When the refining is dependent upon the chemical action of the slag, as in steel refining, it is evident that such an action can take place only at the surface of the contact of the metal and the slag. Hence, the time necessary for such refining would be less the greater this surface. To have such large surfaces in a bath in which the renewing of this surface is accomplished chiefly or only by gravity would mean that the continuous losses of heat would be relatively very large. It would, therefore, be far more desirable to have the dimensions of this chemically active surface small and then increase the actual and effective surface by continually renewing it as, for instance, by circulating the metal rapidly in a more or less vertical direction up against the blanket of slag.

Rapid circulation is, therefore, of the very greatest importance in such electric furnaces for numerous different reasons.

It is claimed by steel makers that steel must be left standing in a quiescent state for quite a time while in a molten condition; this is called "killing" the steel. It is, therefore, alleged that circulation would be bad. The writer believes, however, that what is accomplished by this killing is to give the lighter impurities, such as gases, slag, oxides, etc., an opportunity to rise to the surface by gravity. If this is correct, the same ends ought to be accomplished far more rapidly and, therefore, more cheaply, by a forced systematic circulation from the bottom to the top (as distinguished from a mere agitation) during which all the impurities and gases would in turn be brought to the top into contact with the slag and when they have once gotten there it is presumable that they will stay there. Hence it is believed that, even though circulation is the direct opposite to resting, it will probably be found to accomplish the same object and do it far more quickly, thoroughly and cheaply.

Slag reactions require at least a certain temperature of the contact surface. This temperature is more easily maintained if the heat travels from the good conductor, the metal, to the poorer conductor, the slag, than the reverse, and the maintenance of this temperature is no doubt a very important factor in reducing the time of treatment, which in turn reduces the losses of heat per ton and increases the tonnage per day. Hence the metal should be hotter than the slag, which again becomes possible in the resistance furnace. This also reduces the maximum temperature required to maintain this necessary temperature at the surface of contact of metal and slag. If the slag is hotter than the metal, as in arc, open-hearth, or other furnaces heated from the top, the cooler good conducting metal will more quickly abstract heat from the layer of slag immediately in contact with it, forming a cooler layer of slag just where the heat is most needed; and as slag is a poorer heat conductor, it will take longer time for heat to pass down to reheat this cooler layer; hence, the action would be slower, or else the temperature of the slag must be made very much higher so as to hasten this downward flow of heat through a poor conductor. Moreover, most hot liquids are less dense than when cooler, and therefore heated liquids rise naturally by gravity. To cause heat to flow downward from a hotter to a cooler layer one must rely entirely on the very slow process of conduction, as gravity will not aid such transmission the way it does in the other direction. Rapid circulation, of course, will hasten such a transmission of heat.

The ideal way of overcoming the troubles, cost and loss of power due to the electrodes is, of course, to dispense with them entirely, as in the older forms of induction furnaces. But this also has its disadvantages, and it is significant that in what appears to be the most successful form of induction furnaces electrodes have been added.

If, therefore, these electrodes cannot be dispensed with, the next best thing to do is to make them as unobjectionable as possible. If made of the metal to be treated, their size and the losses in them become considerably less than if made of carbon or graphite, as the writer showed by actual tests some years ago. This becomes possible with resistance furnaces, but not with those using the arc. Moreover, when thus made of metal, if they are immersed under the liquid metal they cannot be consumed, or break off; they then require no adjusting and do not contaminate the molten charge. Hence in such furnaces the cost of electrodes consumed, which in arc furnaces is quite large per ton, would be entirely eliminated.

The possibility of adding fuel heat is often very desirable, especially when cold materials are charged, as the lower temperature heat is generally more economically obtained from fuel. Such heat would also reduce the loss of heat from the top surface of the electrically heated metal, would assist in melting the slag forming materials, and would lessen the amount of heat which would otherwise have to flow from the metal into the slag. The ideal form of furnace would, therefore, seem to be one in which gaseous fuel heat is supplied over the top and electric heat in the charge itself.

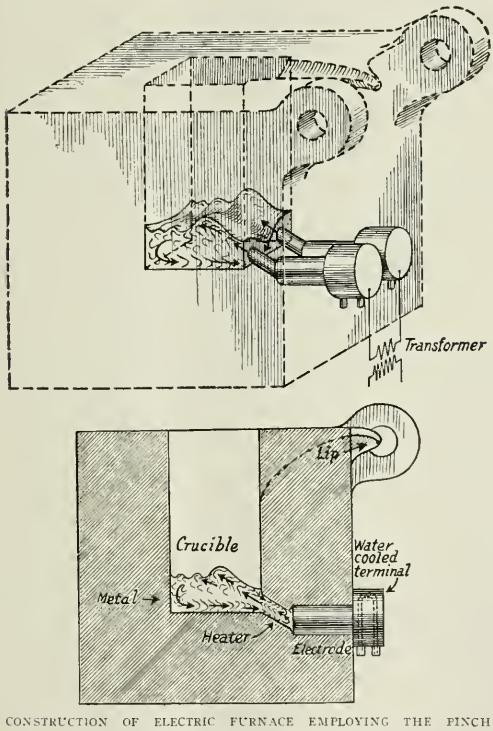
The control of resistance furnaces may generally be accomplished by the simple process of regulating the voltage of the transformer, and this can be done, within reasonable limits,

without appreciable losses of energy; hence it also approaches the ideal.

The reliability of operation, and the cheapness and simplicity of construction and operation, depend, of course, on the particular design, and, of course, the designer will approach the ideals in these features as closely as he can. In general, however, it may be said that resistance furnaces are simpler in their operation than arc furnaces, and that cheapness of first cost and of operation are good properties only when not purchased at too high a sacrifice of other desirable features.

When electric furnaces are designed so as to approach these ideal conditions more closely it is not unreasonable to predict that relatively large furnaces may become available even when the electric current must be purchased from the city mains, when all the accompanying advantages are duly considered.

With the above mentioned desirable features in mind the writer has endeavored to devise a form of furnace which



CONSTRUCTION OF ELECTRIC FURNACE EMPLOYING THE PINCH PHENOMENON FOR AGITATION OF THE CHARGE.

would embody as many of them as possible. To do this some radical departures had to be made from existing forms and types. The underlying principles of a type of furnace whose properties seem to approach some of these ideal features, especially that of rapid circulation, were described by the writer in a recent paper.[1]

Furnaces of this type are now being developed and will soon be undergoing commercial tests. The general form is shown in outline in the accompanying sketches, which represent, in its elements only, a small crucible tilting furnace.

The underlying principle, in which it differs from the usual types, is a new electromagnetic phenomenon which might be called a valveless pump. It follows as a consequence from the now well-known "pinch phenomenon" described by the writer some years ago, that when an electric current of suitable amount is passed lengthwise through a column of a liquid conductor confined in a submerged hole or tube the liquid will be ejected from the middle of this tube quite forcibly by this peculiar electromagnetic action, and at the same time will be sucked in around its circumference.

If, therefore, these holes or tubes are so proportioned that the necessary heat for the furnace will be generated in the metal in them due to its resistance, and that the ejecting force will be great enough to eject it as fast as it is heated, it will be seen that the metal in the crucible above the tubes will not only be duly and rapidly heated, but will at the same time be projected upward against the blanket of slag and be thoroughly and very actively circulated as it passes repeatedly into and out of the heating tube or resistor.

The sketches explain themselves. The heaters or resistors are the short, small tubes in which this electromagnetic force acts and in which all the heat is generated. The arrows show the circulation approximately. The electrodes, preferably made of the same metal, are the larger embedded cylinders, and are provided at their outer ends with water-cooled terminals, to which the leads from the transformer are attached. The direction of this squirting action is independent of the direction of the current, hence either direct or alternating currents may be used. The actions in this furnace have been described in the paper above referred to and need, therefore, not be repeated here.

It may, however, be of interest to add here that the ejecting force, even in a small furnace to which this principle is less well adapted than to larger ones, is great enough to cause considerable agitation, about as shown in the sketches. The suction in one case was even great enough to suck in air with a whistling noise, through a leaky tap hole at the bottom of one of the resistors against a hydraulic head of about 6 in. to 8 in. of molten lead. In another case, when the level of the liquid in the crucible had gotten too low, being nearly down to the top edge of the tubes, pieces of floating charcoal were sucked into the tubes accompanied by a snorting sound due to air being sucked in with it. When the depth of metal is sufficiently great, as it should be when operating normally, the action is entirely noiseless.

Each particle of metal is in the tube apparently for only about a second or two, judging from the velocity of the circulation. By using two tubes for each electrode the metal may be caused to pass into one and out of the other, instead of flowing in both directions in one tube.

The transformer may be directly attached to the furnace, so that the furnace as a whole may be arranged to tilt. The one shown tilts around the nose as a center, so as to enable the metal to be poured directly into the molds, thereby avoiding the use of pots.

[1] A New Type of Electric Furnace. Read at the meeting of the Amer. Electrochem. Soc., in April; abstracted briefly in this Journal for May, p. 277.

A Continuous Cyanide Treatment Method for Gold and Silver Ores.

BY JOHN E. ROTHWELL.

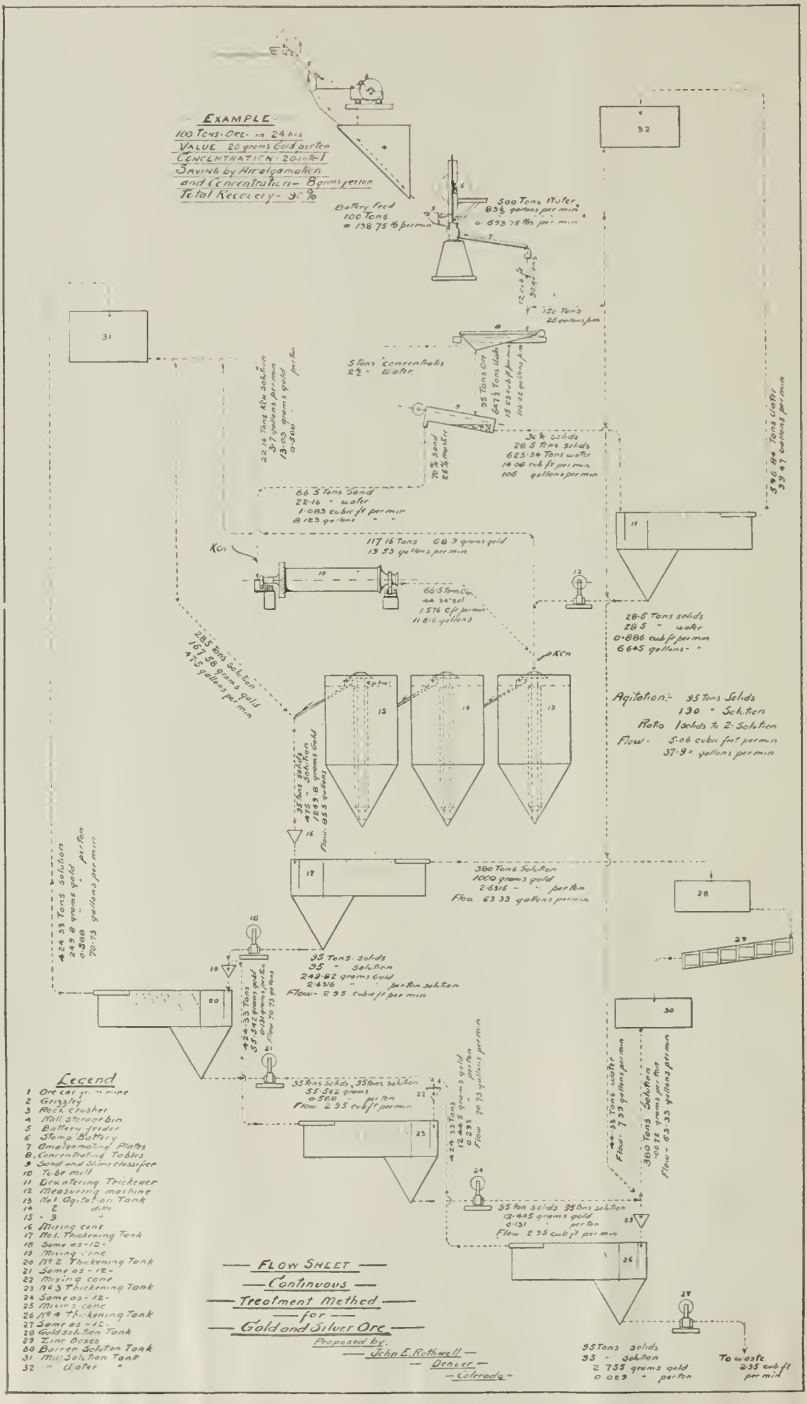
The method of continuous treatment outlined in the following is suggested as a means of operation along lines that will tend to eliminate as much as possible the personal equation which always plays such an important part in all milling operations, and bring the whole treatment to a nearly automatic operation.

Reference is made to the flow sheet that accompanies this paper, on which is shown the general machinery and equipment of a milling plant and the flow of the ore through same. Certain arbitrary conditions are assumed in the illustration, but these conditions are well within the limits of ordinary practice.

The example of milling treatment is taken at 100 tons per day, so the conditions prevailing at different points during the flow can be carefully noted in cubic feet and gallons per minute flow during the daily operation. Thus we get an idea of the volume passing certain points in a given time and the requirements in equipment necessary to handle these amounts. The mechanical automatic regulation and control are matters on which I will not go into the detail, but will say that equipment designed to perform the necessary functions, and register, by record, the actual conditions prevailing at the different points throughout the operation, is entirely a matter for the mechanical and electrical engineer, and presents no serious obstacle in either field.

The mechanical operations outlined in the flow sheet for the preliminary mechanical treatment of the ore down to the continuous agitation tanks, shown as 13-14-15 in flow sheet, are based on common practice. Some of the features shown may not be required on certain classes of ore, the leaving out of any one of which will not affect the general flow.

At the agitation tanks, it will be noted that the continuous flow is carried on through a sampling vane located *above* the pulp level in the tank The object of this is that these tanks also may act as an equalizer for the flow of the ore through the plant. Thus, for illustration, in event of a stoppage of the milling end of the plant for a short period of time, the tank 13 can be drawn on for about one-third of its capacity and maintain a constant flow through the others; similarly the other storage tanks can be drawn



on for a part of their contents and deliver a constant flow to the cycle of operations following.

The flow of the solids after leaving the agitation tanks and the recovery of the dissolved value is based on the idea of a countercurrent flow of diluent against the downward flow of the solids and accompanying dissolved values.

The critical point in this whole operation requires the absolute control of the ratio of the two currents in order that dilutions may be positive. The slight variations of value in gold and silver to be recovered due to the change in the grade of the ore being treated will not require any radical change in the flow in either direction.

The value of the countercurrent lies in the constant change of solvent as well as diluent, the tendency of which is to avoid reaching the chemical equilibrium between the solvent and the metal to be dissolved and obtain for the solvent its highest efficiency.

The recovery of the dissolving chemical (the cyanide) when, due to economic features, a solution is used that requires this recovery, is not shown in the flow sheet; but if we follow out the general features of the recovery of the gold values, in a subsidiary treatment, this recovery can be made to any degree required for economical practice. Absolute accuracy is not claimed for the figures shown in the flow sheet, but they are well within the lines of general practice and are given to show the possibilities of the method of treatment carried out in a continuous manner.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Iron and Steel Institute

The spring meeting of the Iron and Steel Institute was held at the Institution of Civil Engineers, London, on May 11 and 12.

The discussion on the paper by MM. F. A. Daubiné and E. V. Roy entitled "A Process for the Desiccation of Air by Calcium Chloride" (which is printed in full elsewhere in this issue) was opened by Mr. J. H. Harrison, who said that the value of the paper lay in its being a practical contribution dealing with iron and steel manufacture. Some six or seven years ago while the principle propounded by Mr. Gayley's paper on desiccation of furnace-blast air met with some degree of acceptance, his conclusions were received with incredulity if not with ridicule, but now it was recognized that dessication effected a saving of from 10 per cent to 20 per cent in coke. The author's method seemed to be a distinct improvement on the processes previously employed.

Mr. H. Hawdon remarked that in England very few works were equipped with apparatus for washing the waste gases used for drying the calcium chloride, and the heat required would involve considerable cost; and as to cooling apparatus, he thought that a large proportion of the chloride would not come into contact with the water at all.

Mr. J. E. Stead said the desiccation of blast by calcium chloride was first tried fully twenty-five years ago, but there was not then any provision for the obviously necessary cooling of the chloride after it had been dried. The hot blast was sent through the chloride to dry it, and was immediately followed by the passage of the cold blast. That process was, of course, ineffectual, but the author seemed to have met the difficulty.

Mr. F. W. Harbord doubted if the process would in practice do what the authors claimed in keeping surface temperature low. It was probable that as the surface of the chloride liquefied the resulting mass would greatly impede the passage of air through the grids, but if the process were really adaptable to the conditions of every-day work it would be better than the freezing process.

Mr. Lawrence Sterne expressed a desire for definite figures of working costs. The results obtained by refrigeration were known, and simplification of the apparatus with reduction of prime cost and upkeep were progressing favorably.

The balance of the report of the Iron and Steel Institute meeting will be given in our next issue.

"The Times" on Steel Prices

A special article on American Steel Prices and the effect of cutting on British Trade in *The Times* of May 31st is apparently written by an English optimist. The writer discusses the effect of the breakdown of the price maintenance understanding in the United States Iron & Steel trade, and commences by pointing out that "In the United States there are plenty of blast furnaces and rolling mills to undertake about nine-tenths of the world's export business in iron and steel, in addition to meeting home demands. If our United States rivals are able and willing to cut their export prices low enough—just a little below British and German figures—they can easily secure the supreme position in neutral markets, almost wipe out all competitors, and, besides practically destroying nearly all their rivals' export trade, they can make very serious inroads upon the purely domestic business of British steel manufacturers.

"There is a world's export trade in iron and steel of something like 13,000,000 tons a year. Of this total Britain and Germany between them are dividing nearly 10,000,000 tons; America is taking only one and a half million tons. But America has plant enough to do an export trade larger than that of Britain and Germany put together. In minerals, mills and men the United States producers are equal to the task. The only difficulty is that of price. Can our American rivals sell cheaply enough to beat us in the race?"

After offering explanations of the non-fulfilment of the repeated intimations during the last ten or fifteen years of a great export campaign by American steel manufacturers, he proceeds:

"Given the collapse of the American understanding and the prospect of an all-round reduction to the lowest profitable level, involving something like free international competition, how would matters shape themselves? On this point it is impossible to appeal with much confidence to past history. It is known that in the past American competition has been more imaginary than real. Our United States rivals have never approached us in the exportation race. . . . Is America at last about to make her long-threatened bid for the lion's share of the world's export trade in steel?

"This brings us to the root question. Can America produce at a sufficiently low cost to conduct a big export business for any great length of time? Are the costs of production really lower in the United States than they are in Britain or Germany? Twelve years ago it was authoritatively estimated that the cost of producing steel-making pig iron was less in America than in this country by very nearly 20s. per ton, and that in the case of steel rails the difference was 29s. . . . But it is a remarkable fact that since that time productive costs have gone up more substantially in the American iron and steel group of trades than in any other important branch of industry anywhere. The latest reliable figures are those supplied to the Ways and Means Committee two-and-a-half years ago, and they show that between 1899 and 1908 the net cost of producing pig iron increased by 24s. per ton, and of steel rails, 27s. per ton. This in spite of the mechanical and metallurgical improvements effected. Since the late nineties mining royalties have risen; mining costs have increased, while the yield of the ore has declined; cost of coal and coke has gone up; transit charges on water and rail have increased, so that it is not now possible to produce iron and steel with anything like the economy of a dozen years ago. There has been no corresponding rise in our costs of production for the rise in prices of some raw materials has been just about offset by improvements in processes."

One has got so used in this country to depreciation of our industrial potentialities that this article is quite a refreshing change to the much advised British manufacturer. But whether it will be safe for the lion to emulate the ostrich and hide his head in the sand in another matter. Wolf has been cried so often that the sheer novelty of a whisper of safety may have the same effect as the first of the alarms which disturbed our equanimity and make us more wary than we have recently been.

Market Prices.

May, 1911.

Copper was lower during the first part of the month, reaching £53.15.0 by the 10th, and remaining there till the 15th. It then rose steadily to the 23d, when £55 was reached. Closes at £55.

Tin was nervous at the beginning of the month, going between £195.10.0 to £192.10.0 before rising, which commenced on the 10th whence it soared pretty steadily up to its present price of £203.

Lead is slightly stronger at £13.7.6, rising on the 8th.

Haematite is down, having fallen away from 8th to 12th, and now standing at 61/9.

Scotch Pig was rather flat during the opening fortnight, touching 51/10. It commenced to rise smartly on the 18th and finishes at 52/3.

Cleveland has been much in sympathy with Scotch pig-iron and dropped to 45/11½ by the 10th. It has since recovered to 46/2.

Rubber has been low and weak, dropping from 5/- on the 10th to 4/4½ on the 17th. It has since become steadier and closes at 4/2.

Alum, lump, loose, per ton	£5.12.6
Antimony, black sulphide powder, per ton	22. 0.0
Borax, British refined crystal, per ton	16. 0.0
Copper ore, 10 to 25 per cent., per unit	9/1½ to 9.7½
Copper sulphate, per ton	22. 0.0
Carbolic acid, liquid 97.99 per cent, per gal	1.4
Creosote, ordinary good liquid, per gal	0. 0.2
Camphor, 1 oz. tablets	1.9
Caustic soda, ash 48 per cent, ordinary, per ton	5.10.0
Litharge, flake, per ton	15.10.0
Mica, small slab, original cases, per pound	6d. to 2.0
Naphtha solvent, 90 per cent. at 160° C., per gal	0. 0.9
Petroleum, Russian spot	0. 0.5⅛
Sal ammoniac, lump, firsts, delivered U. K., per ton	42. 0.0
Sulphate of ammonia, f.o.b. Liverpool, per ton	13. 5.0
Sulphur, recovered, per ton	5. 0.0
Shellac, standard T. N., orange spots, per cwt.	4. 4.0
Tin ore, 70 per cent., per ton	£125 to 127. 0.0
Hydrochloric acid, per cwt.	5.0
Zinc, sheets, Vieille Montagne, f.o.b. Antwerp	29.10.0
Quicksilver, per bottle	8. 5.0
Platinum, per oz.	8. 6.6

On the month the following differences are noted:

Higher.	Amount.
Copper	£1.0.0
Tin	9. 5.0
Lead	3.9
Copper ore, per unit	0.1½
Copper sulphate	2.5.0
Tin ore	2.0.0
Carbolic acid	0.0.1
Lower.	
Haematite	0.1.6
India rubber, Para	0.1.1
Scotch pig	0.0.6
Cleveland warrants	0.0.4

London, England.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Electrically Refining Molten Pig Iron.—In our March issue, page 127, we published an article by Mr. J. B. Nau on the preliminary refining of pig iron, more especially of phosphoretic pig iron too high in silicon to be treated directly and economically in the open-hearth furnace. The essential principle is to submit pig iron, running from the blast furnace, to the refining action of solid pieces of iron ore. Silicon, phosphorus and carbon will thereby be eliminated, but in order to maintain the required temperature it will in general be necessary to supply heat to the charge during the process, as was shown by the heat-balance sheet in the article referred to. For this reason Mr. Nau proposes to carry out the process in an electric furnace, as it is the easiest way of controlling the temperature absolutely. Three recent patents of Mr. Nau refer to this process. From the last two we reproduce herewith the

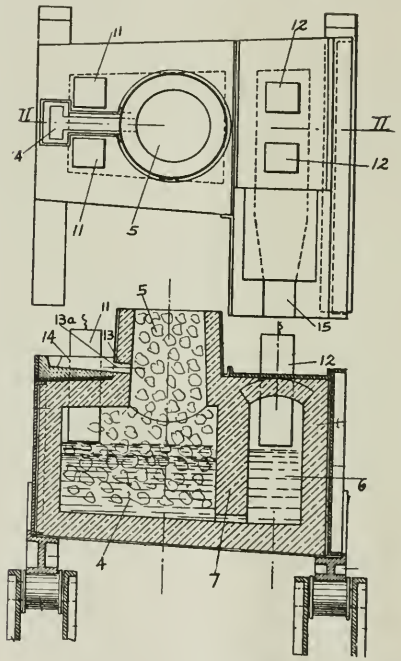


FIG. 1.—ELECTRIC FURNACE FOR REFINING OF PIG IRON.

tilting furnace in which the refining is carried out. As shown in plan and vertical cross-section in Fig. 1, it consists of two parts, a crucible 4 surmounted by an ore shaft 5, and a forehearth 6 separated from the crucible 6 by partition 7 and communicating with it through holes 8 near the bottom. The forehearth 6 is provided with an outlet spout (not shown in the illustrations) for pouring the refined metal when the furnace is tilted. 11 and 12 are the electrodes. After having heated the empty furnace by gas, preheated rich iron ore in solid pieces or in the shape of concentrate ore briquets is introduced into the crucible 4 through the ore shaft 5. The liquid pig iron from the blast furnace, which is to be refined, is poured into spout 14, enters the ore shaft through opening 13, percolates through the ore and passes over to the forehearth 6. Otherwise the illustration is self-explanatory. (984,590, Feb. 21; 990,765, April 25; 991,633, May 9.)

Electric Iron Ore Reduction.—An electric furnace specially adapted to the electric smelting of iron ores by a con-

tinuous process is described in a patent of Mr. EDWARD R. TAYLOR under Electric Furnaces.

Nickel Steel with the Aid of Electrolysis.—See the abstract of a patent of Betts for electrolytically refining alloys under Electrolytic Processes.

Electric Furnaces.

Electrode Holder.—A new construction of electrode holder of electric furnaces, due to Dr. PAUL L. T. HÉROULT, of La Praz, France, is shown in Fig. 2. Diagram *I* shows the carbon electrode *A* attached to the carbon holder *B*, which is constructed as a clamp encircling the carbon. This ring may be

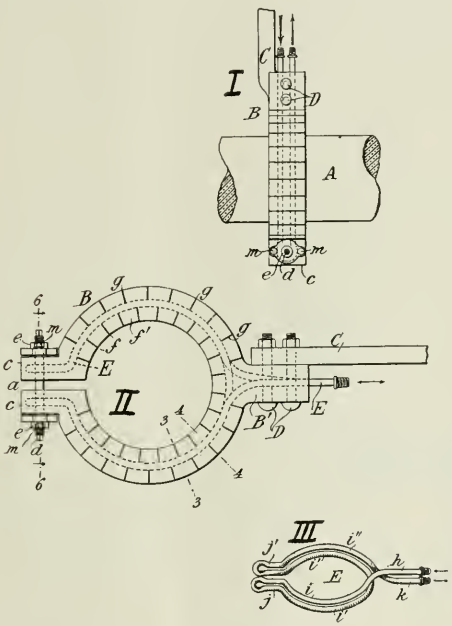


FIG. 2.—ELECTRODE HOLDER.

divided on one side *a*, as in diagram II, or at oposite sides. At the division *a* the clamp is formed with flanges *cc* through which passes a tightening screw *d*. To render the holder sufficiently flexible to readily adapt itself to slight irregularities in the surface of the carbon, it is slitted radially on its inner side at *ff* and on its outer side at *gg*, the outer and inner slits being staggered or alternated. The electric current connection is through the bar *C*, which is clamped to the arm *B'* by bolts *D*. For water-cooling a duct *E* is provided, consisting of an iron tube of the shape shown in diagram III. The main portion of the holder *B* is of copper or brass and is cast around the iron tube *E*, so that the latter is entirely embedded. (989,148, April 11, 1911.)

Electric Resistor Furnace.—The resistor furnace of Thomson and FitzGerald, the characteristic feature of which is heating of the charge by radiation from resistors on top of the furnace, and its application to zinc smelting have been repeatedly mentioned in this journal. To reduce the heat loss due to radiation Mr. JOHN THOMSON has now modified the construction by embedding the carbon resistor (which is otherwise of the same construction as before) in the charge itself. In order to prevent the electric current from leaking into the charge and in order to protect the resistor at the same time it is surrounded by a casing or shield. This casing is not in direct contact with the resistor, but there is an air space between them, which may be filled with an inert gas instead of air. For the construction of the casing fireclay, "crucible mixtures" (combinations of graphite and clay) and recrystallized silicon carbide may be used. (994,217, June 6, 1911.)

Titanium as Reducing Agent.—Mr. JOHN B. HUFFARD, of the Electro Metallurgical Company, of Glen Ferris, W. Va., produces alloys of titanium with vanadium or molybdenum, etc., by reacting on an oxide of the alloying metal (vanadium, molybdenum, etc.) with a titanium reducing agent, for instance, metallic titanium or titanium nitride, carbide or monoxide, etc. The titanium reducing agent is produced by the reduction of rutile; it may contain metallic titanium and relatively small amounts of nitrogen, carbon and oxygen. (992,423, May 16, 1911.)

This same titanium reducing agent is employed by the same inventor for the production of the metals tungsten, vanadium, molybdenum, etc., free from carbon. With tungstic acid the reactions are as follows:

$$2\,WO_3 + 3\,Ti = 2\,W + 3\,TiO_2.$$
$$2\,WO_3 + 3\,TiN = 2\,W + 3\,TiO_2 + 3\,N.$$
$$WO_3 + 3\,TiO = W + 3\,TiO_2.$$
$$WO_3 + TiC = W + TiO_2 + CO.$$

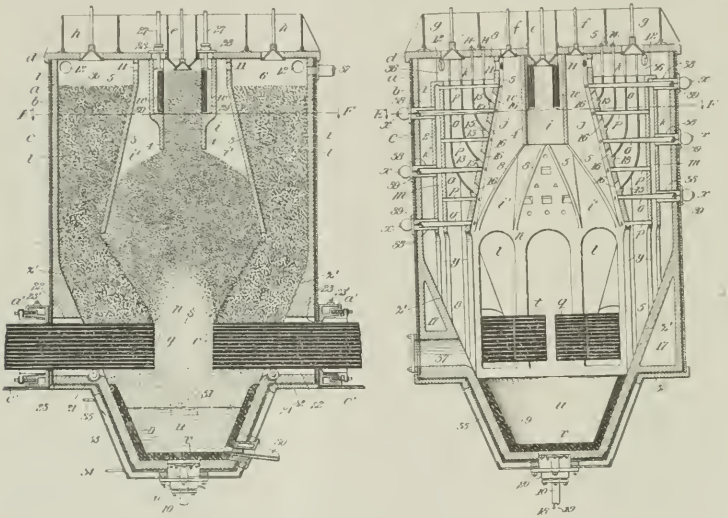
For the production of low-carbon metals the reducing agent should contain a minimum of titanium carbide since the oxidation of the combined carbon is not complete. In order to fully oxidize and utilize all of the titanium it is necessary to react on an excess of the reducible oxide. The resulting slags then contain both the oxidized titanium and the unreduced oxide of the charge. Titanium oxide not being acted upon by acids, the unreduced oxide may be dissolved out of the slags and recovered by evaporation. Tungstic oxide is readily dissolved out by hydrochloric acid. (992,422, May 16, 1911.)

Silicon Carbide, Silundum.—Mr. FRANK J. TONE, of the Carborundum Company, of Niagara Falls, N. Y., has formerly described a method in which shaped pieces of carbon are heated in proximity to a silicon-containing substance to such a temperature that the silicon combines with the carbon *in situ*, so that the shaped pieces of carbon are transformed into dense silicon carbide. (In Europe this is called silundum.) Mr. Tone has now found that if he makes his original shaped pieces not of carbon, but of a mixture of silicon carbide and carbon, he can vary the porosity and resistance of the product within wide limits. A mixture of ground silicon carbide and carbon, with tar, glue or silicate of soda as binding material, is shaped into the desired form and baked to remove moisture and other volatile materials. The article is then placed in an electric furnace and is embedded in a mixture of carbon and silica, or of silicon carbide and silica. The furnace is brought up to the temperature of the formation of carborundum, at which temperature vapors of silica and silicon are formed, which penetrate into the molded article to a considerable distance. These vapors reacting with the carbon change it into silicon carbide. The binding material is volatilized and the resultant product consists of silicon carbide bonded with silicon carbide in a dense, compact form. (992,698, May 16, 1911.)

Titanium Alloys.—For the production of ferrotitanium, Mr. A. J. ROSSI, of Niagara Falls, N. Y., proceeds as follows: A bath of molten iron is first provided in the electric furnace and this bath is covered with a layer of lime, CaO. Upon this is placed a layer of a mixture of titanic acid and carbon. The titanic acid is reduced to titanium, $TiO^2 + 2\,C = Ti + 2\,CO$; the titanium sinks downward and alloys with the iron. At the same time calcium carbide is formed, $CaO + 3\,C = CaC_2 + CO$. The calcium carbide, having reducing properties, promotes the reduction of the titanic acid and at the same time protects the molten alloy both from alterations by oxidation and volatilization. Toward the close of the process an additional blanket of carbon is added on the layer of calcium carbide in order to insure a reducing atmospheric atmosphere. Instead of titanic acid, as described above, titanate of lime, CaO,TiO_2, may be used. (986,505, March 14, 1911.)

Continuous Electric Shaft Furnace.—After many years of

successful experience with his electric furnace for carbon bisulphide manufacture Mr. Edward R. Taylor, of Penn Yan, N. Y., has undertaken in recent years to design an electric-shaft furnace which, while retaining the very satisfactory salient features of his carbon bisulphide furnace, would be especially applicable to reduction of iron ores in a continuous process. His latest design is shown in Figs. 3 and 4 in vertical section in two different planes, the second diagram showing the furnace empty. It is of stack form, the internal structure *a* being of brick or cement incased by a metallic shell *b* with an intermediate layer *c* of asbestos. *e*, *f*, *g*, *h* are feeding hoppers in the top, communicating with flue-shaped passages *i*, *j*, *k*-*k'* and *l* extending downward. The coke is introduced through the central hopper *e* into the passage *i*. The coarse ore is introduced through four hoppers *f* and passage *j*, symmetrically arranged around the center. The fine ore is introduced through the four external



FIGS. 3 AND 4.—ELECTRIC FURNACE FOR IRON ORE REDUCTION.

hoppers *g*, each of which communicates with two passages, a main passage *k* and a peripheral passage *k'*, separted from each other by hollow walls forming a gas space *m* within each wall, open at the bottom and also in direct communication with the working chamber *n* by way of outwardly flaring gas tubes *o*. The peripheral fine-ore passages *k'* communicate with the working chamber at their lower ends only. The main fine-ore passages *k* are provided with baffle plates *p* to retard the flow of the fine ore and permit gas spaces through the body of fine ore. Four other hoppers *h*, situated midway between the four external hoppers *g* and the peripheral passages *l*, are appropriated to the feed of resistance regulating material adapted to flow downward by gravity and to protect the main electrodes and regulate the conductivity of the charge at the heat zone. For this purpose broken electrode carbons may be used, crushed to suitable size.

There are four horizontal main electrodes *q*, *r*, *s*, *t*, arranged in two pairs at right angles to each other. They are located immediately above the hearth *u*. Secondary electrodes *v* and *w* are located at the bottom and top of the furnace respectively. An annular gas chamber 36 at the top of the furnace communicates freely with the coarse ore and coke passages and with the intermediate passages for fine ore and regulating material, and discharges through an outlet pipe 37. The tube *o* for circulating the CO and CO_2 gases within the furnace extend from the coarse-ore passages *j* outward into curbs 38, which project inward to the gas spaces *m* from outside the furnace, and electrically driven rotary blowers 39 are located within the outer ends of these tubes. To insure the inward feed of the ore stoking screws are provided, which are not shown in the illustrations here reproduced. Passages are also provided through which air may be blown into the center of the working chamber.

The height to which the central coke column is kept incandescent is determined by passing more or less current through it by way of the secondary electrodes *v* and *w*. This in turn determines the proportion of carbon in the molten metal as it reaches the hearth and facilitates producing either pig iron or steel at will. The gas-circulating blowers 39 insure the conversion of the carbon monoxide gas formed in the vicinity of the heat zone into carbon dioxide, the conversion of the latter into carbon monoxide, and so on. Considerable details are given in the specifications on the construction of the electrodes. (985,226, Feb. 28, 1911.)

Electrolytic Processes.

Electrolytically Refining Alloys.—Mr. Anson G. Betts, of Troy, N. Y., subjects an alloy which does not contain the two principal metals in a desired proportion to a separation into at least two other alloys, in one of which at least the ingredients are together in a more useful proportion. For this purpose the alloy is used as anode in an electrolytic cell, the two principal metals being dissolved, while many of the impurities are left as an anode slime. By varying the conditions of deposition he then alternately deposits at least two different alloys. Two examples are described, the refining of arsenic-antimony and of nickel-iron alloys.

The arsenic-antimony alloy is dissolved as anode in a solution containing antimony and arsenic trifluorides, an alkaline fluoride and hydrofluoric acid. When the solution is rapidly agitated arsenic deposits out faster than antimony and the cathode metal may thus contain more arsenic in proportion than is dissolved from the anode, until the solution becomes impoverished in arsenic. If now the solution is kept at rest and electrolysis continued an alloy is produced for a time, containing a less proportion of arsenic than is dissolved from the anode. When arsenic has sufficiently accumulated in the electrolyte, the first operation with rapid agitation is repeated. In practice there should be at least two refining tanks in use, through which the solution flows continuously from one to the other, and in one of which the solution is well agitated, while in the other it is nearly at rest. The alloys deposited in the two tanks are different from each other and from the anode and are generally also free of many impurities of the anode, as silver, lead, copper, etc. About one-half of the arsenic in the anode remains in the anode slime.

Iron-nickel alloys are treated as anodes in a solution of the sulphates, fluosilicates or other usable salts of iron and nickel. By varying the temperature the composition of the cathodic deposit is varied. Most of the impurities are eliminated as anode slime or remaining in solution. With an anode containing 6 per cent nickel, 6 per cent copper, 1 per cent manganese, 3 per cent carbon, 1 per cent silicon, by electrolyzing at a tem-

perature of 40° C. to 50° C. he deposits for a while cathode metal with 3 per cent nickel, while the percentage of nickel in solution increases. Then by electrolyzing at a temperature of 90° C. there deposits on the cathode an entirely different product, which may contain 13 per cent nickel and nearly 87 per cent iron, while the percentage of nickel in the solution diminishes. The solution is then cooled and the first operation repeated. In practice the two steps would be carried out in two different tanks.

"Nickel steels containing 3 per cent to 3.5 per cent of nickel are at present the most popular. One of the above-mentioned products of about this composition is directly convertible to an especially fine nickel steel nearly free from harmful impurities containing this amount of nickel, by melting with the proper additions, while the other product may be melted with other relatively impure iron to produce a lower grade of nickel steel. If the entire product made from anode metal of the composition given was of one kind containing about 7 per cent of

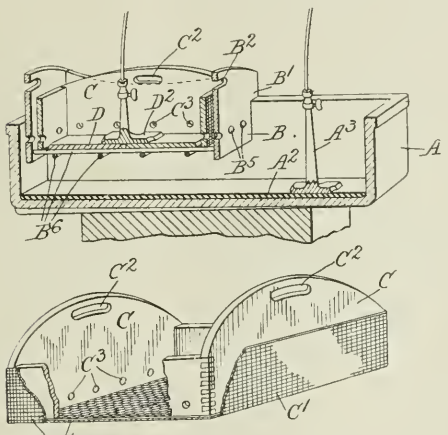


FIG. 5.—ELECTROLYTIC CELL FOR PARTING GOLD AND SILVER.

nickel it would require to be melted with outside non-electrolytic iron of lower grade to produce the very useful 3 per cent to 3.5 per cent nickel steel, and obviously would not be as good for making high-nickel steels as the 13 per cent alloy, for example." (987,947, March 28, 1911.)

Electrolysis of Brine Solutions.—In the electrolysis of sodium chloride solutions in diaphragm cells, like the Hargreaves cell, the solution coming from the cathode compartment contains undecomposed sodium chloride. To recover this and use it again is the object of a patent of Mr. GEORGE A. GABRIEL, of Brooklyn, N. Y. (assigned to the Bleach & Caustic Process Company). A second object is to renew and purify the anolyte and to remove any gases from it. These objects are obtained in a cyclic process in which evaporators, centrifugal driers, filters, etc., are employed. There are thirty-four claims. Claim 8 reads as follows: "In an electrolytic process, in combination with the step of electrolysis, the process of treating the discharged electrolyte remaining in the catholyte, which comprises removing the undecomposed electrolyte by concentrating the main body of the products of decomposition, converting the remaining products of decomposition contained therein into substances which purify the raw electrolyte, and then submitting the purified electrolyte to said step of electrolysis." Claim 24 reads as follows: "In an electrolytic process, in combination with a continuous main circulation of the electrolyte discharged from a cathode chamber, for purifying and supplying the electrolyte, a local or auxiliary circulation of the anolyte, including the steps of electrolysis and filtering." (959,730, May 31, 1910.)

Electrolytic Parting Cell.—What may be considered as an interesting further development of the well-known Balbach cell for parting gold and silver is patented by Mr. WILLIAM THUM, of Hammond, Ind., and shown in section in Fig. 5. *A* is a tank with an inclined bottom on which rests the cathode plate A^2 (for instance, of carbon), connection to the outside circuit being made through the cathode terminal A^3. The anode cell *B* is suported in the tank *A* by means of the projecting ledges B^4. Perforations B^5 are provided in all the walls. The bottom of the cell consists of a grid made up of the bars B^6. The anode basket *C*, which is located within the anode cell and rests upon the grid B^6, is provided with the canvas bottom C^1, as shown in the lower diagram, the handle holes C^2 and the perforations C^3 in its sides and ends. The anode plate *D* is situated within the basket *C* and rests upon the spacing bars, which rest in turn upon the canvas bottom of the basket. D^2 is the anode terminal. A modification of this construction is also described, in which the anode cell and basket are made in one.

If silver bullion containing gold is to be treated an electrolyte of silver nitrate with free nitric acid is used. The electrolyte is contained in the tank *A* at such a level as to be slightly above the perforations B^5 and C^3 in the anode cell and anode basket respectively. The silver dissolves from *D* as nitrate and the heavy silver nitrate solution sinks through the canvas and runs down over the full length of the surface of the inclined cathode plate A^2 on which the silver is deposited in shape of loose crystals. The electrolyte, the specific gravity of which has thus been reduced, rises now upward again and flows back across the upper part of the tank toward and into the anode cell through the perforations B^5 and C^3. The gold in the anode plate *D* is not dissolved and accumulates in the shape of mud or slime resting loosely on the perforated bottom of the basket.

The cell may be similarly used for refining bismuth. In this case an electrolyte of bismuth chloride and free hydrochloric acid is used. The bismuth is obtained in form of more or less coherent easily removable crystals, having the shape of trees, mushrooms and the like. (969,921, Sept. 13, 1910.)

SYNOPSIS.

GOLD AND SILVER.

Cyaniding Tailings.—Details of the treatment of a 500,000-ton tailing dump at the Oroya Brownhill, in Western Australia, are given in the March *Journal* of the Chamber of Mines of Western Australia, by Messrs. STEVENS and DEGENHARDT.. A vacuum filter plant was erected at the end of 1908 for the retreatment of a great mass of filter press tailing. The process consists of belt conveying to the plant, mixing with spent solution, agitation with cyanide, vacuum filtration and precipitation on zinc shavings.

The tailing treated consists of a mixture of 92 per cent. of an all-slimed product from bromo-cyanide treatment, and 8 per cent. of roasted concentrate. It is fine, only 2.5 per cent remaining on a 100-mesh screen. The average value is about $2 per ton, and by agitation with weak cyanide solution an extraction of $1 is obtained. Of the extracted value, one-half is present as dissolved gold in the 6 per cent. moisture contained in the dump; the $1 remaining in the residue when discharged is not soluble in stronger cyanide solution, and can only be extracted by regrinding and concentration, which would not be profitable. The tonnage treated is 730 tons in 24 hours.

A movable belt conveyer is placed along the side of the old dump, and the material is run on to it by men working with picks and shovels. As the dump is removed the belt conveyer is placed closer to it, each move giving about 10,000 tons. Taken over a period of two years the cost of the belt has been about 0.8 cent per ton.

The dump is easily moved, and three men working with picks on the dump, together with two men shoveling from the bot-

tom, can run about 250 tons per shift on to the belt. The material runs easily at 41.5 degrees, but in wet weather the rills must be a little steeper.

At the mill the material is mixed in a vortex mixer with spent solution from the extractor boxes. These mixers have cast iron impellers on 3-in. shafts and are driven at 350 r.p.m. by quarter-turn belts. From the mixer the pulp flows to the agitation vats, which are 17 ft. 6 in. diameter and 9 ft. deep. These vats have bottoms of 18-gauge plain galvanized iron and sides of 20-gauge corrugated galvanized iron, and last about three years. The bottom is usually the first part to wear out. Each vat has four angle-iron arms driven at 12½ r.p.m.

The spent solution tests 0.04 per cent. KCN, and lime to the extent of 1.5 lb. per ton of tailing is added in the mixers, to bring the protective alkalinity up to about 0.002 per cent CaO. The pulp is agitated for eight hours, and the solution tests 0.02 per cent. KCN when it is drawn off to the vacuum filters. Average cyanide consumption is 0.47 lb. of sodium cyanide per ton. Specific gravity of the pulp is maintained at 1.61.

The filter equipment consists of two vats each 10 ft. wide, 21 ft. long and 7 ft. deep, the bottoms being divided into two 60-degree discharge hoppers. The filter vats are filled by an 8-in. centrifugal pump, the wearing shaft of which lasts about 4½ months and the runner 9 months. It is operated 38 per cent. of full time by a 45-hp. motor. Each filter vat contains 34 frames 5 ft. 3 in. deep by 9 ft. wide and set 6½ in. between centers of frames. The frames will carry an average of 22 tons of dry slime at a charge, making cakes 1⅞ in. thick. Of the pulp charged into the vat 43 per cent. is removed as solution and cake, and 57 per cent. is pumped back. The cycle of operations is as follows: Filling vat, 5 minutes; forming cake, 30; emptying excess pulp and putting in wash, 8; washing, 30; emptying excess wash, 4; discharging cake, 7; total, 84 minutes. A vacuum of 27 in. is maintained during filtration, and the cake is discharged with 24 per cent. moisture and soluble gold equal to 6 cents per ton of dry slime.

The pregnant solution is clarified in presses and precipitated on zinc shavings. The consumption of zinc is 0.13 lb. per ton. Cyanide is added in the gold solution vat as strong solution to assist precipitation, and the solution entering the boxes contains not over 0.05 per cent. KCN. About one ton solution is precipitated for one ton of tailing treated. The total cost of breaking and treatment is about $0.52 per ton (2s. 1.28d.).

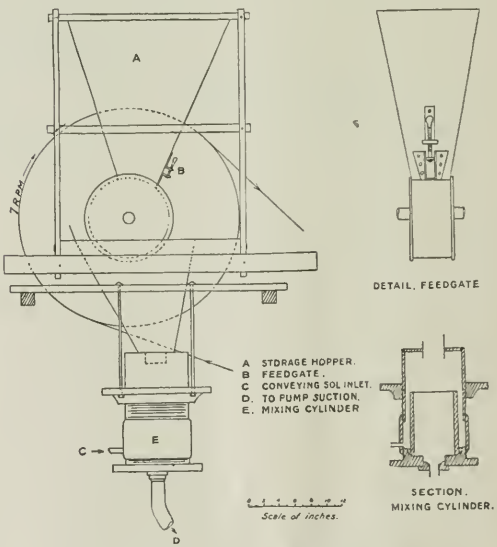
All-Sliming Treatment.—A fine example of the development of a cyanide process is given by Mr. Alex. Fyfe in a paper read before the Australasian Institute of Mining Engineers and published in the *Transactions* of that body. Beginning with stamp-amalgamation and separate treatment of slime and sand, with elimination of sulphides, which were treated separately, the process has been developed to a point where but one product is made and treated by cyanide. The change was gradual and resulted from numerous experiments, from each of which an excellent lesson was learned and applied.

The first step was an attempt to treat the sulphide concentrates at the mill instead of shipping them to smelters. The process included regrinding in tube mills, agitation with compressed air and filtering in presses. The cost of regrinding was high, as one tube 19 ft. 3 in. long by 4 ft. 8 in. in diameter could reduce only 10 tons in 24 hours, and the pebble consumption was 30 cwt. per week. Later coarse sand was added to the concentrate in an endeavor to improve grinding conditions, and this was so successful that the concentrate was fed into the mills grinding raw ore. Later amalgamation was dropped, and toward the end of 1908 cyanide solution was added to the mortars instead of clear water. This shortened the time of treatment and did away with the necessity of dewatering the slime before treatment with KCN, so that the vacuum filtration plant formerly used for this purpose was available for filtering pregnant solutions. Finally additional tubes were added and the sand ground to such a fine state that it was mixed with the slime and treated as one product. Thus from a complex system, with several products to treat, there has been evolved a simple process with but one product.

The solution added to the stamp mortars contains 0.10 KCN, and is added at the rate of 10 tons of solution per ton of ore. Lead acetate to the extent of 0.5 lb. per ton of ore also is added. The stamps weigh 1100 lb., drop 7 in. 105 times per minute, and have a duty of 7.6 tons per stamp per day. From the stamps the pulp flows to spitzkasten, the underflow from which goes to tube mills. The combined slime is settled and drawn off at a consistency of 1.5 solution to 1 of slime, and agitated in high tanks with air at a pressure of 38 lb. per sq. in. Time of agitation, 18 hours; cyanide consumption, 1.258 lb. per ton. Experiments showed that 87 per cent. of the total value is in solution before agitation begins, and that only a small additional extraction is effected in the tanks. The agitated pulp is then filtered in Moore filters and the solution precipitated in zinc boxes at the rate of 1.4 tons of solution per cubic foot of zinc in 24 hours. Expenses per ton of ore treated are distributed as follows.

Rock breaking	$0.104
Stamping	.166
Tube milling	.292
Classification (Wilfley tables)	.072
Treatment	.636
Precipitation and melting	.054
Assaying and sampling	.058
General expenses	.120
Total	$1.502

Zinc Dust Precipitation.—The formation of calcium carbonate during precipitation by zinc dust and the detrimental



ZINC DUST PRECIPITATION.

effects of this salt on precipitation are set forth by Mr. Allan J. Clark in the April, 1911, issue of the *Mining Magazine* (London). The formation of calcium carbonate may result from agitating, with compressed air, the solution before the precipitation of the precious metals, or by agitating with compressed air to form the zinc emulsion used for precipitation.

At the time the author's attention was first directed to the matter in hand, the method of preparing the zinc emulsion was that of agitating by compressed air in an inverted cone with a

continuous overflow to the suction of the pump, where it was mixed with the solution flowing to the presses. The first instance of the effect of calcium carbonate was noticed while precipitating a very weak and low grade solution of high alkalinity. This solution had been collected in shallow vats and agitated with compressed air, and was in prime condition for the formation of the objectionable salt. The precipitation offering some difficulty, the rate of flow was reduced without improvement in precipitation. The press was then opened, and a heavy deposit of calcium carbonate was found, caked on the press cloths or mixed with the precipitate. By eliminating the preliminary agitation of the solution the difficulty was remedied.

The author suggests that the formation of calcium carbonate diminishes the efficiency of the zinc by coating the zinc particles and preventing further action.

By eliminating the mixing cones and substituting the method of allowing the zinc dust to fall from an endless belt into a trough where it was washed by a stream of solution direct to the precipitation line, there was an increase in concentration of 63 per cent.; a decrease of consumption per ton of solution of 35 per cent.; and a decrease of zinc consumption per ounce of gold precipitated of 38 per cent.

Later, on account of the irregular action of the belt feeder, a new design of zinc dust feeder was installed. It is designed by W. H. Todd, foreman at the Homestake mill. It consists of a hopper with feed gate, and a mixing cylinder where a small stream of solution emulsifies the dust. No air agitation is used. Results of the first month's operation of the new feeder compared with the former method of emulsifying in a cone with air agitation show: Increase in concentration, 80 per cent.; decrease in consumption of zinc per ton of solution, 33 per cent.; decrease in consumption of zinc per ounce of gold precipitated, 31 per cent. The results warrant the statement that a reasonably regular supply of zinc is sufficient for all practical purposes when, as is usually the case, the zinc and solution travel some distance before entering the press.

Continuous Filter Press Process.

In the usual practice of handling oil seeds, fish scrap, slaughter house scrap, bagasse, etc., for the general purpose of removing the liquid from the solid, it is exceedingly desirable to employ a continuous process of operation, but in many types of apparatus now in use it is rather difficult to produce the necessary high pressure. With hydraulic or screw presses the pressure can be easily attained, but the process is essentially discontinuous. Any attempt to make the pressing process a continuous one is therefore of considerable interest.

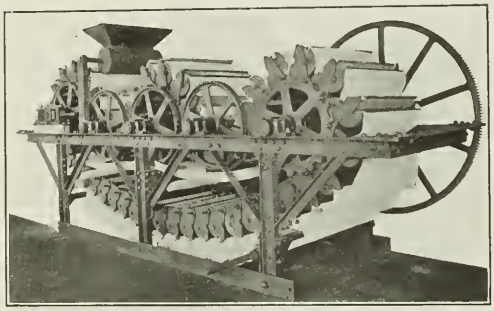
We had recently an opportunity to see in operation a new continuous pressing apparatus which is the invention of Mr. John J. Berrigan, of 136 North Centre Street, Orange, N. J. The apparatus which we saw in operation is reproduced in the adjoining illustration.

It will be seen there is an endless belt, running alternately around the top and the bottom of five successive rolls. But unlike other belt and roll arrangements, the closeness of the pinch between the faces of the belt and roll has nothing to do directly with the pressure to be exerted on the liquid and solid which are to be separated.

The belt is seen to carry a large number of pressure chambers. In the case of the apparatus illustrated their number is 54. The material to be separated is fed into them from the hopper shown on the top on the left of the illustration. Now each of the chambers is changed in shape while traveling with the endless belt around the rolls, that is the walls of the chamber are made to open or close like jaws while they are on top of a roll or on the bottom of a roll respectively.

When a certain chamber reaches the position on top of the first roll, it is wide open and is filled with material from the hopper. Then the filled chamber is carried by the belt downward and the walls are gradually closed, somewhat on the principle of a door jamb. When the chamber reaches the position below the second roll, the pressure becomes a maximum and the liquid is squeezed out and runs off through the filter cloth.

The chamber then begins to rise again and is again wide open when it reaches the top of the third roll. The pressure on the cakes is now relaxed, and at this point, according to requirements, sprays of water, steam or oil may be directed on the cake. When the chamber now begins to pass again downward, the cake in it is shifted from one side to the other side and thereby breaks up. The pressure begins to rise again, and when



CONTINUOUS FILTER PRESS.

the chamber reaches a position at the bottom of the fourth roll there is a second squeeze of liquid from the cake.

Then the chamber rises again and is wide open when it comes to the top of the fifth and last roll. But now after it has passed the top of the last roll and passes downward, it does not close, but on the contrary, when the belt with the chambers gets into the position below all the rolls, the filter cloth turns automatically inside out, as shown in the illustration, and the solid cake drops out.

The capacity of the machine depends on the number of chambers, the size of the chambers, and the speed. The speed is to be adjusted, according to the material under treatment. While the principle of the machine is thus clear, the reader may be referred for details of mechanical construction to three patents recently granted to Mr. Berrigan (994,631, 994,632 and 994,495, all of June 6).

The machine is certainly well designed mechanically. If pieces of wood or melting screws, etc., get incidentally into the charge no harm is done.

Perhaps most remarkable, however, is the small amount of power required since the machine shown in operation, weighing several tons, could easily be driven by hand. This is due to the well-balanced arrangement of the endless belt and accessories.

It will be decidedly worth while watching the introduction of this machine into commercial practice.

NOTES.

Messrs. **Toch Brothers,** the well-known manufacturers of paints, varnishes and enamels, of New York City, with works in Long Island City, have recently opened new works at Toronto, Canada, to supply the Canadian trade.

The **FitzGerald & Bennie Laboratories,** of Niagara Falls, N. Y., have been removed from Falls Street to the corner of Highland and Whirlpool Avenues, Niagara Falls. The new location affords greatly enlarged and improved facilities for the work of these laboratories.

The Elmore Concentration Plant at Sulitelma, Norway, using the oil-vacuum-flotation process, is being enlarged from

a daily capacity of 500 tons to about 900-1000 tons. This will be the largest flotation plant used for the treatment of copper ore. Common fuel oil is used, and the consumption is reported to be 2.7 lb. per ton of ore treated. No acid is used.

The Camp Bird, Ltd., reports that the January operations at its properties resulted in a net balance applicable to profit amounting to $96,515.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in December, 1910, was 68. The total tonnage milled during that period was 1,869,124 tons of ore; 9736 stamps were in operation with an average duty of 7.33 tons per 24 hours. Tube mills in commission numbered 190. The yield for the month was 640,995 fine ounces gold.

The Flow of Heat Through Furnace Walls is the title of Bulletin No. 8, recently issued by the Bureau of Mines. The results of the investigation described in this bulletin justify the following conclusions: In furnace construction a solid wall is better than a wall of the same total thickness containing an air space, as the former is a better heat insulator. This statement is particularly true if the space is close to the furnace side of the wall and if the furnace is operated at high temperatures. If it is desirable in furnace construction to build the wall in two parts, so as to prevent cracks being formed due to the expansion of the brickwork on the furnace side of the walls, it is better to fill the space between the walls with some "solid" (not firm, but loose) insulating material. Ash, crushed brick or sand offer high resistance to heat flow through walls with an air space. Furthermore, any such loose material by its plasticity reduces air leakage, which is an important feature deserving consideration.

The Magma Copper Company is one of the new copper propositions that has recently come into prominence. Unlike many of the Western copper mines, it is a high-grade proposition and will probably be a heavy producer when fully developed. It is in Pinal County, Arizona, 20 miles west of the Inspiration, 12 miles northeast of the Ray Consolidated, and is being developed at depth. The output of the mine will be largely smelting ore.

The Arizona Copper Company produced during the month of April, 1911, 1420 tons of copper.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

362,022, April 26, 1887, H. Liepmann, assignor to the Cassel Gold Extracting Company (Ltd.), of Glasgow, Scotland.

Similar to the preceding, except that the asbestos-cloth diaphragms are supplemented by movable diaphragms of earthenware or compressed layers of asbestos cloth to prevent diffusion of the alkaline liquors from the cathode back into the anode compartment. The supplemental diaphragms are movably carried by screws. In operation the powdered ores are placed in the revolving drum, or anode compartment, the dense partitions being forced up against the permeable ones. The drum being revolved and current supplied, the metals dissolve, but their solution is retained within the anode compartment. The supplemental diaphragms are then withdrawn, and lime added to the anode solution. The solution then passes through the permeable fixed diaphragms into the tanks, where the gold is electro deposited.

370,764, March 20, 1888, Charles F. Croselmire, of Newark, N. J.

Zinc. Pulverizes galena carrying zinc, silver and iron, roasts with an air of steam jet, and treats with dilute sulfuric acid. The zinc, iron and manganese dissolve. The latter are precipitated as oxids by injecting air into the solution. The zinc is then electrodeposited, or it may be precipitated as a carbonate or oxid. The undissolved residue is washed, briquetted and smelted.

387,036, July 31, 1888, Charles Peters Bellows, of Gloversville, N. Y.

Gold and silver. Treats sulfids, titanids, iridids, osmids, chromids, antimonids, to render them free-milling and capable of amalgamation. The ore, crushed to the size of fine gravel, is placed in a perforated crate or basket surrounded by cotton cloth. A copper anode bar is embedded in the ore. The basket is placed in an insulated tub of wood or porcelain containing a solution of sodium chlorid, to each gallon of which is added one ounce of caustic soda. Steam is led into the solution by an insulated pipe, and another steam pipe may extend into the gravel. Current is passed from the anode in the ore to a copper-sheet cathode surrounding the crate, thereby dissolving the refractory substances, or causing them "to chemically combine with the substances in the bath and remain in solution or be deposited in the ore as an innocuous salt."

391,360, Oct. 16, 1888, Henry H. Eames, of Baltimore, Md., assignor to John J. Kelly, of same place.

Gold and silver. Converts sulfied ores into chlorids by electrolytic chlorin, liberated from a solution of sodium or calcium chlorid, acidified by sulfuric acid or copper or iron sulfate. The cell is a circular iron vessel lined with Portland cement and constructed to resist internal pressure. Vertical electrode-plates of copper and zinc are supported on insulators at opposite sides of the cell. Centrally located in the cell is a circular casing for effecting circulation of the solution, having an inlet at the bottom at one side and an outlet at the top at the other side. In one form, this casing contains a revolving agitator, and steam is supplied to the bottom of the vessel through an insulated pipe. In a modification, the casing contains a steam-coil connected to pipes extending through the top and bottom of the cell and having rubber insulating connections. The ore is pulverized to an impalpable powder and is placed in the cell outside of the casing, being kept in suspension by the circulation. In using the first cell, steam is supplied at a pressure of 25 pounds per square inch, maintaining a temperature of 250°.

BOOK REVIEWS.

Practice of Copper Smelting. By **E. D. Peters,** Professor of Metallurgy, Harvard University. Large 8vo., 703 pages, 121 illus. Price, $5. New York: McGraw-Hill Book Company

This is practically a revision to date of "Modern Copper Smelting"—a book so well known and appreciated that its merits need no eulogy. We can, therefore, confine our remarks to the main features of the revision.

Wet processes are outside the scope of the book; electrolytic refining is left out because of it being too large a subject, although it is an almost integral part of the smelting process of producing copper; foreign practice is almost altogether absent. We feel sure that if Dr. Peters were as familiar with foreign practice as he is with American he would have included more of it in his book—that we would have found Mansfield, for instance, represented by more than one little slag analysis. This is the principal shortcoming of the book; it informs Americans wonderfully well on what they are doing themselves, but gives very little information about foreign practice.

There is not a chapter in the book which is not full of valuable information; the work is a high-class treatise on modern American copper smelting practice, which no one interested in copper smelting can afford to be without.

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Trends in Iron and Steel Production.

As most of the important statistics of iron and steel production in the United States in 1910 have now been reported a glance at the results of the year may be of interest. The pig iron output was 27,298,545 tons, and while it broke the previous record, made in 1909, by 3 per cent it was much below the capacity. Indeed, in the twelvemonth from July 1, 1909, to July 1, 1910, production was practically 30,000,000 tons, and conditions were such as to leave room for the belief that in the calendar year between 32,000,000 and 33,000,000 tons of pig iron could have been made. At the lower estimate the actual production was 15 per cent short of the capacity, even though a new record in output was made.

The lead of Bessemer steel over open-hearth, first established in the "off" year of 1908, was more firmly established, as in 1910 production of open-hearth steel ingots exceeded that of Bessemer by precisely 75 per cent, while the lead of basic open-hearth over Bessemer, disregarding acid open-hearth entirely, was 62 per cent.

While the production of electric steel was only the modest quantity of 52,141 tons, this exceeded the production of any other country. As the introduction of this method is very recent, rapid gains in tonnage are to be expected for this year and next.

Without any apparently heavy demand for steel rails the production was 3,634,029 tons, thus exceeding the production of all previous years with the exception of 1906. The production of rails is not destined to see the violent fluctuations which have occurred in the past, when the bulk of the rails purchased were either for the building of new track or the replacement of old and light rails which the rapid adoption of heavy cars and locomotives suddenly made unfit. A large part of the annual rail demand is now due to the wearing out of rails in ordinary service, and as railroad traffic does not vary greatly from year to year the production is correspondingly uniform.

The production of open-hearth rails closely approached that of Bessemer in 1910, the total of 3,634,029 tons being made up of 1,917,900 tons of Bessemer, 1,715,899 tons of open-hearth and 230 tons of iron rails, so that Bessemer production was 53 per cent of the total. The production was much less than the capacity, which may be stated at 7,000,000 tons if all possible equipment be included. It should be observed, however, that many of the rail mills are interchangeable, rolling billets or sheet bars in place of rails upon occasion, and when they are engaged on these products merely the rail finishing department is idle. This flexibility was long ago found to be very desirable on account of the wide fluctuations in rail demand from year to year. It would not be difficult to make nearly as large a tonnage of open-hearth rails in a year as was made of all rails last year, but it is not at all certain that this will be

done. There is still a definite demand for Bessemer rails, and, of course, the difference of $2 a ton in the market price is not an unimportant factor with many roads, particularly for track over which the traffic is relatively infrequent.

The Power Problem for Electrochemical Plants.

In any new electrochemical enterprise, as in any new industrial undertaking, the concrete question always comes up at an early date, "where shall we locate?" It is almost tautology to say, "at the place where we can make the most money." With respect to electrochemical processes an impression prevails that the largest single factor in determining the cost of operation is the cost of power. This is undoubtedly true as a general rule, but at the same time it is true that there are a great many other factors which play an important rôle, and all must be balanced and properly proportioned. When this is considered, the availability of large amounts of cheap hydroelectric power for the electrochemical plant will not be found so absolutely necessary as a cursory examination of the subject might indicate. This is especially true in the case of conditions in the United States. In brief, the location of the electrochemical plant is a question of commercial geography. Besides the problem of power, there must be considered the conditions of the labor market, of shipping facilities by rail or water, of freight rates, of supply of raw materials and fuel and of the market for the finished product.

Each of these elements has its bearing on the final result and each has its own factor of commercial efficiency. In any multiplication of efficiencies each factor, large or small, counts toward the final result; 60 per cent multiplied by 70 per cent by 80 per cent gives a higher efficiency than 95 per cent by 90 per cent by 30 per cent. The balancing of the different commercial efficiency factors so as to produce the maximum net return is the problem which confronts the modern engineer. To consider all the various commercial factors and to evaluate the resulting equation is something that never is perfectly done, but usually only moderately well done. It is not so hard to reason, given certain premises, but to make accurate observations and to draw particular conclusions, especially unpleasant conclusions, is difficult.

It is instructive to consider the conditions which have resulted in the formation of the three great electrochemical centers of the world. Considered in their basic relations, these are Niagara Falls, the coast of Norway and New York Harbor. Decidedly there must be unusual features that have existed to condense electrochemical plants around these places as nuclei. Norway has the cheapest power and is the most remote from the centers of civilization, though transportation by water is convenient and cheap, but the labor supply is rather poor. New York can produce power at a fair, but not low, cost, but is itself a great financial and industrial center and the supply of labor is abundant. Niagara Falls has fairly cheap power, midway between the cost in Norway and in New York, and lies between three great American manufacturing districts, with a good labor market. We can see the pretty balance of the commercial factors in the different characteristics of the products made in Norway, New York Harbor and Niagara Falls by Electra's fiat.

Norway makes principally nitrogen products—both nitrates and cyanamide—from atmospheri air. She needs cheap power and air. Power is cheaper in Norway than anywhere else in the world. Air is cheap anywhere, hot in the tropics and from the promoter's lips, but no location has any monopolistic rights on air. Electrochemical plants locate in Norway when the cost of power is the greatest determining factor.

The electrochemical industry of New York Harbor—located in New Jersey as well as on Long Island—is entirely the electrolytic refining of copper bullion. The closeness of the brass and copper mills, the ease with which coal for reverberatory smelting is supplied, the dominance of American copper mines, the extent of the labor market, the Metropolitan banking facilities whereby loans on the valuable staple can be negotiated—all these factors have contributed many advantages for this special purpose, so that almost all Western copper smelters now ship their bullion East for refining and 75 per cent of the copper of the world is refined near New York.

At Niagara Falls power during the inception of the electrochemical industry from 1890 to 1900 was abundant and cheap (and while power is no longer abundant there for electrochemical work, it is still cheap). To this had to be added the good conditions for labor and supply of raw materials and the closeness to the potential markets for the then new products, such as aluminium, carborundum, graphite and calcium carbide. The optimist can see here why things as they are now have succeeded. Copper refining is impossible in Norway and nitrate or cyanamide manufacture from air is impossible in New York Harbor. Each is possible within certain limits at Niagara Falls, where the manufacture of nitrates from air has been a failure and the manufacture of cyanamide is a success.

Having in mind the bearing of commercial conditions on the location of these three electrochemical centers, we can see that in the United States we have widely varying possibilities for electrochemical industry. After the first period of industrial electrochemistry is over, characterized by the creation of new industries for the manufacture of hitherto unknown products, electrochemical activity has entered during the last year into a new era by taking hold of a wider and broader field. Electrochemical methods are being introduced into the older metallurgical and chemical industries. Steel refining in the electric furnace is the most momentous single feature of this evolution. But there loom up now at the horizon the latent potentialities of electrochemical methods—both wet and furnace methods—in non-ferrous metallurgy in the big and beautiful West, with its unlimited metallurgical possibilities; and it is no exaggeration to say that no one appreciates more clearly these potentialties of electrochemistry than the progressive Western metallurgists themselves. So this development is bound to come, and, in the nature of things, it must be profoundly affected by the availability of one or the other source of power.

Three sources of power are essentially available—water, gas and steam. The beginnings of electrochemistry were intimately connected with water-power developments. Besides Niagara Falls and Norway, this is true for Schaffhausen, for France (especially Savoy) and Italy, as well as for numeous smaller centers of electrochemical activity, of which only Sault Ste. Marie, Lockport, Penn Yan in this country may be mentioned; and it is noteworthy that the erection now going on of the first American works for the manufacture of nitrates from atmospheric air is intimately connected with the water-power de-

velopment of the Southern Power Company. Yet water-power has its limitations for electrochemical work in two directions. Firstly, if the specific nature of an electrochemical plant requires a location not too remote from an industrial center, such location means the nearness of existing industrial plants that can pay a higher price for power than an electrochemical plant can usually afford; if the electrochemist and the power salesman come together and the electrochemist offers half a cent per kw-hour (which may be the limit he can afford to pay) he is liable to turn the promoter of the hydroelectric concern irate. Secondly, the number of water-powers which can be developed very cheaply is limited. In many cases the engineering difficulties to be overcome in construction of dam and pipe line are so enormous and the capital investment becomes so high that, in spite of the relatively low operating cost, the rate charged for power must be high, as it is in at least one instance which we have in mind in the West. Nevertheless, it would be wrong to assume that such a water-power plant could not expect to sell power for electrochemical work. A water-power plant of this kind would not be built in a district in which a gas plant or a steam plant could supply power at less cost. And, on the other hand, if an electrochemical plant is erected in such a district, this is done in spite of high power cost, for the sake of the other advantages which the district may offer.

The erection of its own hydroelectric power plant does not seem good business for a new electrochemical enterprise in most cases, in view of the large capital investment that would be required. It is generally hard enough to get together the usually considerable capital which is required for the electrochemical installation itself proper. It is, therefore, better business, at least in the beginning, to buy the power from a water-power company. Water-power companies are usually reasonable in such circumstances.

From water-power we turn to gas. The gas engine is an attractive proposition, especially in a natural gas field or where blast furnaces or by-product coke ovens furnish cheap gas. The capital cost is much lower than that of a water-power plant and for a natural gas plant one-fourth cent per kw-hour may be considered a fairly representative figure. Indeed, were it not for the uncertainty about the life of the natural gas field, power generated from natural gas in a gas engine would be ideal for the electrochemical plant. The minimum figures for both capital cost and operating cost exist in this case. With blast-furnace gas or with by-product coke-oven gas both the capital cost and the operating cost are somewhat higher, but, on the other hand, the uncertainty is removed. Further, the gas will not be given away and some charge must be made for it. Probably the two most magnificent blast-furnace gas-power plants in existence are the two U. S. Steel Corporation plants, working in parallel, at Gary and South Chicago. At South Chicago the charge made to the electric furnace plant for steel refining by the Héroult process is half a cent per kw-hour. Of course, for electric steel refining the blast-furnace gas plant is logically the ideal solution of the power problem.

In many cases producer gas offers decided advantages for electrochemical work, as well as for chemical and metallurgical work in general. We have especially in mind a chemical manufacturing plant in Denver, which is conspicuous by the ingenuity and originality of its conception and methods. It shares with Standard Oil the distinction of turning absolutely everything into some useful product. Cheap bituminous slack is used in the gas producer and the cost of power is something like half a cent per kw-hour. But as the company refuses to be discussed in print, its plant is simply mentioned here to show that the advantages of producer gas for chemical work do not only exist on paper, but are a proved reality.

In turning finally to steam the advances made by the steam turbine in recent years are certainly striking. At the present time steam turbo-generators and accessories are sold very cheaply. The use of high speed and high pressure means that a maximum of $\frac{1}{2} m v^2$ is raced through a minimum of cast iron, steel and copper. Whereas the capital cost of a steam power plant was formerly $70 to $80 per kilowatt, the enforced competition among the companies building machinery for generating stations and especially the competition against the needs of the industries has reduced this figure about 25 per cent. Hard times make low prices. With coal of fair quality at $2 per ton f.o.b. plant and with plenty of condensing water, electrochemical power can be furnished by the turbo-generator very cheaply—so cheaply indeed as to make the average electrical engineer look skeptical.

But it must be remembered that the average electrical central station engineer does not appreciate the conditions of the electrochemical plant and its requirements. The usual electrochemical plant has a load-factor of over 95 per cent and also a high power-factor. Moreover, its land is cheap as compared with city land and its taxes low. Remembering all this, it is probably no exaggeration to estimate that with all proper charges power can be supplied under the conditions stated above at one-third of a cent per kw-hour, if the plant has a capacity of 5000 kw. Under the usual load conditions of central stations the cost will probably be more than twice this figure. There is necessarily a vast difference between a plant designed for and operating under a fluctuating load, as is the case with a central station, and a plant designed for a maximum of economy under a load that is fixed and constant, as is the case with the power plant for an electrochemical works.

While we have not discussed the possibilities of the steam engine for electrochemical work at all, it is but fair to say that this old reliable prime mover has been doing and is doing faithful service in numerous electrolytic copper refineries and will hardly be discarded here in the near future.

To sum up, the object of this discussion was to emphasize two points. Firstly, that the power cost is not all-important in an electrochemical process, as the different efficiencies must be balanced against each other. Secondly, that every kind of prime mover as its field in the electrochemical power plant. On the streets of New York City the push cart, the horse-drawn vehicle, the gasoline truck and the electric truck compete with each other in carrying goods. Each is used according to the needs and ability of the owner. So, in supplying electric power for electrolysis, electric smelting and refining, the steam engine, the hydraulic turbine, the gas engines and the steam turbine will continue to compete against each other. This freedom of choice in the power plant augurs well for the inevitable expansion of the electrochemical industry.

Chicago Meeting of the American Institute of Chemical Engineers.

The third semi-annual meeting of the American Institute of Chemical Engineers was held in Chicago from June 21 to June 24. All business sessions and meetings for reading papers were held at the Congress Hotel.

At the opening meeting on June 21 the members of the Institute were welcomed to Chicago by Mr. Wade, representing the Mayor of Chicago, and by Dr. T. B. Wagner, representing the Local Committee. Members were invited to visit the important chemical industries of Chicago, many of which were inspected during the meeting.

The reports of the various officers and committees were then presented. The membership committee reported a healthy growth, the present membership being 150. A very large number of requests for information with reference to conditions of membership are received, but on account of the high requirements for membership the number admitted is small.

The secretary reported that the third volume of Transactions would soon be issued. The growth in papers published is shown by the fact that the first volume contained 11 papers and 204 pages, the second 19 papers and 308 pages, while the third will contain 25 papers and about 410 pages. All papers are read by every member of the publication committee, which results in the rejection of papers of little value and the improvement of those published. A rapidly increasing demand is noticed for the Transactions.

The report of the treasurer showed a balance on hand of about $1,100.

The committee on medal reported $300 on hand and presented for approval or criticism a number of designs for the medal. The committee was opposed to the bust of a prominent chemical engineer on the medal or the use of the name of any chemical engineer for the medal, a symbolical devise being preferred. Non-members as well as members may compete for the medal which shall be known as the American Institute of Chemical Engineers' Medal, but the paper for which the award is made must be presented to the institute and published in the Transactions.

The report of the committee on chemical engineering education was read by the chairman, Dr. Samuel P. Sadtler. The most important part of this report consisted in the digest of the replies received to the questionaire on chemical engineering education sent out with the December Bulletin of the Institute. Thirty-seven replies were received to the 19 questions submitted. In reply to the question as to whether the prevailing system of chemical engineering education is adequate an emphatic "no" was given in a large majority of the answers. As the question evidently referred to a four-year course, the vote may be taken as approving the present tendency to lengthen the chemical engineering course to five years.

With reference to the subjects to be included in the course on chemical engineering a number of responses stated emphatically what seemed to be the general opinion that students should not waste time in college playing with "chemical practice," but that fundamentals should be thoroughly taught and experience with practical work acquired in the factory.

With reference to the defects of present courses the opinion seemed to be general that the college failed to devote sufficient time to those chemical manufacturing operations and appliances which constitute the element or operative units out of which nearly every chemical factory is built up.

A paper stating what could be accomplished in the four-year chemical engineering course was then read by Prof. J. H. James, of the Carnegie Technical School, of Pittsburg.

A paper giving a summary of the work of the committee on five-year engineering courses of the University of Ohio was read by Prof. James R. Withson, of the same university. The report of the committee and the two papers read on education will be printed in the form of a bulletin so as to afford discussion by mail.

During the afternoon the plant of Swift & Company was inspected. Dr. William D. Richardson gave a brief explanation of the processes used after luncheon had been served by the company. The entire operation of killing, cutting, curing, smoking and cold storage as well as the soap plant and glycerine refinery were shown.

During the evening President Frerichs delivered an able address on "Methods of Attacking Chemical Industrial Processes." Complete detailed drawings and models of an installation for the manufacture of soda by the Solvay process were shown. The great difference in conditions and prices between Germany and the United States was shown by estimates on a process which would be a failure here, although very successful in Germany. A process for the extraction of bismuth from its ore was also given in detail.

On Thursday morning by a special car members reached Argo, Ill., for a visit to the large new plant of the Corn Products Refining Company. The very extensive use of concrete in this plant was noted and members were interested to learn how this material would withstand the action of sulphurous acid. All parts of the plant were inspected, including the converters, sugar and starch houses. When the plant is completed it will have a capacity of 100,000 bushels of corn per day. Many of the members spent the entire day at the plant. Others visited the laboratories of the Dearborn Drug & Chemical Works, after which an automobile sight-seeing trip around Chicago's beautiful boulevards and parks was taken.

On Thursday evening a subscription dinner was given at the Congress Hotel. The most interesting feature of the dinner was the presentation of the institute medal to Mr. F. J. Wood, of Marx & Rawolle, for the presentation of his paper on "The Multiple Effect Distillation of Glycerine," which was published in volume 2 of the Transactions of the institute. The process is remarkably economical in coal consumption, ease of control and operation as well as purity of product. It is in operation at the Brooklyn plant of Marx & Rawolle as well as in Toronto, Canada, and in England.

Dr. Edward Gudeman, of Chicago, acted as toastmaster and kept the company in the best of humor. The first speaker was President Frerichs, who called attention to the fact that the institute was organized just three years ago with a charter membership of 40. Mr. A. Bement, Dr. Geo. A. Rosengarten, Secretary Olsen and Prof. A. H. White also gave toasts.

On Friday morning the following papers were read:

"Industrial Chemical Calculations," by Prof. Jos. W. Richards. This paper will be printed in form of a bulletin and will come up for discussion at the next general meeting.

"Rapid Determination of Tin in Food Products," by Dr. Edward Gudeman.

"Distillation on the Continuous System," by Chas. L. Campbell. This paper gave a complete discussion of both the theory and practice of purification by distillation of complex liquids. Very fine samples of lead and silver linings on copper and steel were also shown by Mr. Campbell.

Friday afternoon the plant of the By-Products Coke Corporation at South Deering was visited, while some members visited the works of the Carter White Lead Company at West Pullman.

On Friday evening a paper by Dr. William M. Grosvenor on "The Institute and the United States Patent System" was read. In this paper the defects of the present patent system were shown. A communication from the St. Louis section of the American Society of Electrical Engineers was read on the same subject. It was urged that a special patent court should be established and that judges should be permitted to employ experts for independent advice. A committee was appointed to draft resolutions to be presented to President Taft urging reform of the patent system.

A paper on "The Practical Value of Calorific Tests on Anthracite Coal" and another on "Two Methods of Testing Asphalt," by S. F. Peckham, were read.

On Saturday the plant of the Indiana Steel Company at Gary, Ind., was inspected. An inspection car was provided to carry the visitors around this very extensive plant. Aside from its huge size the most noticeable feature was the clean and orderly appearance of the entire plant. The coke used is produced in Lemet-Solvay by-product ovens from which 80 tons of ammonium sulphate are produced daily. The motive power used is largely electrical, produced by 17 3600-kw generators driven by 34 gas engines using furnace gases. The open-hearth furnaces, of which there are 14, are heated by producer gas. Much interest was shown in the huge electrically driven rolling mill.

A portion of the members also visited the Standard Oil Refinery at Whiting, Ind., as well as the Universal Portland Cement Company at Buffington, Ind.

The attendance at the meeting was about 50 and marked interest was shown in all the sessions and excursions.

The Iron and Steel Market.

The moderate burst of market activity in June has been followed by a very dull July. The June activity was clearly due to the release of orders which had been piling up during the period of uncertainty in the minds of buyers as to whether prices would decline and which released by the reductions of May 29 indicating that price questions were settled for the time being. After this artificial freshet the stream of business dropped to the low level which is normal for midsummer. Rather strenuous efforts were put forth in the fore part of July by interested parties to conceal the relapse and make it appear that the market was still active. Incontrovertible evidence was furnished, however, that new buying was very light and that the mill activity, although reduced slightly from the June average, was still due in considerable part to business which had been booked in June. By the closing week in July mill activity had been further reduced, the average output of finished steel products by all interests being probably well below 60 per cent of total capacity, whereas in June a level of about 70 per cent had been maintained for a time.

The market has been quite colorless as to distinct developments. Buying has been light simply because to the reserve felt by buyers in a general period of conservative business action there is added the restraining influence of its being the summer season when many men are away on vacations and business is normally light.

The control of finished steel prices is substantially as firm as it has been at any time, though possibly not as secure as before the action of the Republic Iron & Steel Company in May, withdrawing from the counsels of the steel producers and pursuing an independent course as to the price of steel bars. This company is maintaining the same prices as other interests, but it has not formally returned to the fold, and with that example before the trade the possibility of one interest or another electing to pursue independent action, should such a course appeal to it, is somewhat increased. In no quarter can be seen any influence which would tend to advance prices, while the possibility of accident effecting reductions at one point or another, if remote, is not invisible. Should it at any time become apparent that a general reduction in prices would be likely to stimulate such a general buying movement as to fill up the mills and make possible an advancing market through individual action of producers, there is every reason to believe the course of declaring an "open market" would be pursued. This was probably the main influence prompting the break of February, 1909, for while that movement had the appearance of being an accident, time has given evidence that the course of the market, with a great dip and subsequent recovery, was fairly well thought out in advance.

Only a small number of the mills closed July 1 for repairs and inventory. In some cases the only closing was for Tuesday, Independence Day, while in other cases Monday was taken as well and in fewer instances the whole week.

The wage scales of the Amalgamated Association have all been signed. These scales affect only certain iron, sheet and tin mills. Most of the iron mills of the Central West are governed by this iron scale, which was signed before the expiration of the old scale on June 30 by the Republic Iron & Steel Company and by the mills in the Western Bar Iron Association after a two days' conference at Cambridge Springs, July 6-7. The leading sheet and tin plate interest is entirely non-union, while about 40 per cent of the independent mills are organized. These agreed upon a continuance of the existing scale.

Little buying of wire products has followed the reduction of $2 a ton effective June 20. Existing contracts were adjusted to the new basis, and for a fortnight specifications on such contracts were slightly increased. It is between seasons for the wire trade, but the usual fall buying is expected to commence during the second half of August.

Pig Iron.

The drop of Southern iron to a basis of $10, Birmingham, which occurred in June did not bring out the buying for second half which was expected in many quarters. Early in July the directors of the leading cast-iron pipe interest, which frequently leads in the buying of Southern iron, met and concluded not to buy for the present. Other large consumers have shown little interest, and buying of Southern iron has continued from hand to mouth. In most cases of market swings Southern iron leads Northern by from two to four weeks, and Northern iron has also lagged, there being scarcely any forward buying, while here and there prices have softened slightly. A very interesting development has been the continued blowing-out of merchant furnaces in the Mahoning and Shenango Valleys until only six of the eighteen are in blast. Most of the furnaces which went out merely kept in blast until their supplies of ore were exhausted, while purchases were not made for the current season. If these furnaces do not take action soon they will pass into the winter without ore and an unprecedented situation will be revealed. Pig iron prices are regarded as below cost of production to these furnaces, unless they have ore supplies of their own, and can use the profit on ore to cover the loss on pig iron. A number of lake front merchant furnaces are idle, but not nearly as large a proportion as in the valleys. There has been further blowing-out of furnaces in Virginia and eastern Pennsylvania, and at last it is believed that production of merchant iron is below the consumption. The market at the close of July may be quoted as follows: F.o.b. Birmingham, $10 for No. 2 foundry; delivered Philadelphia, $15 for No. 2X foundry and $14.25 to $14.50 for basic; f.o.b. valleys, 90 cents higher delivered Pittsburg, Bessemer, $15; basic, $13; No. 2 foundry, $13.50; gray forge, $13; malleable, $13.25 to $13.50.

Steel.

Specifications for July deliveries of sheet bars against contracts have been very good, fully equaling those of June and exceeding those of May; billet specifications have been lighter than for June. There has been scarcely any buying in the open market. Billets continue quotable at $21 and sheet bars at $22, f.o.b. maker's mill, Pittsburgh or Youngstown, while with the June reduction of $2 a ton rods are quotable at $27, Pittsburgh.

Finished Products.

Prices of finished products, f.o.b. Pittsburgh, except where otherwise stated, are as follows:

Plates, 1.35 cents for tank quality, ¼ in. and heavier.

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Shapes, 1.35 cents for beams and channels, 3 in. to 15 in. inclusive, zees and angles 2 x 3 in. and larger.

Steel bars, 1.25 cents, base; iron bars, 1.25 cents to 1.30 cents, Pittsburgh; 1.27 cents to 1.30 cents, Philadelphia, and 1.20 cents to 1.25 cents, Chicago.

Wire nails, $1.70 per keg, base; plain wire, 1.50 cents; galvanized barb wire, 2 cents.

Black sheets, 28-gage, 2 cents; galvanized, 3 cents; blue annealed, 10 gage, 1.50 cents; painted corrugated roofing, $1.40 per square; galvanized, $2.55.

Tin plates, $3.70 per base box for 100-lb. cokes.

The Western Metallurgical Field.

Chemical Engineering at the University of Washington.

The growth of chemical engineering education in the West is typified by the work done in the University of Washington at Seattle. Special courses leading to the degree of Bachelor of Science in Chemical Engineering are open to students, under the direction of competent industrial chemists who are directing attention to the solution of some problems of the Northwest.

Referring to the course of study, the work in mathematics, physics and shopwork, is the same as that required for students in civil, mechanical and electrical engineering. The general engineering work taken by the student in chemical engineering includes mechanical drawing, surveying, mechanics and hydraulics from civil engineering; machine design, steam engineering and experimental engineering from mechanical engineering, and industrial electricity from electrical engineering. This work comprises 54 per cent of the chemical engineering course. Special emphasis is laid on the fundamental processes underlying industrial chemistry and the student is taught general inorganic chemistry, analytical, organic and physical chemistry, with supplementary courses in special lines of work. The study of raw materials used in manufacturing processes in the Northwest also receives attention and a special laboratory is provided in which industrial processes may be tried out on a suitably large scale.

This special laboratory is equipped with a steam still, steam superheater, continuous extraction apparatus, hydraulic press, filter press, alcohol still, wood pulp digester, steam jacketed vacuum still, vacuum dryer, clay testing furnaces, pulverizers, melting furnaces, rotary cement kiln, wood grinder paint mills, mixing machines, electrically heated wood distillation retort and cement testing apparatus.

The university invites the co-operation of local industrial concerns in solving industrial chemical problems. To this end the concern desiring any investigation to be made furnishes the means for prosecuting the work, and the university places its industrial laboratory at the disposal of a competent investigator. In this way it is hoped that some problems may be solved to the mutual advantage of the university and the community.

Dr. H. K. Benson, who is in charge of the work, has organized his students into a Society of Undergraduate Engineers for the purpose of better informing the student of the possibilities of chemical engineering, and giving him a broader outlook of the field before he enters commercial work.

First National Copper Smelter to Close.

An echo of the old controversy between the farmers and smelters of northern California is heard in the report that the First National Copper Company is to close its plant at Coram. This smelter, better known perhaps as the Balaklala, is one of several that were originally ordered by the court to make arrangements to abate the nuisance alleged by the farmers to arise from the free discharge of smoke and fume into the atmosphere. Pursuant to that decree the company installed, at considerable expense, what is considered to be the most efficient means known of condensing fume and reducing the sulphur content of the discharged gases. Perhaps it was too sanguine, and agreed to more rigid stipulations than were necessary. In any event, it appears that the company is unable to live up to its agreement, and the consequence is that the farmers have pressed their case to the extent of causing the company to cease operations.

The decision probably will have a more widespread effect than merely to close this one plant. Other smelters operating in the vicinity are working under an agreement to remove noxious solid and gaseous substances from their furnace gases, and the farmers will make tests to see that the provisions are being carried out. The First National Company undoubtedly will continue to make refinements in its methods in the hope that the plant can be operated some time in the future.

Bureau of Mines.

Dr. Joseph A. Holmes, director of the Bureau of Mines, was in Denver last month en route to Alaska to investigate conditions there for the Department of the Interior. While in Denver he was entertained by Colorado mining men who took advantage of the opportunity to present to him the mining situation in the state, and point out the possibilities for federal assistance through the Bureau of Mines. Perhaps Colorado is not so much in need of new mines as improved metallurgy, and it was on this theme that the speakers at two meetings dwelt at length. An effort was made to impress the Director of the Bureau with the fact that there were many problems to be solved in connection with the treatment of low grade ores, and that it would be to the advantage of the mining and metallurgical industry in general if these problems could be attacked by the Bureau. Federal assistance is sought because a long line of experimentation must be planned and pursued, and private enterprise is not always equal to the task.

Dr. Holmes explained that the Bureau was in full sympathy with the needs of metalliferous mining and the problems of ore treatment, but that there were no funds available with which to carry on such work. The metal mining interests will have to impress Congress with the need of this work if they expect to secure the help of the Bureau of Mines in solving their metallurgical troubles.

Improvement at Cananea.

In the annual report of L. D. Ricketts, president of the Cananea Consolidated Copper Company, some interesting details are given of the metallurgical work during 1910. The impounding of tailings and the recovery of the water is now accomplished in an inexpensive manner. The tailings are stacked in such a way as to allow the water to settle back of the dam. This water contains about 0.5 lb. copper per 1000 gal., and this metal is recovered by precipitation on iron before the water is re-used in the mill. The copper recovered repays the cost of impounding the tailings and in the future no more of these will go down the creek. The concentrator is making a saving of 75 per cent, which is exceedingly satisfactory.

In the reduction division the cost of treating a dry ton of new copper-bearing material was $2.69 as compared with $3.09 in 1909. Reverberatory costs increased slightly, but this will be changed when the extensive improvements inaugurated last year are in operation.

Tests of dust losses in the various departments showed that serious loss was suffered in the converter fume, amounting to about $30,000 a year. Dust chambers and a stack were accordingly constructed to handle these fumes, and were so proportioned that when all converters are running the velocity of the gas in the flues will not exceed 3 ft. per second. As a result of all the improvements made at the plant Dr. Ricketts predicts that the cost of treating a ton of new copper-bearing material in the reduction division will be not more than $2.25.

Drying in Vacuum.

By B. Viola, M.E.,
Member American Society Mechanical Engineers.

(The conclusion of the serial on evaporators and vacuum pans, published in the April, May, June and July issues.)

If materials form crystals, as in the manufacture of sugar or other crystallizable materials, we are able to concentrate the liquors till the crystals begin to form, let the whole cool off and separate the superfluous liquors from the crystals by some mechanical device.

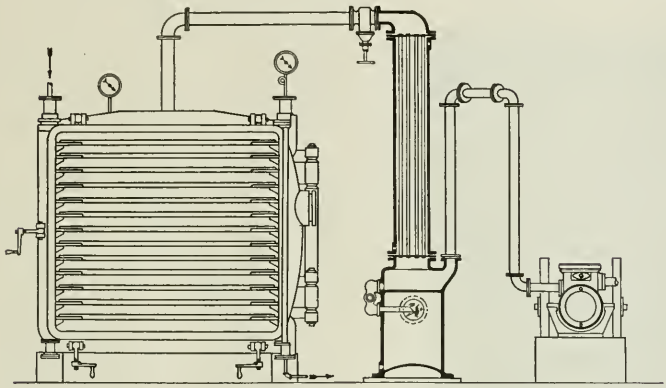


FIG. 50.—ASSEMBLY OF THE PASSBURG VACUUM CHAMBER, DRYER AND AUXILIARIES.

It is, however, different when we want to get the dry substance of a solution which does not crystallize. Then all the dissolving liquor (solvent), whether it is water or some other liquid, has to be evaporated. To a certain density these solutions can be evaporated either by open fire or in a vacuum evaporator, yet in a great many cases the concentration does not suffice; in fact, when it is desirable to remove all dissolvents, it has been found impossible to do so by such an evaporation without a detrimental effect to the dry product, for the reason that such a high concentration causes an overheating of the material, and in this respect vacuum pans fail.

As the water contained in the solutions is reduced, the ebullition of the liquid becomes more sluggish and the temperature of the boiling point of the remaining water rises; consequently an overheating of the material when the final stage of concentration is reached, can very easily take place. The degree of concentration where such overheating would be injurious varies with different materials.

The concentration of liquids by the open or vacuum-pan method, effective as it is, fails more or less according to the material when the last traces of moisture have to be removed.

To supply the need thus created, and realizing the varied uses to which such an apparatus might be put, Messrs. Emil Passburg and W. Strohn, about thirty years ago, commenced investigations and conducted a series of practical experiments from which ultimately the invention of the vacuum-chamber dryers, Fig. 50, were brought about. This is the foundation of the present system of vacuum drying.

The apparatus consists of an evacuated iron chamber fitted with steam-heated shelves or horizontal coils, on which the drying material rests in comparatively thin layers on trays. The result obtained from these vacuum drying chambers was for many materials highly satisfactory, but at the same time a certain amount of labor was involved in charging the trays with the wet material and again emptying them from the dry material. An apparatus which gave, if possible, still better results than the chambers, and at the same time insured a great saving of labor, was the outcome of the constructions, Figs. 51 and 52.

This apparatus, Fig. 51, which is being used in many industries with the greatest satisfaction, is known as the Passburg vacuum single-drum dryer. The apparatus consists of an iron casing of sufficient strength to withstand the outer atmospheric pressure when evacuated, which is very considerable, approaching about 15 lb. to the square inch, a steam-heated rotary drum which dips into the liquid and coats itself with a thin film.

On account of this very thin film of absolute evenness, which can only be obtained by dipping, the wet material is dried in one revolution, the number of revolutions, of course, being regulated according to the nature of the material, the percentage of water it contains, etc.

The thickness of the film of the rotary drum is regulated by the level of the solution contained in the trough or lower part of the outer casing of the apparatus and into which the drum dips. For this purpose observation glasses are provided, by which means the level of the liquid can easily be observed and regulated by a valve.

In the trough provision is also made for coils, through which the cold water passes in order to keep the liquid at the required temperature below the boiling point in the particular vacuum maintained in the apparatus. This feature is quite necessary, where milk, glue, and similar materials are being dried. Where, however, emulsions have to be treated, such as white lead, etc., and where a rapid precipitation of the solids

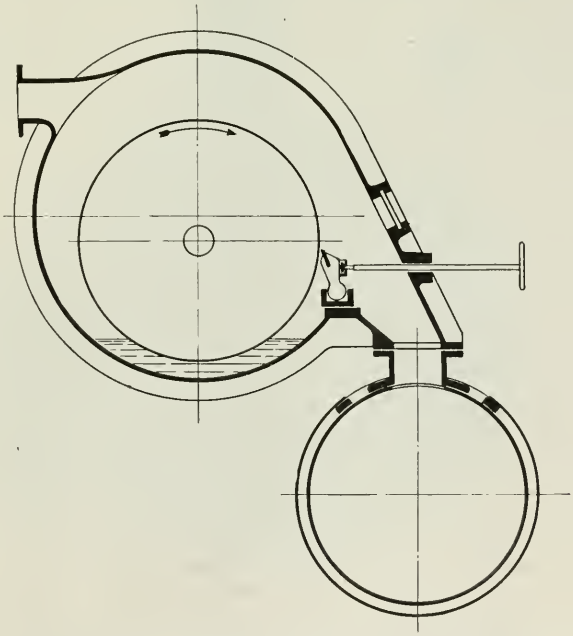


FIG. 51.—CROSS SECTION OF A PASSBURG SINGLE DRUM DRYER

may occur, the place of the cooling coils is taken by suitably-designed agitators.

The ends of the rotary drums are protected by shields so that no liquid whatever may coat the former; in fact, it is only the cylindrical drying surface of the drum, which is at each revolution coated with and freed from the dried film and which comes into contact with the liquid to be dried.

The temperature to which the drying film is exposed rarely exceeds 95 to 122° F. (35-50° C.), and as heating steam of from 20 to 25 lb. pressure may be used for heating the rotary drum, it will be apparent that the enormous difference between the temperature of the heating medium and of the boiling point of the water contained in the wet material, which in such a case as given above approaches 176° F. (80° C.), will insure a most rapid evaporation of the water, notwithstanding the low temperature at which such evaporation takes place.

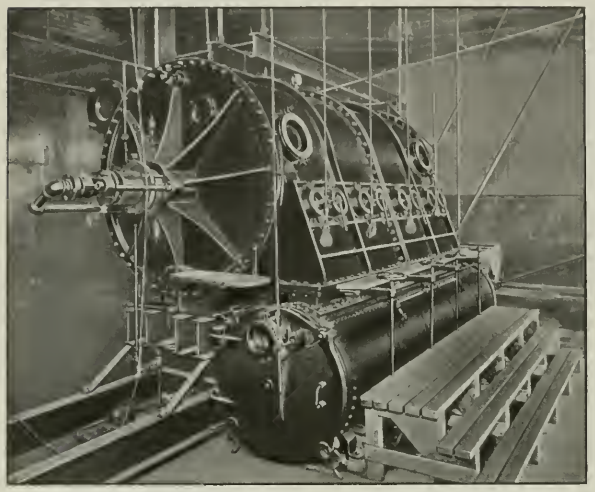


FIG. 52.—ASSEMBLED PASSBURG VACUUM SINGLE DRUM DRYER.

These drum driers work absolutely continuously and automatically, the liquid being drawn in by the vacuum maintained in the dryers and the dry material, which is removed by means of a scraper from the drum, drops into suitable receivers which can be arranged in such a manner that they can be emptied alternately and periodically without breaking the vacuum in the apparatus, and consequently without even temporarily interfering with the drying process.

Such a thin film of wet material in direct contact with the heating surface requires only a fraction of a minute to have all the water it contains evaporated therefrom by boiling; that is, by rapid and not by slow evaporation, and as this process takes place under a high vacuum, the temperature to which the drying material is subjected is consequently a very low one.

The number of revolutions of such an apparatus varies from 4 to 10 per minute, or in other words, the time during which the drying material is subjected to a low temperature can be counted by seconds.

Provision is also made by specially-designed apparatus, so that no loss of dust occurs.

The saving of steam, as no loss by radiation takes place, is naturally very great.

These apparatus offer the further advantage that where material of a poisonous nature, as, for instance, cyanide of potassium, white lead, etc., are dried therein, the laborers are fully protected, since by means of the automatic feed and discharge of such rotary-drum dryer neither the wet nor the dry material enters the drying room, and consequently all detrimental effect on the health of the laborers is absolutely prevented.

The author had the opportunity to work with such apparatus and to find the great advantage of the system, comparing it with the old-time hot-air drying system; that is, by slow evaporation. The most important fact is that the received dry material is always of the same quality, supposing a little attention is paid to the apparatus.

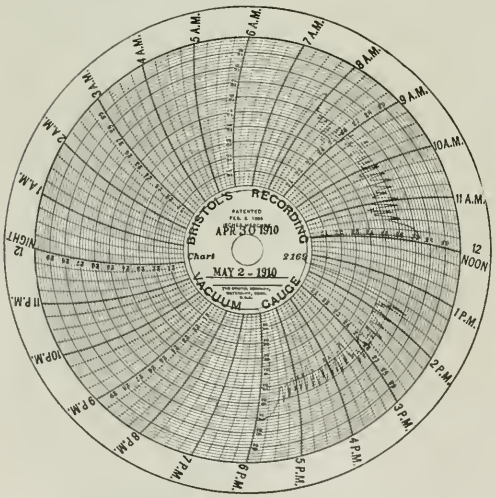


FIG. 53.—VACUUM RECORDING CHART OF A SINGLE DRUM VACUUM DRYER.

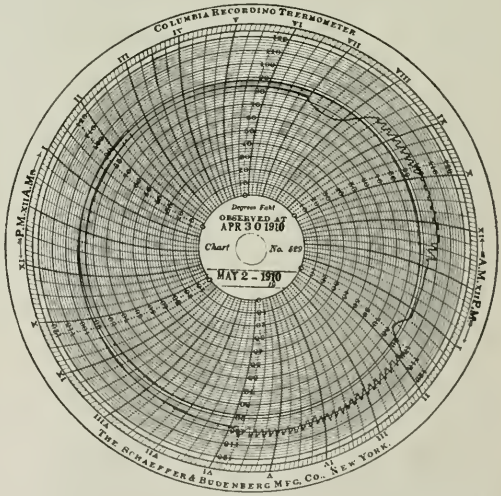


FIG. 54.—TEMPERATURE RECORDING CHART OF THE SURFACE CONDENSER CONNECTED WITH A SINGLE DRUM DRYER.

The feed into the drum should always be carefully watched so that the liquid has approximately the same height in the receiver, and the heating surface of the heating drum be kept steady. Should the liquid in the receiver be used up to a certain extent, then it has to be fed again. This can be done either

by some automatic device so that the feed of the solution corresponds with the evaporating capacity of the drum, and if the former is once set for that purpose the liquor level within the dryer, into which the drum dips, will automatically remain the desired one, suitable for the material. Or it can be done by the attendant himself.

In this way little variations can be noticed on the vacuum, as

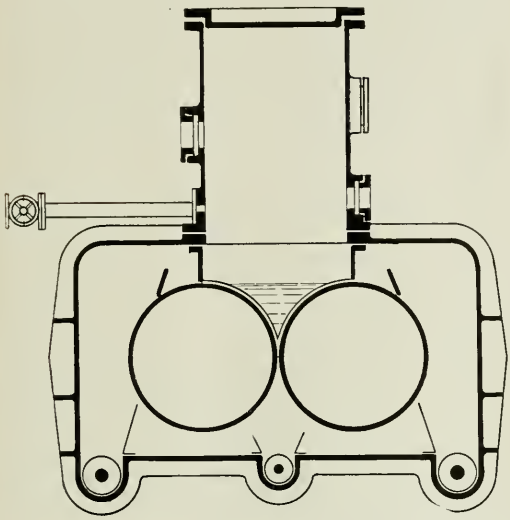


FIG. 55.—CROSS SECTION OF A PASSBURG VACUUM DOUBLE DRUM DRYER.

naturally, by feeding the drum with fresh liquid, a certain quantity of air is drawn in. These changes can be seen plainly on Fig. 53, the recording vacuum chart, and hardly amount to more than 1 in.

At what vacuum the drying process has to be performed depends upon the material, and it is an important fact that no two materials can be handled exactly alike. Every material must be treated on its own merits, and oftentimes small auxiliaries are very essential to the success of an entire apparatus.

Another important factor in vacuum drying, which affects eventually also the vacuum, is the temperature in the condenser. This should be kept approximately constant, and if this involves some difficulty it is the best to equip the plant with a temperature controller, to regulate the flow of the condensing water.

The chart of such a controller can be seen in Fig. 54, where the variations of this temperature are not more than 3° F., giving an excellent result.

The most remarkable difference can be seen on the chart at 7 a. m., when the recording thermometer shows about 81½° F., this being the temperature of the working room; but at that time the cold water was turned on, the drying began, and by and by was raised to the working temperature. At 11:45 a. m. the drying process was stopped again till 1:15 p. m., and this variation of the temperature can be seen.

Though the above-mentioned apparatus have proven most useful for drying liquids, still they could not be employed where materials of high concentration or pasty or similar form had to be dried, and this led to the invention and successful introduction of Passburg's vacuum double-drum dryer as illustrated in Figs. 55 and 56.

With this apparatus the wet material is either pumped or forced by some other means into the evacuated chamber, within which two steam-heated drums rotate as shown. The material is fed on top of the two rotary drums, the latter by their rotary movement coating themselves by pressure with a thin film of the wet material, which is also dried in one revolution, similar to vacuum dryers containing a single drum only.

The thickness of the film on both drums is regulated by the space between them, which can easily be adjusted. Provision is made that any wet or insufficiently dry material which may drop from the drum is collected and again conveyed back to the feed pump, so that no loss in that respect occurs. The dry film is also removed from the drums by means of scrapers similar to the one employed in the single drum, and the dry material drops into suitable receivers, which again are emptied in a similar manner as described with single-drum dryers.

For materials which do not coat the rotary drum with an even film by means of the drum dripping into the material, a single-drum dryer with roller feed can be used, Fig. 57. This kind of dryer is used for heavy color pastes as produced in the manufacture of aniline dyes and similar materials. In fact, these pastes are so heavy that they cannot be drawn into the dryer by means of the vacuum maintained and have to be forced into the apparatus by means of the feed-pump.

From the construction or design of the above-mentioned vacuum shelf, single or double-drum dryers, it will have been noticed that no heat transmitted to such dryers has any means of escaping unused except by radiation from the outer shells

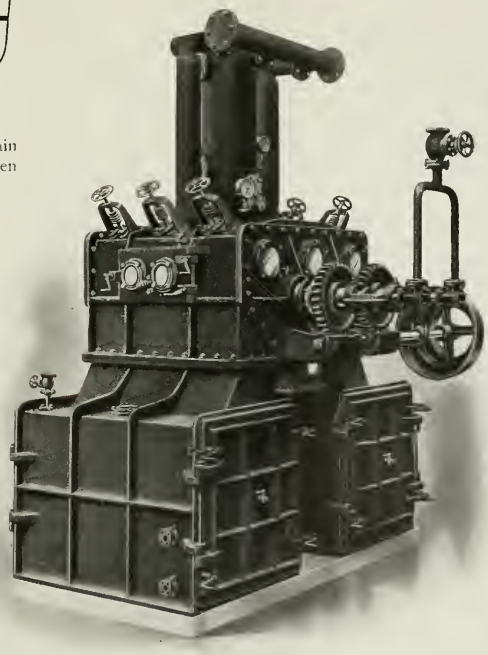


FIG. 56.—ASSEMBLED PASSBURG VACUUM DOUBLE DRUM DRYER.

of the drying apparatus, and this can be minimized by an efficient coating of a good non-conductor of heat.

All other heat imparted by means of the heating medium, be it either steam or hot water, is absorbed by the water contained in the drying material for its own evaporation. There is no other outlet for any unused heat, as in the case of drying

under atmospheric conditions, where, as stated before, it is absolutely necessary to frequently exchange the air in the drying room, and in many cases heated air, in order to allow at least a fairly rapid evaporation.

As soon as the air in the drying room, be it hot or cold, has charged itself with a certain amount of moisture; that is, all that it can carry, it must be removed, as otherwise it will

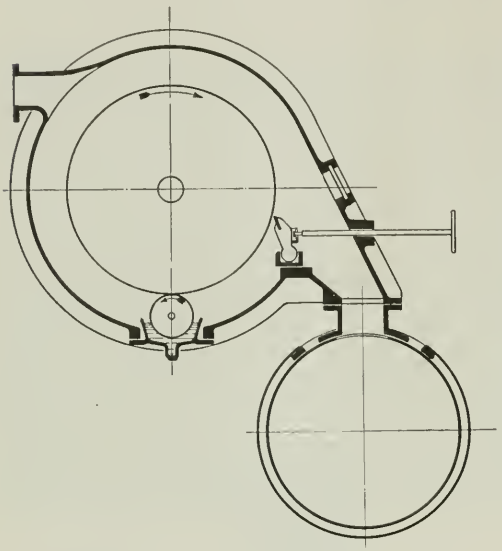


FIG. 57.—PASSBURG SINGLE VACUUM DRUM DRYER WITH ROLLER FEED EQUIPMENT.

directly stop a further evaporation of moisture from the material. This steam-saturated air, if remaining stagnant, forms a kind of fog or moisture-laden atmosphere in the drying room. A quick exchange of air is obtained by means of fans or similar apparatus, and the drying process is thereby aided.

With heated air an enormous quantity of heat is carried away by such air, runs to waste, and leaves the drying room unused.

The B. F. Sturtevant Company of Boston, Mass., states in one of its catalogs that even under favorable circumstances, where some of the heat was saved for reuse, the loss of heat by means of hot-air drying was as follows:

	Percent of total heat.
Required to vaporize moisture in stock	32
Lost by leakage, radiation from tower, and unaccounted for	37
Lost by radiation from return duct and by introduction of fresh air at fan room	31
Total heat imparted to air	100

The above data shows that only about one-third of the heat imparted to the drying air was really utilized for removing the moisture contained in the material. As this statement was made by a firm who are certainly experts and specialists in building hot-air drying systems, it may be considered as a fair statement, and certainly does not underrate the value of such drying systems. It gives an impartial proof of the unavoidable loss which is connected with any drying system where an air current is used.

Where materials of a hydroscopic nature have to be dried, absolute dryness can, of course, never be obtained by means of hot air, as naturally the drying material can never be dried below the moisture contained in the hot air used as a heating medium, and with materials which are of an inflammable nature at a certain temperature, hot-air dryers show great disadvantages, as they can easily produce an explosion, while with materials which easily oxidize the continuous heated air current has also naturally a detrimental effect.

Another important advantage of drying under vacuum is that the evaporation takes place in the absence of atmospheric air with all its impurities. Under such a rapid exchange as would be necessary when evaporating under atmospheric conditions, the air would partly deposit such impurities and thereby add to the quick deterioration of perishable materials, which deterioration may even continue in the dry material and thereby nullify the results of the drying or concentrating process.

Where discoloration is feared from the deposit of dust, the drying of such materials has no difficulty at all. When white lead, for instance, is dried by hot air under climatic conditions where the atmosphere is heavily laden with moisture, it is impossible to obtain either a dry product or a product with an even moisture content.

However, in a vacuum dryer the dry white lead has an even moisture content of .25 per cent, is free from a coating of dust or dirt, and has a white color.

Milk or egg powder, produced on the single or double-drum dryers, have proven to be practically free from obnoxious bacteria, lactic acid and overheated butter-fat, which latter tends to give the dry product a rancid taste.

On account of the low temperature employed during the rapid evaporation of the water in these liquids, which are partly emulsions and partly solutions, the albumen remained soluble also, and this is the most important and practically indispensable factor.

The result of the drying process is not only an increased keeping capacity of the dry products, as no germs are prevalent to start decomposition in the absence of water, but also the flavor of the re-dissolved powders could scarcely be distinguished from the fresh material.

This drying apparatus and process will consequently obviate the cold-storage system with all its recently much-discussed drawbacks and give the consumer a cheaply preserved and wholesome article of food at a reduced cost in all seasons of the year.

The old slow-drying method, which was only possible during the summer months when the atmosphere was not too heavily charged with moisture, has caused great loss to the manufacturers of glue; and even now, where a more rapid drying system by means of hot air and fans is frequently used, the gelatinized glue cakes either become mildewy and spoiled through the long drying process or the gelatinized glue becomes liquid again and flows through the net on which they were put for drying.

The losses have been obviated by the adoption of the vacuum drum dryers, which produce, quite independent of climatic conditions, a glue powder of even moisture contents, solubility and color.

I have briefly touched upon some of the various devices and types of apparatus used in evaporation and for the final removal of the last traces of moisture from materials which are desired to be thoroughly and perfectly dry in their final state.

The principles involved in their operation are many and complex. The removal of moisture from given materials is a scientific problem.

That its elimination cannot be successfully accomplished by "rule of thumb" methods is becoming recognized by those engaged in the manufacture of products in which drying is an important factor.

Brooklyn, N. Y.

The Application of the Turbo Blower in the Iron and Steel Industries.*

By J. E. Johnson, Jr.

I have been very much interested in the subject of this turbo blower since it was in its initial stages in the shops of the General Electric Company, the subject of blowing engines being one of profound importance to the blast-furnace manager. Men of this calling are commonly not members of the American Society of Mechanical Engineers and I trust that some discussion by one of the few who are will not be amiss.

The efficacy of the control of the volume of air blown is the question which will arise in the mind of every practical furnaceman as soon as he gives this subject consideration. Changes in quantity of air as small as 1 per cent or 2 per cent are frequently made in the quantity of blast delivered to the furnace as one of the means of controlling its operation, and this makes it obvious that exact control is a very necessary feature of any blast-furnace blowing apparatus.

I had the pleasure of seeing the installation of the turbo blower at the Port Henry plant of the Northern Iron Company about a year ago, and the simplicity of the governor is certainly admirable, but is a feature not exactly easy of comprehension to a man without thorough mechanical training, and a prejudice against these machines is likely to exist in the minds of many furnacemen on this account, which will make their introduction slower than it perhaps otherwise might be.

It is a well accepted fact that the efficiency of the turbine is far higher at low pressures than it is at high, while, on the other hand, it is equally well known that for high pressures only the reciprocating engine has a considerable advantage over the turbine. The result is that the most economical form of steam-driven apparatus is a combined unit in which the steam expands down to about atmospheric pressure in a good reciprocating engine and is then delivered to a turbine on its way to the condensers.

The very remarkable results obtained with this combination by Mr. Scott at the Interborough plant in New York have, in the opinion of many, given a decided check to the development of the gas engine.

In a similar way it has been known for years that for very low pressures the centrifugal blower, of which the turbo is simply a development, was more economical than a piston blower.

On the other hand, for higher pressures the piston compressor is undoubtedly to be preferred, particularly in blast-furnace work, on account of its operating as a meter as well as having much the highest efficiency as a compressor.

A condition of piston-blowing engine design is that the air cylinder must be large enough for the greatest volume to be handled and strong enough for the highest pressures attained, with the result of enormous size and very massive and expensive construction in modern blowing machinery.

In spite of their high cost it is notorious among operating men that really reliable blowing engines are exceedingly scarce. The problem of the valves is in itself one of great difficulty. It is almost impossible to get an air-inlet valve with sufficient area to allow the cylinder to fill absolutely without heavy loss by suction, and at the same time quick enough to give correct results at the speeds required of modern engines.

The turbo blower, on the other hand, suffers from the disability that it must have stages enough to blow the highest pressure ever required, although this high pressure will not, in ordinary operation, be required 1 per cent of the time, the normal pressure being perhaps only half as great.

If, therefore, we make a turbo blower to deliver air of a certain nearly constant, moderate pressure, and a blowing cylinder of comparatively small size, strong enough to deliver air to the desired maximum pressure we shall have bettered the conditions of operation of both apparatus, increasing their efficiency and reducing their cost. In other words, the turbo blower and reciprocating compressor combined are more efficient than either apparatus alone, exactly as the engine-turbine combination is more economical in the consumption of steam.

For the best and cheapest blowing engine, therefore, we should have a combination of these two units, a turbine-driven turbo blower, delivering air partly compressed to the compressing end of a steam-driven blowing engine, the steam from the cylinder of this engine driving the turbine before passing to the condenser.

It may be objected that this is a complicated unit, but this is not the case. Most large modern furnaces require three of the best types of large reciprocating blowing engines to operate them. These are, in good modern usage, disconnected compounds—a high-pressure steam cylinder on one engine, a low-pressure on the next, etc.

A turbo-blower supplying air compressed to 8 lb. to two of these would enable them to deliver as much air as three drawing air from the atmosphere, and the efficiencies which are guaranteed in the use of exhaust steam by the builders, combined with the guaranteed efficiency of the turbo, indicate that this can be done at a decided saving in first cost and with an efficiency certainly no lower, probably higher, than that of either unit alone. This does not make any allowance for the possibilities of spray inter-cooling between the turbo and the reciprocating compressors, which, in ordinary practice, would be good for about 4 per cent additional saving at a very low cost.

The advantage of maintenance is undoubtedly with the turbo, as compared with reciprocating engines. At the same time the magnitude of the alternations of stress in the reciprocating engines is greatly reduced by delivering to them partly compressed air, and the problem presented by their inlet-valve gear, which has given so many bad hours to both furnacemen and engine designers, is practically eliminated by delivering to them denser air under pressure.

It, of course, cannot be denied for a moment that the efficiency of the reciprocating machine as a compressor is far higher than that of the turbo, except for the lowest pressures, while the advantages of positive metering of the air by the reciprocating engine will cause furnacemen to accept the combined apparatus without that hesitation which they feel toward the non-positive turbo.

The governing of such a combined unit is extremely simple. The ordinary governor on the engine unit does all that is necessary and no governor other than one to prevent racing is necessary on the turbine. The steam supply of the engine passes directly to the turbine. As long as all conditions remain constant the speed of the turbine will be unchanged, but if the pressure required by the furnace increases a little, more steam is admitted to the steam engine by its governor, and this increased quantity of steam speeds up the turbine to a slight extent and so causes it to deliver air compressed to a slightly higher pressure. This slight increase automatically compensates for any slight lag due to the greater load on the engine and for the lower volumetric efficiency of the piston-blowing engine at higher pressures as compared with low.

If, on the other hand, the speed of the piston engine is deliberately increased to deliver more wind, the increased supply of steam to the turbine will enable it to deliver a proportionately increased quantity of air at virtually the same pressure. Of course, some hand control will be applied to the turbine to enable the receiver pressures to be adjusted to the best conditions, but, once this is done, other changes will seldom be required.

For these reasons I believe that the steam-driven blowing plant of the future should consist of a turbine-turbo, piston-engine, piston-compressor combination.

*A discussion of the paper of Mr. Richard H. Rice, presented at the recent convention of the American Society of Mechanical Engineers.

This combination will have its greatest field in applications to existing plants where, as is so often the case, the existing blowing engine, while old, is by no means worthy of the scrap heap, but is unable to deliver quite the pressure or quite the volume demanded by modern conditions. In such a case the introduction of a turbine-turbo, taking steam from the engine and supplying precompressed air to it, will use what is now waste energy in the steam and apply it in such a way as to increase both the capacity and the permissible pressure of the existing blowing cylinder.

A further advantage which the turbo has as the primary machine is that of an inlet free from pulsations. These are much more serious in inlet than in outlet, for the reason that the volume and velocity of the air are so much greater.

One of the serious problems which had to be overcome in the application of the dry blast was the construction of cooling chambers whose walls would not be thrown down by the pulsations produced by the piston compressor on the incoming air. The smooth inlet of the turbo will eliminate entirely difficulties arising from this source.

The necessity of massiveness both in stationary and in reciprocating parts, and the conditions of operating the air valves, have operated to limit the best size of blowing engines, so that these are not only not increasing in size, but those now built are commonly as large as some that were built ten or twelve years ago.

The turbo-blower, on the other hand, lends itself admirably to enormous capacities in the same way that the turbine is commonly built for certain classes of service in sizes that have never been approached in reciprocating engines. For this reason it will not only be possible, but advisable, at large plants to install one turbo to supply several reciprocating engines, all of which, of course, will exhaust to it. This will contribute still more to low first cost, small space requirements and small maintenance charges.

I wish to express in conclusion my conviction that the introduction of the turbo compressor marks the beginning of a better era for the mechanical equipment of blast furnaces, but that its best and easiest application will be in conjunction with rather than in supplanting piston-blowing engines.

Ashland, Wis.

The Vacuum Filter Process in Western Australia.*

By T. B. Stevens and W. R. Degenhardt.

The stationary-frame vacuum-filter process was introduced into Western Australia four years ago, and has since been developed and improved until it now forms one of the chief methods of slime treatment on these fields.

The object of the present article is to point out, more particularly, the novel features of the plants in operation in this State.

Nature of Material Treated.—The material treated in vacuum-filter plants varies from the finest oxidized slime to an all-slimed product from sulphide ore. No slime of any grade has yet been tried that could not be economically handled.

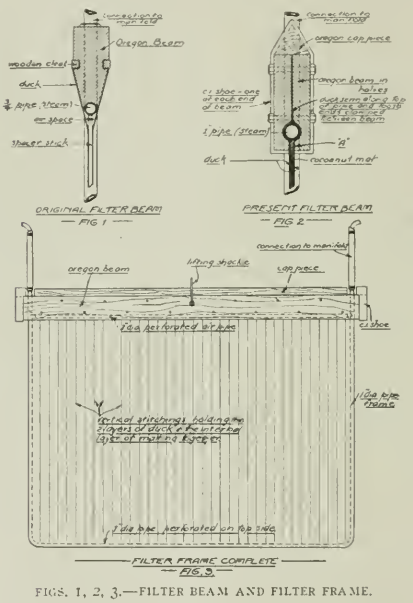
The worst filtering slime yet tested was an oxidized clay containing no fine sand, obtained from the Never Never mine (near Southern Cross). This slime gave a cake only ⅜ in. thick, containing 3.46 lb. of dry slime per square foot after 40 minutes of vacuum. It could be successfully washed and discharged. Average oxidized slime will form in 25 minutes a cake 1¼ in. thick, carrying 8 lb. of dry slime per square foot. An all-slimed sulphide product will, in the same time, form a cake 2½ in. thick, carrying 18 lb. of dry slime per square foot.

It is always advisable to make experiments with a single filter frame before a plant is erected. The appearance of slime as regards its cake-forming power is often deceptive; thus, Rand slime is considered a perfect slime free from sand and is practically a pure clay, but with this slime a 3-in. cake can be readily formed. The slime that contains most sand is not always the most easily formed into a cake. It appears that a well-sized slime, i.e., one containing particles of approximately equal sizes, best forms a cake, the reason being that the interstitial space through which the solution can pass is greater than in a mass of closely packed particles of variable sizes. This subject has been carefully studied in connection with sand leaching.[1]

The cakes from oxidized slime will, as a rule, contain 30 per cent to 33 per cent of moisture, while an all-slimed sulphide product will contain from 23 per cent to 25 per cent.

The Filter Frames.—The present filter frame is the result of considerable experimental work. The trouble with all the earlier frames was that the cakes cracked on exposure and consequently fell off. Practically all frames are now built of 1-in. steam pipe and are covered with 16-oz. duck, with cocoa-mat between.

The filter cloths are sewed with vertical seams 3 in. apart, this spacing having been found to be the best. Undoubtedly a



FIGS. 1, 2, 3.—FILTER BEAM AND FILTER FRAME.

great part of the filtered solution passes through the stitch holes instead of the cloth, and it has been found that bags with seams 6 in. apart do not take cake so quickly as those with 3-in. spacing. Bags with seams only 1 in. apart have been tried, but these did not form the cake so quickly as the 3-in., the reason being that the close stitching drew the cloth so tightly to and compressed the cocoa-mat that very little space was left in the mat for the solution to pass through.

The weight of the pipe, frame and cake is hung from two supporting beams of special section. The detail of these beams is important. The beam at first used (Fig. 1) had many disadvantages. The wooden cleats were not strong enough to hold the cloth and the weight of cake. The cloth also had no support where it joined the beam, thus causing cracks to form there on exposure. It is also obvious that if the cloth were allowed to stretch the cake would crack at the top.

Another difficulty was that the cake formed on the cloth where it was stretched over the beam and could not after-

*Slightly abstracted from the *Journal* Chamber of Mines of Western Australia.

[1] See "Cyaniding of Gold and Silver Ores." Julian and Smart; 1st edition; p. 49.

wards be dislodged by the air discharge, thus necessitating constant hand-cleanings. The air pressure also caused rapid wear of the cloths, for when the air was turned on a great strain was put on the vertical seams at the top of the bag, and it was only a matter of a few charges before holes appeared, allowing the slime to leak into the filtered solutions.

All these disadvantages have been overcome in the improved frame. (Fig. 2.) In this frame the support of the cloth is

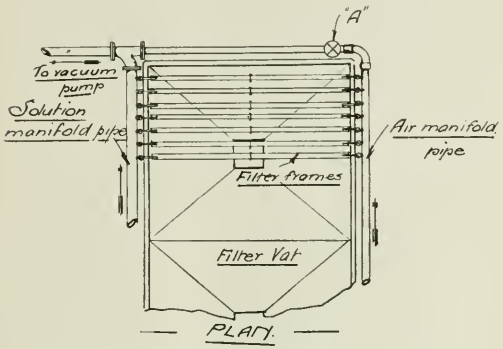


FIG. 4.—MODIFIED METHOD.

satisfactory, as it is clamped securely in the split beam. No cloth is exposed on the outside of the beam, consequently no cake can form on it. The shape of the edge *A* is designed so that it just presses on the cloth, but not so tightly as to prevent the passage of compressed air. The vertical seams come just up to this ledge, and it is, therefore, impossible for them to rip.

Frames are usually made of a uniform size (Fig. 3) of 9 ft. wide by 4 ft. 9 in. deep; this permits the use of a covering of duck 72 in. wide. The capacity of a plant is adjusted not by the size, but by the number of frames employed. Taking into account the handling of the frames for renewals, and the time occupied in filling and discharging the filter vats, this size of frame has been proved, from practical experience, to be the most efficient; deeper frames take considerably longer to discharge owing to the choking of the cakes.

If a plant is working on slime that will form a cake less than 1½ in. thick piping of smaller size than 1 in. may be used for building the frames, but with cakes above this weight nothing less than 1-in. steam pipe will stand the shock of the falling cakes. Pipes are liable to break where they join the beam, owing to the leverage exerted on them when the falling cakes spread the frames at the bottom. It is advisable not to fix the frames in the filter vat too securely, as a slight motion of the beams decreases the shock to the frames.

Various devices have been tried for obtaining the necessary spaces for the flow of the solution between the cloths of the frame. Wooden laths were first used, one being placed between each 3-in. seam; these were fairly satisfactory, but they did not give the support to the cloth that the cocoa-mat does and the cakes are, therefore, more liable to crack during exposure. Ripple iron, i.e., galvanized iron with small corrugations, as used at the Waihi mine, New Zealand, was also tried, but as this had to be used without vertical seams trouble was experienced in discharging the cakes. When the air was turned on the cake was stretched but not discharged.

With very free-filtering slime, i.e., slime that will form a cake of 2 in. or more in 20 minutes, it has been found that the cakes form thicker at the bottom than the top. The reason for this has been traced to the fact that the filter mat is not sufficiently porous to allow the solution to be drawn off as quickly as it passes through the cloth. This difficulty has been obviated by using the top pipe of the frame, as well as the bottom one, for drawing off the solution. Fig. 4 illustrates this method, which may be described briefly as follows: In taking cake or washing, both the solution and air cocks on the frames, as well as the valve *A*, are left open, thus enabling each cock to draw from approximately half a frame. When the cakes are ready to discharge valve *A* and the air valves on the frames are closed; compressed air is then admitted to the other end of the air manifold and the air cocks are opened one at a time to discharge the cakes.

The same result could be obtained by using larger pipes for the frames and a larger solution manifold, but this would add needless weight to the frames. This arrangement is also useful as an addition to an old plant, where, by reason of the internal deposition of lime, the pipes become too small for the flow of solution.

Filter Vats.—The chief point to be considered in the design of filter vats is the method of discharge. It was obvious from the beginning that in Australia sufficient water could not be obtained to discharge them by sluicing with water pressure as practised in America. It was required to discharge the residue into the residue dam in the form of a pulp as thick as could be handled. This has been effected by mixing devices of different designs placed in the bottom of the filter vats.

Fig. 5 shows the most successful type, in which the cakes are dumped through discharge doors into a separate mixing agitator below. This allows of very quick discharge—less than 10 minutes—as the mixing can be completed after the discharge doors are shut and the filter vat is at work on the next charge. This type is always adopted where water from the Goldfields Water Supply is used for sluicing purposes, since water used exclusively for sluicing and not entering the treatment plant is supplied at a cheaper rate than that used for general purposes.[2]

Filling Filter Vats.—It is essential that the filter vats should be filled and emptied quickly. Two methods have been employed:

1. Filling by a small pump and gravitation. In this arrangement the pulp and wash are elevated to storage vats by small continuously running pumps, and the filter vats are rapidly filled from these through large pipes, as required. The excess pulp and wash from the filter vats is run out into two storage vats situated below and then elevated to the upper vats.

2. Filling by a large pump running intermittently. In this arrangement only one set of storage vats is required. These are placed on the same level as the filter vat, from and to which the pulp and wash are pumped as required.

Both methods have their advantages. The capital cost in each case is much the same, for, while with the gravitation plant extra storage vats are necessary, the pumping plant requires

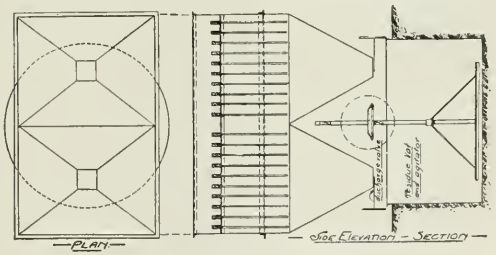


FIG. 5.—FILTER VAT.

a more powerful pump and larger piping and valves. As regards power cost, there appears to be little difference, since the gravitation plant requires power for running two small pumps continuously on a high lift, and for stirring two extra storage agitators, whereas the pumping plant requires only sufficient power to drive a large pump on a low lift.

[2]For general water services the price is 7s per 1000 gal. and for sluicing water is 6d per 1000 gal.

For a mine with a large power plant or one buying power from a central generating station, the sudden addition of a big load, such as a large pump, is of little account, but for a mine with a small power plant of its own the smaller but continuous load is the more convenient. As a rule, it has been found that where only one filter vat is required the gravitation filling is good, but where two or more filter vats are necessary the pumping plant is undoubtedly better.

Pumps.—For a high lift triplex plunger pumps are most efficient, and these call for no remarks. For a large capacity on a low lift the centrifugal pump is the best; the pumps most in use are of the Kershaw type. The casings of these pumps are made in halves with internal hard liners, which, with other working parts, are easily renewable. The cast-steel ported runner is fitted with cast-steel renewable shoes and is carried on a heavy mild-steel shaft. The shaft is run in a specially sealed four-part bearing with wedge-adjustable brasses, inside wearing ring and nipple. A stuffing box and gland for greasy packing are also provided. The runner shaft is connected by a muff coupling to a short shaft carrying the fast and loose pulleys between two outer standards having heavily flanged gunmetal bearings and loose thrust collars. All bearings are provided with water-circulating pipes and cocks to supply clean water under pressure to keep the bearings free from sand and grit. Stauffer lubricators are also fitted. The loose pulley is brass-bushed and fitted with a Stauffer lubricator; it is run on a cast-iron sleeve, which is secured to the spindle and which is the full width of the pulley face.

Wet-vacuum pumps without intermediate receiver have been found quite satisfactory and capable of maintaining a vacuum up to 27 in. These pumps are usually of 8½ in. diameter, single cylinder, 10-in. stroke, double-acting up to 40 r.p.m., according to the number of frames to which they are connected. They are fitted with vacuum and air vessels to promote uniform action. The pistons are of the trunk type, ordinary square packing being used.

Valves.—For filling and emptying the filter vats 6-in., 8-in. and 10-in. pipes are used. Where it is possible to do so quick-opening valves are arranged so that they may be operated from the working platform by means of extension spindles and hand-wheels.

In a plant lately designed 12-in. valves are proposed to be pneumatically operated. This improvement is suggested for the reason that most of the work in operating vacuum slime-plants consists of opening and shutting valves. Considerable power is required to open and shut a large valve, and even with a treble thread the manual operation is slow. The pneumatically operated valves act at once and will thus effect an appreciable economy of time and labor.

Clarifying Solutions.—In most cases it has been possible to make use of old filter presses for clarifying solutions; the solutions are, as a rule, very clear and a by-pass is arranged so that in the case of a leaky bag the delivery of the vacuum pump can be turned back into the filter vat. Where filter presses have not been available, filter-bag frames, placed in an elevated gold solution vat, form an inexpensive and effective clarifier. The frames are similar to those used in the filter vats, but are long and narrow. The suction is obtained by the fall to the zinc-boxes. The frames are usually cleaned once in each shift by closing the delivery to the extractor boxes and inflating the bags with air; this causes any deposited slime (which is very tough) to peel off into the bottom of the vat. After the frames have been cleaned the delivery is kept shut off as long as possible to allow the slime to settle. The settled slime in the bottom of the vat is sluiced out once a week into one of the treatment agitators.

Washing of the Cakes.—The time of economical washing varies greatly with the nature of the slime, and can only be definitely determined by taking progress assays from the discharge of the vacuum pump. This subject has been already discussed in an earlier number of the *Journal*. The time occupied in washing is always greater than that required for taking cake. It has been found that a low-grade slime (under 8s. per ton) obtains a good wash when the amount of solution sent to the extractor boxes is equal to about one-tenth more than that contained in the original pulp. As in all other filter processes, it is essential that the wash should be kept low in gold; to do this the filter vat must be quite free from pulp when the wash is put in and this can be easily effected by spraying down the sides of the vat with spent solution as the pulp is emptied out. If this be done, the wash will not gain more in gold by contact with the vat and cakes than can be counteracted by dilution with the make-up of barren solution and fresh water.

Notes on the Metallurgy of Wrought Copper.

By F. Johnson, M.Sc.

Metallic copper is unique among all the known metals by reason of its malleability, ductility, electrical and themal conductivity, strength, toughness, comparative resistance to corrosion and immunity from oxidation, strength in torsion, capability of taking a high polish, high melting point, beautiful luster when chemically treated, ability to be worked into any conceivable shape, etc.

Its property of forging well at a red heat is one which enormously enhances its usefulness.

By reason of its great malleability, in which property it stands third in the table of metals, it can be hammered or rolled into sheets which have an immense range of application in the arts and in the engineering industries .

Its ductility enables it to be rolled and drawn into the finest wire, its possession of this property ranking it high in the order of ductile metals.

In electrical conductivity it is second only to silver, and this property alone has accounted for the production of millions of tons of the metal.

Its resistance to corrosion has enabled it to successfully withstand the advance of all competitors, as a material for such purposes as sheathing ships' bottoms or lining pump cylinders, while it also enables it to be used for culinary articles, chemical and distilling apparatus, boilers, etc.

Its immunity from oxidation is of great service wherever it is used, and this property, where nails, bolts or screws of the metal are used in contact with wood, gives it a peculiar advantage over iron. Unlike rust, what superficial oxide is formed has no corroding influence on the wood into which the metal enters.

Its thermal conductivity places it supreme among all competitors for use as firebox plates, stays and tubes of locomotive boilers, where its economical advantages over iron and steel are in the direction of greater durability and evaporative power.

Wherever heating surfaces are employed copper is pre-eminently desirable, its thermal conductivity being two and one-half times greater than that of iron. Steam, feed-water and many other pipes are also best made of copper.

Its toughness, tensile strength and torsional strength enable it to withstand all the stresses which it is ordinarily called upon or expected to encounter, e.g., telegraph and telephone wires and overhead electric trolley wires, etc. Any means of improving these properties, however, will always be acceptable.

Its luster, color and applicability to various shapes render it a favorite metal with art metal-workers; chasing, embossing and polishing being operations of ease to the skilled workman.

It is not proposed to say anything of the use of copper for alloying purposes, except that in all alloys which have to be rolled or drawn freedom from arsenic, lead and antimony is desirable. Copper is used for alloying with tin, zinc, aluminium and nickel chiefly, the number of variations of its alloys with these and other metals being almost infinite and their name legion.

The casting of copper, whether in the form of ingots for alloying, cakes for rolling into sheets, bars for drawing into wire, or billets for forging and drawing into tubes, etc., is an extremely important and delicate operation owing to the property which molten copper possesses of dissolving its own oxide. The elimination, in part, of oxygen is the keynote of copper refining, just as the regulation of the quantity of carbon in iron is the keynote of steel making. Copper containing too much oxygen is cold-short, and, if oxygen is in great excess of the percentage contained in an alloy consisting entirely of the copper-cuprous oxide eutectic, the metal is also hot-short. If oxygen is removed to too great an extent, the metal then has greater solvent power for reducing gases, which it is incapable of retaining entirely during solidification, the expulsion of these gases causing the surface of an ingot of the metal to rise in the mold, thus showing a central ridge, or perhaps a worm-like excrescence, such a phenomenon being popularly known as "spewing" or "the worm coming out." (See Fig. 4.)

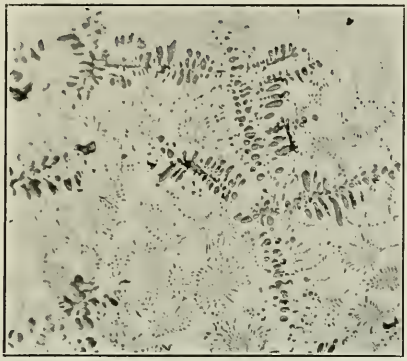


FIG. 1.—STRUCTURE OF OXIDIZED COPPER BEFORE POLING, SHOWING DENDRITES OF CUPROUS OXIDE IN A MATRIX OF COPPER CUPROUS OXIDE EUTECTIC. MAGNIFICATION 100 DIAMETERS.

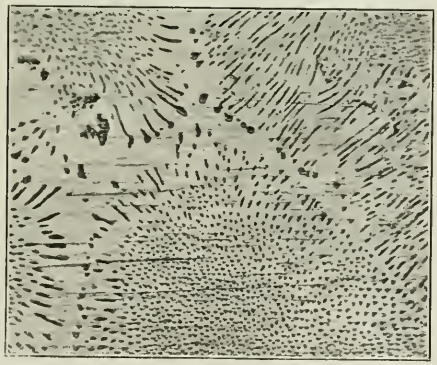


FIG. 2.—STRUCTURE OF COPPER AFTER ELIMINATION OF SOME OXYGEN BY POLING. SHOWS COPPER CUPROUS OXIDE EUTECTIC (0.47% OXYGEN). MAGNIFICATION 300 DIAMETERS.

All the phenomena occur with all kinds of copper when refined on a large scale, even with the purest electrolytic copper. When the oxygen is in excess the surface of an ingot shows a depression and it is said to be "under-pitch." When it shows an elevation, or "spews," it is "over-pitch"; when it exhibits a level surface it is said to be "tough-pitch," and in this condition the combination of its physical (including electrical) and mechanical properties has reached its highest possible degree (for "refined" copper).

The foregoing are the broad general principles of refining, but every refiner of impure copper knows that it is possible to pole up charges of certain brands until an ingot will show a level surface, whereas the copper will be "red-short," and only by stopping poling at a stage where an ingot shows a depression can such copper be rendered fit for rolling or forging hot.

Micrographs 1, 2 and 3 show the structure of copper during the process of poling, illustrating clearly the reduction of cuprous oxide as poling proceeds. The cuprous oxide is completely soluble in the molten copper, but, as shown in the micrographs, is insoluble in the solid metal, separating out as a definite constituent, Cu_2O, which is light blue in color and is easily distinguished from cuprous sulphide by the property which it possesses of etching black when treated with hydrofluoric acid. Cuprous sulphide remains unchanged when similarly treated.

Where the Cu_2O exceeds 3.5 per cent to 4 per cent dendrites of Cu_2O make their appearance, as shown by the firtree-like constituent in No. 1. This is the first constituent to separate out during cooling of the liquid solution and its presence increases the viscosity of the copper before and during solidification, causing the copper to pour thickly. The other constituent which forms the matrix is the eutectic alloy comprising an intimate mixture of minute copper and cuprous oxide crystals, or, as they may be conveniently called, "oxidules."

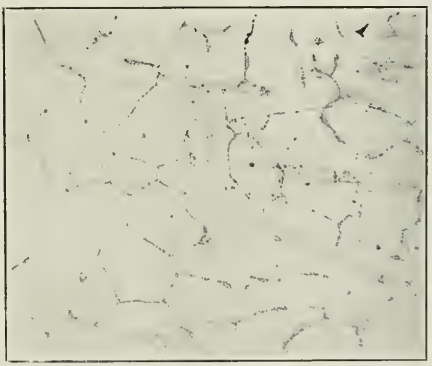


FIG. 3.—STRUCTURE OF TOUGH-PITCH COPPER, SHOWING CRYSTALS OF COPPER SURROUNDED BY EUTECTIC. MAGNIFICATION 300 DIAMETERS (0.08% OXYGEN).

This eutectic structure is clearly seen in No. 2; its melting point and composition, according to Heyn,[1] are respectively 1084° C. and 3.5 per cent Cu_2O (see Fig. 5).

Fig. 3 shows the structure of "tough-pitch" copper, where, instead of the one constituent as in No. 2, there are two, viz., first, copper crystals which have separated out from the liquid solution, leaving the mother liquor (solution of cuprous oxide in copper) to feed them with copper until it reaches the eutectic composition, and, second, the solid eutectic itself.

Some interesting observations in connection with the properties of copper at various stages of the "pitch" have been made by Messrs. Hofman, Hayden and Hallowell in a paper read before the American Institute of Mining Engineers (Vol. XXXVIII). They prove that the gradual elimination of oxygen is synchronous with an improvement in electrical conductivity, but, strangely enough, the conductivity reaches a maximum while the copper is still decidedly "under-pitch" (0.2 per cent oxygen). They also prove that those traces of iron and sulphur which remain in "set" copper, after the oxidizing stage

[1] Mittheilungen aus den Königlichen Versuchsanstalten zu Berlin, 1910, XVIII, p. 315.

of refining, are not affected by the subsequent poling operation.

In a recent paper[2] the writer advanced a theory to account for the behavior of copper as influenced by the process of poling.

It must be remembered that when poling starts the bath of molten copper is very nearly saturated with cuprous oxide and

FIG. 4.—INGOT OF OVERPOLED COPPER, SHOWING "WORM."

that the effect of thrusting a pole of wood into such a bath is the rapid reduction of the oxide accompanied by the evolution of gaseous oxides of carbon, hydrogen, water-vapor, etc.

As the poling continues and the proportion of metallic copper increases, the latter has the property of dissolving some of the evolved gases, which it was formerly incapable of dissolving when saturated with oxide.

As more and more copper is thrown out of combination with oxygen, so more and more gases are dissolved. The gas for which copper has the greatest affinity under these conditions is hydrogen, which is probably derived from the decomposition by molten copper of the hydrocarbons in the pole, free carbon being precipitated and rising to the surface.

Trial ingots taken from time to time will show a depression at the surface and are said to be "under-pitch." This depression, however, becomes smaller and smaller until, when the charge is up to "pitch," it disappears and the surface of a trial ingot is quite level. If the poling be pushed beyond this point the surface will rise and form a convexity.

If sample ingots of (1) "under-poled," (2) "tough-poled" and (3) "over-poled" copper be broken and their fractures examined it will be seen in case (1) that the fracture is "dry,"

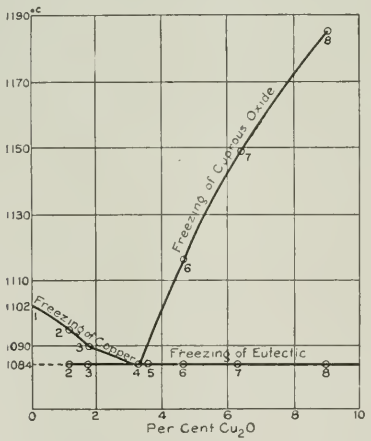


FIG. 5.—FREEZING-POINT CURVE OF ALLOYS OF COPPER AND CUPROUS OXIDE (HEYN).

with no silkiness of grain and no vesicular cavities; in case (2) the fracture is silky, with tiny vesicular cavities, uniform in size and distribution; in case (3) the fracture is still silky but there are more cavities, some of them larger than others, the larger ones occurring nearer the surface.

These cavities were evidently filled with gas, released at the moment of solidification; the bubbles of gas, by reason of their natural buoyancy, exerted an upthrust which tended to oppose the effect of liquid shrinkage and so prevented the surface from being depressed.

In the case of "under-poled" copper they were not sufficiently numerous to oppose this effect, while in the case of "over-poled" copper they occurred in such quantities as to not only oppose the formation of a depression, but to elevate the surface and in some cases to cause some of the semi-molten copper to be projected from the ingot.

In the case of "tough-poled" copper the bubbles of gas were present in just sufficient quantity to oppose the formation of a depression and no more.

Thus all ingots of comparatively pure copper, when poured at the proper "pitch," show a level surface.

The Effect of Mechanical Work on Copper.

Copper, after hot-forging, is stronger, more ductile and of closer grain than the cast metal. Hot-rolling can be carried to such an extent as to confer on the copper a tenacity which the same copper, after severe cold-working and annealing, cannot exceed. But hot-working should be stopped at such a temperature as will allow a certain amount of molecular rearrangement or recrystallization to take place following on

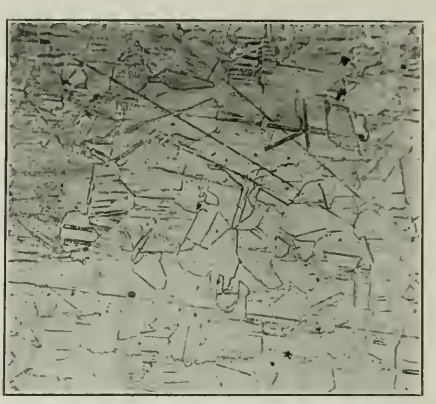


FIG. 6.—CAST COPPER DEOXIDIZED WITH MANGANESE, SHOWING "TWINNING." MAGNIFICATION 100 DIAMETERS.

distortion. That is to say, hot-wrought copper should have the closest and finest crystalline structure attainable short of the confused structure by working the metal at too low a temperature.

Subsequent cold-working may then be possible to a greater extent before annealing will be necessary. Cold-working, stage by stage, continuously strengthens, i.e., increases the tensile strength and limit of elasticity, but diminishes the extensibility. Le Chatelier states that pure copper can attain to a tenacity of 27 tons per square inch when drawn into wire, but this cannot be exceeded even by repeated drawing. Annealing has the effect of lowering the tensile strength and raising the elongation.

These figures (for cold-wrought and annealed copper) have a limit, and the standard to take is that where maximum strength and ductility are just obtained in cold-wrought specimens which have been annealed for a known length of time at a known temperature in a non-reducing atmosphere.

Cold work produces a state of instability which may render the metal susceptible to "age-cracking" or secular brittleness.

There are two clever theories with regard to the cause of this state of instability. Firstly, however, let it be understood that the cold-working must be of sufficient severity to stress the metal beyond its elastic limit in order to produce deformation of a serious nature.

[2]Inst. of Metals, No. 2 (1910), Vol. IV.

Beilby is responsible for the theory that those layers of the soft crystals which are contiguous to their boundaries assume a different condition—a hard, amorphous condition which confers on the metal, by virtue of its enveloping inter-cellular hardness, greater elasticity and tensile strength, greater brittleness and reduced toughness.

In short, cold work distorts the crystals, causing gliding of

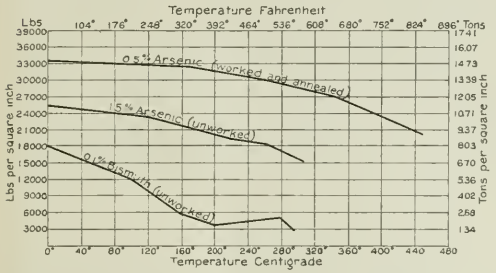


FIG. 7.—TENSILE STRENGTH AS FUNCTION OF TEMPERATURE.

crystalline elements over one another, along those planes where there is least cohesion; then appears the hard, non-crystalline variety which, thanks to its instability at a definite temperature, reverts back to the soft crystalline state once more when the annealing temperature is reached.

To Ewing and Rosenhain we are indebted for the theory of "slip-bands," and it will best serve the purpose of the writer to quote from an article written by Mr. Rosenhain for the *Times Engineering Supplement* of Nov. 6, 1907:

"It has been found that, when the application of force causes a crystal to change its shape, it accommodates itself to the new shape imposed upon it, by a process of sliding or slipping which takes place between the layers of minute crystalline elements of which each crystal is built up. After deformation the crystal remains a crystal as before, but the layers of its elements have been displaced relatively to each other. Further deformation results in further slipping, until, ultimately, the surfaces upon which slip can occur—originally plane in the undeformed crystal—have become converted into complicated systems of minute steps. If the attempt be made to push the deformation still further, then at some point or other of the cross-section of the metal two adjacent layers will slip far enough over each other to become entirely separated, and the fracture thus started will run through the metal and will follow all the minute steps of the surfaces upon which slip was at that moment taking place.

"The surface of the resulting fracture will thus be minutely stepped, and, in the case of a tensile fracture, the surfaces of these steps will all be sharply inclined to the direction of the extension. According to the degree of ductility of the metal and the scale of its original crystalline structure, the resulting fracture will show a more or less minutely stepped surface, giving the appearance of a fibrous fracture. That fractures so produced actually possess this stepped character can be clearly seen in properly prepared sections examined under high magnifications."

Mr. Rosenhain explained this theory somewhat fully at the second meeting of the Institute of Metals in January, 1909, during the discussion on "The Mechanism of Annealing of Certain Copper Alloys," by Messrs. Hudson and Bengough.

If metals are submitted to stress and at the same time their polished surfaces are viewed under the miscroscope the development of "slip-bands" may be clearly seen; they increase in number as the stress intensifies. Where the fracture ultimately occurs, parting takes place not along an irregular line following the boundaries of the crystals, but along the line coinciding with the least adhesive slip-bands or gliding-planes which have been developed through each crystal while submitted to stress.

Consequently where the structure is fine, the line of fracture will be straight and sharply defined; where coarse, it will be less regular and serrations will appear. So we may expect metal of perfectly homogeneous fine structure to be stronger than (a) metal of heterogeneous structure, i.e., where fine and coarse crystalline areas occur side by side, or (b) metal of homogeneous, but uniformly coarse structure.

In case (a) fracture will develop and rupture occur where the structure is coarse and the strength of the coarse areas will represent the strength of the whole metal.

In case (b) gliding-planes will be fewer in number but more pronounced, and their development less hindered by the hemming-in and arresting action of those less mobile inter-crystalline planes of greater mutual adhesion, which are more frequent where the structure is finer.

The effect of "twinning" is characteristic of wrought copper. Its explanation is perhaps rather obscure and is more easily understood by a study of crystallography. Suffice it to say that a crystal boundary of wrought copper may circumscribe one or more pairs of crystals (parallel laminæ), each pair having two sides coinciding, the other two sides running parallel thereto.

Distortion and recrystallization have completely disturbed the original polyhedral structure, the deformed and compressed crystalline elements having been compelled to take new and, as it were, unnatural situations. Even hot-wrought copper exhibits "twinning." Very homogeneous castings of copper or its alloys have a structure which is very susceptible to "twinning," the mildest deformation producing "twins." Micrograph (Fig. 6) shows a copper casting, deoxidation having been effected by means of manganese. The specimen was merely sawn from the casting, annealed for twenty minutes at 800° C. and slowly cooled, then polished and etched.

Behavior of Copper at High Temperatures

The influence of increased temperature on copper and its alloys, as on all metals, is to weaken the structure. Active crystalline growth, increased amplitude of molecular oscillation,

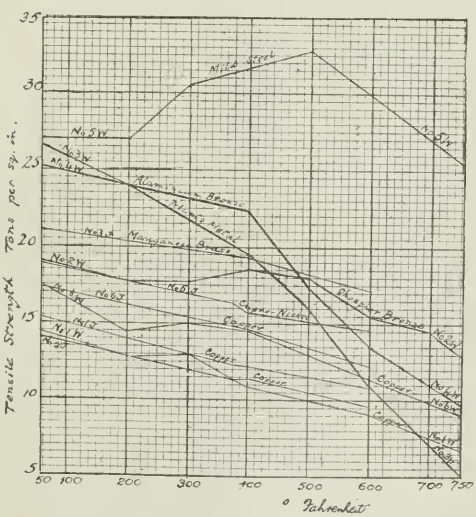


FIG. 8.—ALTERATION OF STRENGTH OF ALLOYS AT HIGH TEMPERATURES.

developing at still higher temperatures into incipient disintegration or, in the case of "burnt" metals into inter-crystalline oxidation, all render the metal less tenacious. Le Chaterlier found that pure copper fell off in tenacity with increase of temperature from a tensile strength of 14.7 tons per square inch

at 15° C. to 12.9 tons at 200° C., and then fell off rapidly until at 530° C. the tensile strength was 4.6 tons per square inch.

Roberts-Austen [see plate 31 (reproduced in part herewith, Fig. 7) of the Second Report Alloys Research Committee[3]] showed that pure copper and arsenical copper showed little decrease in tensile strength compared with copper containing 0.10 per cent bismuth, which, at 160° C. or 320° Fahr., possesses only one-third of the tenacity which it possesses at 15° C. The copper containing bismuth was considerably weaker than the pure and arsenical forms, even at normal temperature. Although throwing considerable light upon the malignant influence of bismuth the results tend to give one an exaggerated idea of the potency of the evil for the following reasons:

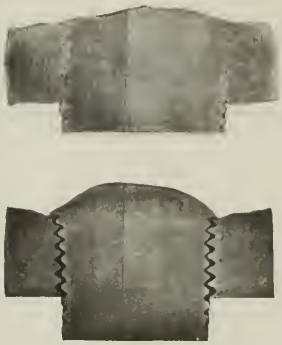


FIG. 9.—SHOWS ADVANTAGE OF COPPER OVER STEEL FOR FIREBOX STAYS, WHERE FIREBOX IS MADE OF COPPER.

Firstly, copper containing 0.10 per cent bismuth is rarely met with nowadays, and it may be said safely that the effect of bessemerizing is to remove bismuth from the pale of practical hurtfulness. Bismuth is very volatile, and Keller has shown that over 90 per cent of the bismuth contained in the original matte was volatized during the converting operation.

Gibbs states that "not only does the degree of elimination of any impurity depend upon the proportion originally present, but also upon the proportion of other impurities present." Be that as it may, bismuth is rarely met with in bessermerized copper in sufficient proportion to render the copper unfit for commercial purposes, although the writer has had personal experience of bessemerized copper containing sufficient bismuth to render it unfit for rolling hot.

As to the smaller quantities of copper produced by methods of roasting in reverberatory furnaces, including the processes of "bottoming" and "direct" refining, bismuth is not eliminated to nearly such a high degree as it is in the converter, while copper produced direct from blast-furnace smelting may possibly contain very nearly all the bismuth, arsenic, antimony, etc., originally present in the charge, so searching is the reducing action of the fuel and sulphur in contact with the metalliferous materials of the charge. Needless to say, very little of the world's supply of copper is produced direct from the blast furnace.

Secondly, commercial copper invariably contains an equilibrium-proportion of an element which profoundly modifies the harmful effect of bismuth, viz., oxygen.

Thirdly, in all copper required by engineers for purposes where electrical conductivity is of no account, and where toughness, strength and ductility are the desiderata, the presence of arsenic up to or even exceeding 0.5 per cent is eminently desirable, and this again suppresses the deleterious influence of bismuth, both at low and at high temperatures.

So it is important to know, having shown that Prof. Roberts-Austen's results are exaggerated by the artificiality of his otherwise splendid experiments, in what proportion bismuth is admissible in copper required by engineers to possess strength, toughness and ductility at all temperatures.

It is here that the writer would make rather full reference to a splendid paper read before the Institution of Civil Engineers in 1902 on "Locomotive Firebox Stays." The author, Mr. F. W. Webb, conducted a series of experiments on the suitability of various kinds of copper and copper alloys for staying locomotive fireboxes. The most frequent sources of trouble are those arising from *wastage of heads,* from *leakage* and from *breakages.*

(1) Wastage occurs principally in the vicinity of the brick arch where the heat is most fierce.

(2) Leakage occurs wherever the joint between the stay and the firebox plate has become loosened.

(3) Breakages occur principally at the top corners and may be attributed to the expansion and contraction of the inner copper casing. When they occur at the lower part of the box they are invariably due to overheating, consequent on becoming thickly coated with a non-conducting layer of boiler scale.

Exhaustive practical trials were made with engines fitted with stays of various kinds and extending over a number of years with the following results:

The above table has been compiled from the data furnished by Mr. Webb in the paper to which reference has been made.

In Fig. 8 are plotted the results of Webb's experiments on the various alloys used when tested at high temperatures. Their numbers and compositions are given in Table II, together with those of similar alloys tested by Mr. S. W. Johnson under similar conditions. Webb's alloys are marked W, those of Johnson being marked J.

TABLE I.

Material of Stay.	Total Mileage.	Mileage per Replaced Stay.	Description of Engine.	Boiler Pressure.	Cause of Replacement.
Copper	5,622,064	844	Compound Express Passenger	200 lb.	Wasted heads and leakage
Copper	277,800	277,800	Compound Express Passenger	200 "	Nil.
Muntz Metal	169,058	262	Passenger	150 "	Wasted heads and leakage.
Brass { Cu 91 Zn 9	284,978	5,588	Coal-Engine	150 "	Wasted heads, leakage and breakages.
Bronze { Cu 97 Sn 3	113,657	791	Express Goods	150 "	Wasted heads and leakage.
Aluminium Bronze	2,400	Not stated	Express Goods	150 "	Breakages.

So it would appear that high tensile results at normal temperatures serve rather to indicate unreliability in practice. Nor do good tensile results at high temperatures necessarily indicate reliability in practice, e.g., mild steel. This only shows a mileage per stay of 610 as against 844 for copper, and this under much less severe conditions than the copper stays had to endure. The hardness of a metal when used for firebox stays where the fireboxes are of soft copper seems to be the main cause of failure. Screwing a hard metal into a soft one has the effect of abrading the thread in the latter, but, what is much more serious, subsequent riveting has the effect of expanding the hole in the copper firebox plate, causing imperfect contact between the stay and the firebox, and so leading to leakage. (See Fig. 9.)

Uniformity of tensile tests over a wide range of temperature is more probably a truer guide where the metal is of sufficient softness; the comparatively soft phosphor-bronze No. 2 W shows higher results and greater uniformity than copper Nos.

[3]Inst. Mechanical Engineers, 1893

6 W, 1 W, I J and 2 J, but even its slightly higher degree of hardness leads to greater leakage. In this connection Mr. Webb's experience was that nothing provided so excellent a joint as a copper stay, the best joint being where the stay was of copper containing slightly less arsenic and therefore possibly softer than the copper plate into which it was screwed.

What is required is a metal as soft as or softer than commercial copper, of greater strength in tension, of greater elasticity, greater resistance to alternating stresses and capable of maintaining its superiority under the variable thermal conditions of a modern locomotive boiler working under high pressure. In addition it must have a melting point not appreciably lower than that of copper and a similar coefficient of expansion.

Cupro-nickel and cupro-manganese seem to be the most suitable alloys yet tried, although cupro-manganese does not wear so well as copper in the vicinity of the fire.

The greater tendency to red-shortness in all copper alloys

(b) The elongation for equal loads within the stretching limit increases generally with increasing temperature.

(c) The elongation is raised by cooling to $-20°$ C.; between $+20°$ and 300° C. it is nearly constant, while higher temperatures generally lower the elongation of the *annealed* material and increase that of the *mechanically hardened* material.

(d) The increased tenacity and elasticity conferred by mechanical working remain up to 200-300° C. practically unaltered, then, as the temperature increases, these properties fall off until at 500° C. they are the same as those possessed by the annealed material at that temperature.

The elongation of mechanically worked material remains lower than that of annealed material up to 300° C.; beyond this temperature it may have a higher or lower value up to 600° C., depending on its composition.

(e) As regards chemical composition it would appear (although this point is not clearly brought out in the research)

TABLE II.

Analyses of Materials Tested at High Temperatures (See Curves, Fig. 5.) (Webb & Johnson.)

No.	Description of Material.	Cu %	Zn %	Sn %	P %	As %	Ni %	Bi %	O %	Sb %	S %	Pb %	Mn %	Al %	Comb. Carbon %	Fe %
1W	Copper, deoxidized	99.76	nil	nil	0.01	0.20	0.04	nil	nil	trace	0.01	trace	nil	nil	nil	trace
2W	P. Bronze	97.07	....	2.91	0.02	....	....	....	....	....	....	....	....	....	.....	....
3W	Muntz Metal	60.23	39.61	....	trace	0.10	....	....	....	0.01	....	trace	0.03	....	.....	0.28
4W	Aluminium Bronze	92.91	....	....	....	....	....	....	....	....	....	....	....	7.13	.....	....
5W	Mild Steel	....	....	....	0.07	0.04	....	....	....	....	0.03	0.48	....	....	0.13 Si 0.05	99.20
6W	Copper, deoxidized	99.89	....	....	0.08	0.01	0.02	....	....	....	trace	....	....	....	.....	trace
1J	Arsenical Copper	99.18	....	trace	....	0.520	0.088	0.003	0.124	0.019	0.005	....	....	....	.....	trace
2J	Arsenical Copper	99.42	....	....	....	0.27	0.04	0.012	0.108	0.013	....	0.06	....	0.008	.....	0.022
3J	Manganese Brass	61.50	37.47	....	....	....	0.20	Mere trace	....	....	....	0.12	0.52	....	.....	0.27
5J	Cupro-Nickel Alloy	95.51	....	....	....	....	2.91	....	....	....	....	....	0.02	..	Si 0.13	1.59
6J	Copper, deoxidized	99.84	....	...	0.103	0 .007	0.16	0.001	0.009	0.005	....	....	....	...	.....	....

TABLE III.

(From Reports from the Imperial Technical Research Inst. in Berlin.) Vol. XVI., 1898.

Material P = Plate R = Rod	Composition %								Annealed Material.			
	Sn %	As %	Sb %	Bi %	Pb %	Zn %	Ni %	Fe %	Elastic Limit kg/sq. mm.	Tensile Strength kg-sq. mm.	Elastic Ratio	Elong. % on 60 mm.
P I	2.25	nil	trace	trace	nil	0.37	nil	0.03	8.0	29.9	27	58.2
R IX	1.86	nil	trace	trace	nil	0.43	0.04	0.10	6.5	28.6	23	55.1
P II	0.03	0.01	0.17	trace	0.02	nil	0.29	0.03	5.6	23.6	24	48.7
R X	trace	0.02	0.19	0.01	trace	nil	0.20	0.04	6.5	23.9	27	48.3
P VI	nil	nil	nil	trace	nil	nil	nil	0.01	4.9	21.9	22	47.7
R XI	0.02	0.04	nil	0.01	0.05	nil	0.15	0.04	4.8	23.8	20	51.0

N.B.—It is to be regretted that the oxygen contents were not included in the above figures.

of whatever composition is a serious defect, the breaking down of cold metal being more expensive than breaking down hot in course of manufacture.

Alluding once more to the testing of metals at high temperatures it should be noted that mechanical tests at the temperature of an air or oil bath are not comparable with the actual conditions in practice, e.g., where a stay is strongly heated at one end and kept cool at the other.

Mention must now be made of the work of the Royal Technical Research Commission of Berlin (1898) concerning the influence of heat, and of the conclusions at which that commission arrived after an exhaustive series of experiments on copper rods and strips of different composition and of different mechanical treatment.

Table III gives the composition of the different kinds of material used in the research and the following is a summary of the results of the investigation into the influence of heat thereon.

(a) The tenacity of the material is diminished with increase of temperature, the effect being much greater above 200°-300° C. than below.

that the more impure copper retains the proportionality derived by mechanical working better at higher temperatures and gains greater tenacity where the treatment has been the same.

It is very interesting to note that these experiments have been amply confirmed by Webb as regards the beneficial influence of tin. Copper containing 2 per cent or 3 per cent of tin, if carefully deoxidized, seems to behave better than pure or even ordinarily impure "refined" copper at high temperatures.

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A New Primary Cell.

In a paper presented before the British Institution of Electrical Engineers, Mr. W. R. Cooper describes a new primary cell invented by a Hungarian engineer, Stephan Benkö. As described, it is chemically an ordinary zinc-sodium bichromate cell and its original feature rests in the mechanical construction, which permits to overcome polarization at the carbon electrode. The fundamental idea is that polarization is due to the electrolytic change of the electrolyte at the carbon

electrode, and that, therefore, if the layer of electrolyte next to the carbon electrode is removed as soon as it is changed electrochemically the e.m.f. should be maintained constant and no polarization should set in.

The construction is indicated diagramatically in Fig. 1. Here the carbon is shown at C, and this is best described as a flattened cylinder, open at both ends. The hard surface of

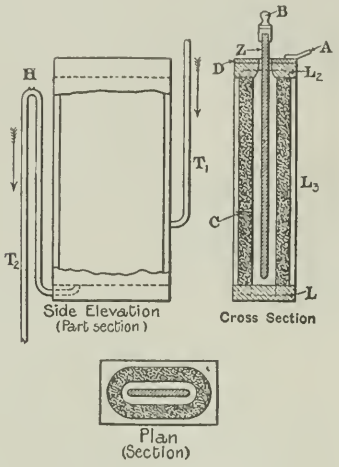

FIG. 1.—DIAGRAM OF BENKÖ CELL.

this carbon, as received from the carbon makers, is removed by scratch-brushing so as to leave it easily porous. The carbon is then provided with a lead cap, L_1, at the bottom, and a lead ring, L_2, at the top, thus providing a vessel open only at the top. A lead shell, L_3, of sheet lead 1 mm to 1½ mm in thickness, is then fitted round the carbon, leaving a small space between the carbon and the shell, and is jointed autogenously to the lead cap at the bottom and the lead ring at the top so as to form a chamber all around the carbon.

Thus there are two chambers, one inside the carbon and one outside. The outer chamber is provided with a Tube, T_1, and a second tube, T_2, is fitted to the lead base and is carried through to the inner chamber. The latter tube is carried up and bent over, and there is a small hole, H, at the top of the bend. Finally, a copper plate, D, corresponding with the lead ring, is soldered on to the lead, and forms one terminal, A, of the cell.*

The zinc, Z, is inserted into the inner chamber and carries the second terminal, B. When in action the electrolyte is delivered by the lead tube, T_1, into the outer chamber, when it percolates through the carbon to the zinc: it then flows off through the lead tube, T_2, the form of which maintains the level constant within the carbon. As there is a hole at H, syphoning does not occur, though this may be made to take place in cells so constructed by closing the hole with a finger for a few minutes. This is often convenient when it is desired to empty the cell.

It will be noticed that the current is carried away from the carbon at both top and bottom, and owing to this form of construction, and to the excellent contact between the carbon and the lead, very much heavier currents can be taken than is possible with the usual form of joint. A working drawing of what is termed the "Type A" cell is reproduced in Fig. 2, in which the lettering of Fig. 1 is retained as far as possible. The zinc plate is carried in an ebonite frame, F. The tubes T_1, T_2, are provided with unions and rubber washers, W_1, W_2, for coupling to other tubes as may be found necessary.

The carbons at present are not made larger than is shown in Fig. 2, but if larger cells are required this is effected by fitting two or more carbons into one shell. These are thus permanently connected in parallel, and the zincs are correspondingly coupled together.

The particulars of the "single" cell are:

Width	1¾ in.
Length	6¼ in.
Height (including terminals)	9¼ in.
Weight (including zinc)	8.8 lb.
Maximum current continuously with L solution	17.5 amp
Power	26.25 watts

The current stated is for a voltage of 1.5 volts per cell, the e.m.f. being 2 volts.

In the case of a battery the electrolyte is contained in a lead-lined tank, from which it flows (due to its own head) down to a common supply pipe from which each of the cells is supplied by a tube, the junctions being by ebonite unions and rubber washers, as already described. Similarly the waste from each cell is carried into a common waste pipe, from which an inverted U-pipe is taken to maintain the level in the cells, and this discharges into a lower tank. By means of a cock the common waste pipe can be opened to this tank so that the cells can be drained when the supply of electrolyte has been cut off.

The arrangement is compact, the width across the front of the stand being only 25½ in. and the depth from front to back being 24½ in. In order to give the necessary head the overall height is 7 ft. 9 in. for a 7-cell battery (and has recently been reduced). Air pressure might, of course, be used instead of a gravity feed; this would give greater compactness and might be preferable under certain conditions. But even with the arrangement shown, for a battery giving a minimum work-

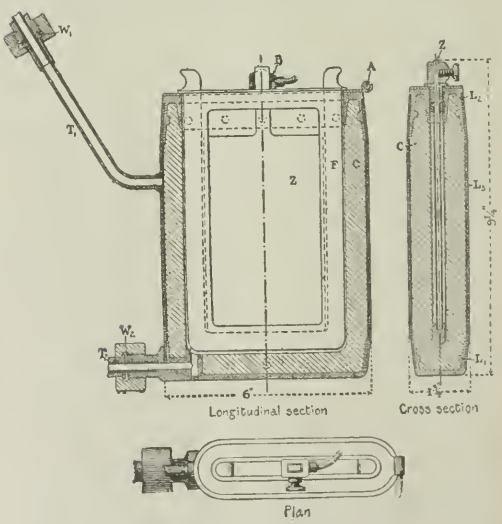

FIG. 2.—STANDARD TYPE OF SINGLE BENKÖ CELL.

ing pressure of 10.5 volts and a power up to about 260 watts continuously, the space required must be regarded as relatively small.

With a given flow of electrolyte an extremely constant current can be maintained up to a certain limit. This limit depends largely, of course, upon the composition of the electrolyte. Various electrolytes may be used, and in the case

*In later designs this copper plate has been abandoned, the terminal being jointed together to the lead ring

of bichromate solutions Mr. Benkö recommends the following formulæ: For electric light, 60 grams of sodium bichromate and 120 c.c. of sulphuric acid per liter of water (solution "L"); and for heavy currents, 100 grams of sodium bichromate and 150 c.c. of sulphuric acid per liter of water (solution "P").

The limiting currents obtainable continuously with the two first solutions in terms of the area of the zinc (taking both sides into the measurement) may be taken as follows:

	Per square decimeter	Per square foot
Solution "L"	6 amp	56 amp
Solution "P"	10 amp	93 amp

There is some local action due to the reaction of the acid electrolyte with the zinc, and this local action varies with the flow of electrolyte and with the current taken from the cell

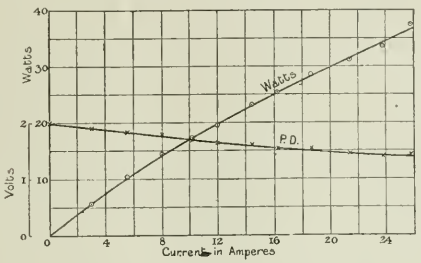

FIG. 3.—RELATION OF TERMINAL VOLTS AND TERMINAL WATTS WITH AMPERES.

(owing to the fact that this alters the composition of the electrolyte) and with the state of the zinc.

There is also some polarization, but it is difficult to separate it from the internal resistance. Fig. 3 shows how the terminal pressure and terminal watts vary as the current is increased. The currents were not taken momentarily from the

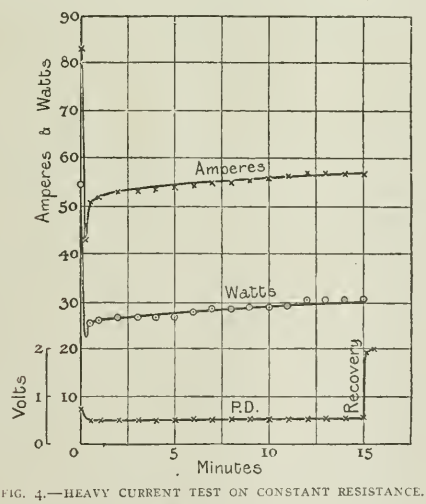

FIG. 4.—HEAVY CURRENT TEST ON CONSTANT RESISTANCE.

cell, but the external resistance was decreased step by step as quickly as the readings could be taken, and therefore polarization no doubt reduced the higher current readings to some extent. The highest outputs shown would not be maintained for any long periods, as the cell is overloaded. This test was made with solution L.

The test shown in Fig. 4 was made with a similar single cell, using the stronger solution P at a head (between the bottom of the tank and the top of the overflow tube) of 45¾ in., which gave a flow of about 1.9 liters per hour. In this test the external resistance was maintained constant, and the fall of current was observed for 15 minutes. The current might have been maintained longer, but the cell became very hot with such a heavy discharge.

Short-circuit tests with the L solution showed momentary currents of 100 amp at a terminal voltage of 0.5 to 0.6 volt, the current falling to 80 amp at the end of 5 seconds. With the stronger P solution momentary currents of 200 amp were taken at a terminal pressure of 0.5 to 0.55 volt. It follows, therefore, that the internal resistance is well below 0.01 ohm, an extremely low figure. This result is no doubt due largely to the excellent contact between the lead shell and the carbon, both at the top and at the bottom.

The Benkö cell has the advantage that it is small, easy to handle, and requires no careful attention. When it is not required it can be left on one side without fear of deterioration, the only desirable precaution being the running of some water through it to free it from the used electrolyte after heavy work. The laboratory, the charging of ignition cells, electroplating on a small scale, and possibly submarines (owing to the peculiar conditions involved) are fields in which such a battery would prove useful. There is also the electric lighting of yachts having no generating plant. It is also thought that the cell would be useful for telegraphy and in special cases for country-house lighting.

The simplest method of working such a battery is to turn on the flow of electrolyte when the light is required, and the flow would be turned off on retiring at night. With regard to labor, it may be mentioned that the zincs weigh 1 lb., and if we take 3 lb. (allowing for waste and unused zinc) per kw-hour, then a single cell will give ⅓ kw-hour and a battery of 17 cells will give 5 2/3 kw-hours before the zincs need be changed. If a house is using 1 to 2 kw-hours per day this means that the zincs are changed once or twice a week. As the operation is very simple, it takes very little time.

From the point of view of labor the electrolyte is more important than the zinc. With a constant output the amount of electrolyte per kw-hour will be, say, 7¾ gal. If, however, a battery such as we have assumed is allowed to flow steadily at a rate of ½ liter per cell, then a 5-hour run would require 9½ gal. This emphasizes the desirability of automatic regulation of the flow.

If hand regulation is adopted and not properly applied the amount of solution required might thus be as much as 15 gal. per day in the winter, or say, 10 gals. per day if the electrolyte is used twice and "refreshed" by adding new electrolyte to it. On the other hand, with automatic regulation this quantity once or twice a week for 1 to 2 units per week would be enough. With proper arrangements the handling of these quantities would take very little time. These figures emphasize the fact that the cost of running depends largely on the method of running.

As far as can be judged the cost of repairs should be small or negligible, and equally the interest on capital is scarcely worth considering. Mr. Cooper concludes that electrical energy can be generated in this cell at about 50 cents per kw-hour under favorable conditions, and this cost will vary according to the carefulness of the regulation.

The Analysis of Manganese Bronze.*

By James R. Huber.

In nearly all of the specifications for manganese bronze castings it is quite noticeable that attention is called to the physical requirements and very little is said regarding the chemical analysis. This is no doubt partly due to a lack of knowl-

* A paper presented at the Pittsburgh meeting of the American Brass Founders' Association.

edge on the part of the engineers who draw up the specifications of the exact composition of this rather complex alloy.

A method for the analysis of this alloy was worked out in our laboratory last year and published in the Metallurgical and Chemical Engineering for August, 1910 (vol. VIII, p. 463). This analysis presents several difficulties, especially the determination of zinc and aluminium. As we have gained knowledge by experience, we have been able to make some improvements on the original method.

A 5-gram sample is dissolved in 40 c.c. of 50% nitric acid (HNO_3). The small precipitate of metastannic acid obtained is invariably in such a finely divided state that much time is consumed in endeavoring to obtain a clear filtrate. A slight cloudiness in the filtrate will mean a result from 0.05% to 0.15% too low on the tin.

We find it advisable to cool the solution after boiling off the brown fumes, stir, and pour on a double S. & S. No. 589 blue ribbon filter paper. Fill the funnel once or twice and then apply suction, gradually at first, until the paper is well clogged with the precipitate. If the first of the filtrate is not clear, return it to the beaker and pour it on the paper again.

With these precautions, it is seldom necessary to repeat this operation and a practically clear filtrate is obtained. It is better to wash the precipitate with hot 10% nitric acid (HNO_3) instead of 2%, as the more dilute wash water has a tendency to start the precipitate running through the filter paper.

The filtrate is then evaporated to white fumes with concentrated sulphuric acid (H_2SO_4). Five c. centimeters is not enough, as the mass forms a hard cake and has a tendency to spatter when nearly dry. We now use 15 c.c. to 20 c.c. With this amount it will not be necessary to make a further addition when removing copper by electrolysis. The bronze will not contain any lead if made with high-grade spelter, but, should a trace be present, it is filtered off and weighed as lead sulphate ($PbSO_4$) on a platinum Gooch crucible. The filtrate is diluted to 500 c.c. in a 500-c.c. graduated flask.

Four 100-c.c. portions are measured out with a pipette or graduated flask into tall non-lipped 200-c.c. beakers. Two of them are neutralized with ammonium hydrate (NH_4OH), 7 c.c. of a mixture of 2 parts 50% nitric acid (HNO_3) and 1 part 50% sulphuric acid are then added and the solution diluted to about 150 c.c. The copper is plated out, using either a rotating anode or a solenoid. With the latter a brighter and more adherent deposit is obtained in less time than with the former.

The shape of the spiral anode causes the deposit to be more evenly distributed over the surface of the cathode and a much stronger current can be used. With a gauze cylinder cathode having 150 to 200 sq. cm of surface, 10 amp will deposit about 1¾ grams of copper in 15 minutes. Three amperes was the highest current we were able to use with the rotating anode and obtain a satisfactory deposit. We find it better to dry the cathodes in an air bath at about 100° C.

From these two portions we have duplicate determinations on the copper. In the filtrate from one the iron is precipitated with ammonium (NH_4OH), dissolved in dilute hydrochloric acid (HCl), reduced with stannous chloride ($SnCl_2$) and titrated with potassium permanganate ($KMnO_4$), according to Blair. In the other portion the manganese is determined by the well-known bismuthate method. Rinse off the anode with a little sulphurous acid (H_2SO_3) to remove the slight deposit of manganese dioxide which forms upon it and add these washings to the solution in the beaker. Bromine water and ammonium hydrate (NH_4OH) are then added, the solution boiled and the precipitate of manganese dioxide (MnO_2) filtered out and redissolved in sulphurous acid (H_2SO_3). Ten centimeters nitric acid (HNO_3) and 30 c.c. water (H_2O) are added, the solution boiled a minute and ¼ gram of sodium bismuthate added. Heat till the pink color entirely disappears. Add a few drops of sulphurous acid (H_2SO_3) to clear the solution, and boil again to expel all nitrous fumes. Cool the solution until quite cold and again add a slight excess of sodium bismuthate. Stir a minute or two and then add 50 c.c. of 3% nitric acid (HNO_3) and filter through asbestos on a platinum cone, using suction. Wash with 50 c.c. more of 3% nitric acid (HNO_3), reduce with standard ferrous ammonium sulphate solution and titrate the excess with potassium permanganate ($KMnO_4$), which has been standardized against the ferrous ammonium sulphate.

The copper is removed from the other two 100-c.c. portions by electrolysis without the addition of nitric acid (HNO_3). The zinc is separated from the aluminium, manganese, and iron by precipitation with hydrogen sulphide (H_2S). Instead of reducing the iron by boiling with metallic iron or aluminium, or sodium thiosulphate, as suggested in Waring's modified method, we find it saves time to reduce with sulphurous acid (H_2SO_3). The use of aluminium is prohibited as we determine aluminium in the filtrate from the zinc. Methyl orange is used as an indicator when neutralizing the solution, and as it is not sensitive in a hot solution, it would be necessary to cool after boiling with any of the above reducing agents. In using sulphurous acid (H_2SO_3) it is only necessary to add about 10 c.c. to the cool solution, stir and neutralize with sodium hydrate ($NaOH$) and sodium bicarbonate ($NaHCO_3$), using methyl orange as an indicator.

We obtain the zinc sulphide in a granular condition and entirely free from iron by precipitating in the presence of hydrochloric acid (HCl) and ammonium sulphocyanide, as suggested by Kemmerer in an article in the *Journal of Ind. and Eng. Chem.* for September, 1910.

After neutralizing, dilute the solution to 400 c.c., add 4 or 5 drops of hydrochloric acid (HCl) and about 2 grams of ammonium sulphocyanide (NH_4SCN). Heat to boiling and pass a rapid stream of H_2S for about a half hour, keeping the beaker on a hot plate. Under these conditions the precipitate is rarely, if ever, slimy, but, should it form in that condition, it might as well be thrown away as it is impossible to filter it clear.

After the gas has been passing about 15 minutes, and the zinc is nearly all precipitated, add a drop of a dilute solution of sodium bicarbonate ($NaHCO_3$). If the black precipitate of ferrous sulphide (FeS) formed dissolves instantly, add another drop. But do not add any more if the black precipitate dissolves slowly. This precaution is taken to insure complete precipitation of the zinc.

There is a possibility that enough acid will be liberated in addition to the 4 or 5 drops added to prevent the last traces of zinc sulphide (ZnS) being precipitated. As this addition is not made until after most of the zinc sulphide (ZnS) is formed there is no danger of iron being carried down. Continue passing the gas about 15 minutes longer. Allow the zinc sulphide (ZnS) to settle, filter and wash with hot water. Dissolve in dilute hydrochloric acid (HCl) and precipitate as phosphate with sodium ammonium hydrogen phosphate ($NaNH_4HPO_4$).

The aluminium is determined in the filtrate after boiling off the hydrogen sulphide (H_2S) and concentrating to about 200 c.c. Add a little nitric acid (HNO_3) to decompose the ammonium sulphocyanide (NH_4CNS). Boil off the cyanogen fumes, add 5 grams of ammonium chloride (NH_4Cl) and precipitate the iron and aluminium as hydroxides by making the solution exactly neutral with ammonium hydrate (NH_4OH). Boil, filter and dissolve in dilute hydrochloric acid (HCl). Dilute to 400 c.c. Add 10 c.c. of a 10% solution of sodium ammonium hydrogen phosphate ($NaNH_4HPO_4$) and then ammonium hydrate (NH_4OH) until a slight but permanent precipitate forms. Then add 1½ c.c. of con. hydrochloric acid and 50 c.c. of a 20% solution of sodium thiosulphate ($Na_2S_2O_3$). Heat to boiling and add a mixture of 15 c.c. of a 20% solution of ammonium acetate and 8 c.c. of strong acetic acid. Boil 10 minutes, allow to settle, filter and wash with hot water. The precipitate is ignited and weighed as aluminium phosphate.

Lumen Bearing Company, Buffalo, N. Y.

Notes on Smelting in British Columbia.

By E. Jacobs.

Sinter Roasting at Trail.

The Dwight & Lloyd Metallurgical Company, of New York City, has put in two of its sintering machines at the smelting works of the Consolidated Mining & Smelting Co. of Canada, Ltd., at Trail, B. C.

There are a number of these plants in operation in various parts of the world, namely, nine in the United States, one in Mexico, five in Australia and several in Europe. The largest plant at present is one of those in the United States, with a capacity of 450 tons per diem. One still larger, of 500 tons capacity, is being erected at Broken Hill, N. S. W., Australia. Several others are under construction in the United States and Australia, respectively. Of those installed in the United States all but one have had, or are having, their capacity increased. The plant lately completed at Trail is the first to be installed in Canada, and has a daily capacity of about 200 tons dry weight fed to the machine.

The Dwight-Lloyd machine is particularly adapted to handling concentrates, flue dust and all classes of fine ores, converting them into a coarse cake which, owing to its strength and porosity, is highly suitable for reduction in the blast furnace. It will economically sinter and desulphurize mixtures having from 8 per cent to 18 per cent sulphur, and of sizes from ¼ in. down to 200-mesh or finer, making from them a coarse, porous product which will contain between 2.5 per cent and 5 per cent sulphur.

The machine proper consists of a structural steel framework carrying a suction box, a hopper, a line of cars or pallets, tracks for these to run on, and driving mechanism. The last mentioned consists of a pair of steel sprockets driven by a train of gears. The cars or pallets are planed on the bottoms and ends, so that they make an absolutely air-tight joint one with the other. In the bottom of these pallets are fastened cast-iron herring-bone grate bars, with slots ¼ in. wide. These pallets slide along the top of a large steel-plate suction box with planed cast-iron top. At each end of the suction box is a dead-plate which serves to make an air seal for the pallets as they pass on or off the suction box.

At Trail there are two large sprocket wheels about 8 ft. in diameter, on a heavy shaft, which constitute the means of operating the whole machine. On the underside of the machine there is a straight inclined track connecting at each end of the machine with a piece of curved track. The manner of operating the machine is as follows: The pallets are delivered by gravity into the sprocket wheel, the teeth of which engage the pallets by their small wheels or rollers, picking them up one pallet at a time. The curved track at the end of the machine serves as a guide to keep these wheels engaged with the teeth of the sprocket wheel. When they reach the top there are two guides on the end of a wind box on which the pallets come down, and the sprocket teeth then push them in a straight line over on to the wind box. At the same time they pass under a hopper full of ore which feeds a layer of ore mixture of uniform thickness onto the grate bars. Just at the point where a pallet passes onto the end of the wind box there is a gasoline burner by means of which the top part of the layer of ore is ignited. A high pressure exhauster is connected with the wind box by a pipe, and this draws air through the layer of ignited ore. By this means most of the sulphur contained in the ore is burned away, and at the same time the heat resulting from its combustion sinters the material, there being enough heat developed to slag the silica and iron contained in the mixture. While this is going on the pallet is slowly traveling across the wind box, and the speed of travel is so regulated that by the time it shall reach the end of the wind box the material is completely sintered. At the end of the wind box the wheels of the pallet are again engaged by the track which lifts the pallet and allows it to fall about 24 in. around the curved track at the end. It strikes the fallen pallet ahead of it with such force that the sintered cake is discharged from the grates and to a great extent cleans the slots of any adhering sinter. The cake then falls into a hopper or directly into steel cars with sufficient force to break it into pieces suitable for feeding into the blast furnace. The sintered product, therefore, does not require to be broken by hammers as is the case with some other processes. The length of the machines at Trail is 47 ft. over all. The effective hearth area is 42 in. by 264 in. There are 42 pallets on the machine, each 42 in. by 24 in.

The system of ignition depends largely on local conditions and the size of the plant. Gasoline is being used at Trail, but gas, crude oil, coal, coke, or even wood, may be used if more convenient. The gasoline burner is simply an arrangement which gasifies the gasoline in a coil made of brass tubing into the body of which the liquid fuel is sprayed by a needle valve. The gasoline comes out through a number of exceedingly small holes in the bottom of the body of the burner, and the flame is made to impinge against the surface of the moving layer of ore.

The supply of ore comes from large storage bins, one for each class of ore treated. From the bottoms of these bins the materials are fed automatically onto a conveyor belt, the proportion of each kind being regulated by varying the discharge opening in the bin. The conveyor belt discharges into a bucket elevator which, in turn, discharges to another belt conveyer placed near the roof of the building leading to a storage hopper above the sintering machines. By means of automatic feeders the ore is fed from this bin into a pug mill, and at this stage the required moisture is incorporated in the ore mixture, a special valve delivering the water in the shape of a fine spray. The mixed and moistened charge then falls into the hopper of the machine, whence it is fed onto the moving pallets.

The motive power for the whole system of mixing and feeding, and for the machines themselves, is simultaneously regulated, so that once the feeders are set to the proper openings for a given mixture the whole plant may be accelerated or slowed down at will by adjusting one speed regulator. This can be done without changing the proportions of the ore mixture, the composition of which will, therefore, remain uniform.

Any source of continuous power may be used. At Trail the power is supplied by three alternating current motors, two of which drive the fans and one the machines, feeders, elevators, etc. The total power consumption of this plant is about 75 hp, viz., 30 hp for each of the two exhaust fans and 15 hp for the remainder of the plant. The fans are specially constructed for heavy service, with water-jacketed bearings. They are run at a speed of 850 r.p.m. and produce a vacuum of 10 in. to 11 in. of water. The 24-in. pipes connecting the suction boxes with the fans are of riveted steel.

The Dwight-Lloyd machines are used for both lead and copper ores, both of which are smelted at Trail. The whole process, from the time the ore is dumped into the first receiving bins until the sinter is discharged into cars for removal to the blast furnaces, is absolutely automatic. At the Selby works, California, the sinter is also taken away from the machine by a steel bucket conveyer and automatically delivered to a 100-ton steel storage bin. The 200-ton plant at Trail is operated with only three men on a shift.

Mr. Henry J. Stehli, of New York City, superintendent of the Dwight & Lloyd Metallurgical Company, was at Trail in May, personally supervising the starting of the plant there. If successful, as it is confidently expected it will be, the operation of this plant will do away with the use of the Huntington-Heberlein roasting system now in use at Trail; also with the briquetting of flue dust, for this material will be mixed with the charge and sintered. The Huntington-Heberlein installation comprises seven roasters and 24 converters.

It is of interest to note, by the way, that the Dwight-Lloyd process is also being used for sintering fine iron ores and iron-furnace flue dust. A mixture of these is made and the heat developed by the combustion of the carbon in the flue dust is sufficient to sinter the entire mass into a solid but porous cake, which closely resembles that made from lead or copper ores.

It is claimed that three points of superiority are noted in comparing the Dwight-Lloyd process with other roasting systems, viz.:

1. The cost of installation per ton-day capacity of this plant is only one-half to one-quarter that of any other roasting appliance.

2. The cost of operation is from $1.50 to $2 per ton less than that of the old hand-rabbled reverberatory roasters.

3. In connection with several of the plants installed there has been a noticeable increase of the capacity of a blast furnace running on a charge containing sinter produced by this process; in one case as much as 20 per cent. This is due to the great porosity of the sinter and the fact that it is practically fused to a slag.

Trail Smelter Statistics.

The extent of smelting operations at Trail will be indicated by the following approximate figures, which are for the calendar year 1910: Tons smelted, 445,210; metal contents, gold, 138,901 oz.; silver, 2,017,007 oz.; lead, 33,871,837 lb.; copper, 5,282,139 lb. These figures for the company's fiscal year ended June 30, 1910, were generally higher, especially in lead production (42,368,816 lb.), while gross value of metals produced was nearly $400,000 greater ($5,911,767). Figures for nine months ended March 31, 1911, show a further decrease in tonnage and gross value of production, the tonnage smelted during this period being 304,587 tons, while the approximate gross value was $3,334,000, of which 55 per cent was gold.

The following figures give details of the company's operations for the nine months mentioned:

1910— Month.	Tons Received.	Tons Smelted.	Approximate. Gross Value.	Percentage of Gold.
July	35,641	35,138	$384,000	62.7
August	33,453	32,301	420,000	49.5
September	33,333	31,105	288,000	57.5
October	34,356	34,132	430,000	47.2
November	35,053	36,066	421,000	56.5
December	35,310	36,364	351,000	53.4
1911—				
January	34,505	32,958	349,000	57.7
February	32,323	32,315	326,000	57.8
March	35,096	34,208	365,000	54.8
Totals	309,070	304,587	$3,334,000	55.2

Of the total quantity of ore received, as above, 46,755 tons was custom ore and the remaining 262,315 tons came from the company's own mines, as under:

	Tons.
Center Star Group, Rossland	139,972
Snowshoe, Boundary	83,201
Sullivan, East Kootenay	25,362
St. Eugene, East Kootenay (concentrate)	6,129
Richmond-Eureka, Slocan	2,476
Queen Victoria, Nelson	1,984
No. 7, Boundary	1,718
Phoenix Amalgamated, Boundary	1,143
Molly Gibson, Nelson	330
Total output for nine months	262,315

The greater part of the ore from the company's mines was gold-copper, namely, 228,018 tons; the remaining 34,297 tons was lead-silver ore and concentrate.

Slag Disposal at Grand Forks.

The Granby Consolidated Mining, Smelting & Power Company, Ltd., operating a copper smeltery at Grand Forks, Boundary district, B. C., is preparing to dispose of its slag by a different system than that which has been in vogue for some years.

Up to the present time the hot slag has been hauled from the furnaces to the dump in slag cars by steam locomotives, but hereafter the slag will be granulated and disposed of in a manner described below.

The extent of the smelting operations of this company, the smeltery of which is stated to be the largest copper reduction works in the British empire, and second only to the great Washoe works at Anaconda, Mont., is indicated in the following brief outline of the smelting plant and equipment at Grand Forks.

There are in use eight rectangular, water-jacketed blast furnaces; six 44 in. by 266 in., and two 48 in. by 260 in. at the tuyeres; depth, 15 ft. 8 in., having a total capacity of 4000 tons to 4500 tons of ore per diem; 12 Connersville blowers, with a total air capacity of 239,000 cu. ft.; specially designed mechanical furnace charging system; hot slag haulage by 14-ton steam locomotives with 44-cu.-ft. slag cars; three converter stands 84 in. by 126 in., operated by electricity, and one stand 72 in. by 100 in. operated by hydraulic power; converter capacity, 50,000,000 lb. of copper per annum; three blowing engines with a total capacity of 19,700 cu. ft. of air per minute, and much accessory plant and machinery.

During the earlier years of the company's operations the slag was granulated by water and run to the dump by gravity in the usual manner. Then, as the dump filled to an extent, making it difficult to maintain the grade requisite for this method of disposing of the slag, it became necessary to substitute the hot slag haulage system which is now about to be dispensed with in favor of a return to water granulation and a method of disposal described below.

Four large receiving bins will be erected below the present level of the slag dump and at a distance of about 150 ft. from the blast furnace settlers. The slag will be granulated by water immediately after its discharge from the settlers, in the customary manner, and be conveyed thence to the receiving bins in the launders. From the bins it will be distributed over the dump by a series of conveyer belts, at first up a gradual incline to a maximum height of about 100 ft. and then by a cross-conveyer to the dumping place. These belts will be the ordinary 30-in. conveyers. The incline will at first be over a series of trestles, but later the grade of the dumped slag will be so regulated as to give the requisite incline up to whatever height it will be found economically practicable to carry it. The belts will be operated by two electric motors.

Two main reasons have caused the management of the company to make this change in the method of slag disposal. One is the excessive cost of the present haulage system, owing to the slag having to be hauled about three-quarters of a mile to the dump, and the high cost of maintenance of locomotives and slag cars. Second, the increasing difficulty of disposal by the present system, as the dumping-room below the present level becomes more and more curtailed. The substitution of the new method for that long in use is expected to involve an outlay of between $30,000 and $40,000, but important compensating results will be lower cost of slag disposal and provision of dumping-room for fully 10,000,000 tons of slag. To long continue the present hot dumping system would mean the changing of the course of the north fork of the Kettle River, with resultant cost of land that would be destroyed thereby, so the change determined upon will avoid a large expenditure in that direction and at the same time bring about a substantial reduction in operating costs.

It will probably be about four months, or near the end of September or beginning of October before the proposed change will be effected.

The Utilization of the Wastes of a Blast Furnace.

By Edward M. Hagar.

Until the last decade, practically the only utilization of the wastes or by-products of a blast furnace was the use of a portion of the waste gases to raise the temperature of the incoming blast through heating the brick work in so-called hot stoves, and in some cases a small portion of the power value of the gases was obtained by burning them under boilers to generate steam for driving the blowing engines.

At the present time the calorific value of the waste gases is being utilized directly in gas engines for blowing purposes and for generation of electric power, a considerable portion of the slag is used in the manufacture of Portland cement, and the flue dust, consisting of the finest ore and coke particles, is being collected and converted so as to be rechargeable into the furnaces.

The aggregate saving or profits resulting from these three developments is a matter of millions of dollars per annum, and in a modern blast-furnace plant it would almost seem that pig iron was the by-product, and, indeed, the investment in the equipment to utilize these former wastes exceeds that of the blast furnace itself.

The writer, in his work, has come in contact with these evolutions, with plants in operation or under construction, of a capacity to produce 12,000,000 bbl. of Portland cement per annum from slag and limestone, using over 1,300,000 tons of slag in a year, these plants being driven entirely by electric current generated by gas engines directly from the waste blast-furnace gases, the power requirements being 40,000 hp for 24 hours every working day.

In one of the cement plants the first commercial method for reclaiming flue dust was discovered.

By using the blast-furnace gases directly in combustion engines, after suitable washing to remove the grit, the power obtained from a given amount of gas is equal to at least two and one-half times that obtainable by burning the gas under the boilers for generating for use in steam engines.

A modern blast furnace of the usual size, with gas-blowing engines and gas engines driving electric generators, will provide sufficient gas to furnish 7000 kw of electric power, in addition to driving its own blowing engines.

This permits the most modern steel works, such as those at Gary, Ind., to practically do away with the use of coal for power purposes, operating the rolling mills by electric power from the surplus gases.

The United States Steel Corporation, of which the Universal Portland Cement Company is a subsidiary, has already installed 250,000-hp gas-blowing and gas-electric units, which, it can easily be figured, displaces or saves the consumption of approximately 1,000,000 tons of coal per annum as compared to the old-fashioned method.

With the modern high-blast pressures and the use of fine Missabe ore, the finest of the particles, together with the coke dust, are blown out through the top of the furnaces and are caught in the flues, dust catchers and gas washers.

The iron ore in this dust amounts to fully 3 per cent of the total ore charged, which aggregates the large amount of approximately 1,250,000 tons per annum in this country. Until within a few years this dust has been thrown away or used as filling, although containing about 40 per cent metallic iron.

For many years efforts were made to use this material by compressing it into briquets, but the cost of the operation, together with the fact that the briquets disintegrated and the dust was again blown out, led to an abandonment of the briquetting plants.

The first commercially successful method of utilizing the dust was discovered by passing the material through the cement kilns at South Chicago. Experiments showed that with the proper heat treatment the coke dust could be burned off and the iron ore conglomerated into nodules or nuggets averaging over 60 per cent iron content. These nodules, when fed to the blast furnace, were easily and completely reduced. The fact that the sinter of the flue dust contains such a high percentage of iron and that all of the sinter is reduced, together with its physical shape assisting the steady movement of the charge downward in the blast furnace, thereby preventing so-called slips, makes the sinter more valuable per ton than any ore.

It was necessary to derive mechanical means for preventing the accumulation of the sinter on the walls of the kiln. Plants have been in operation for some years using this process, with endless chains carrying scrapers constantly passing forward through the kiln, and cooled in water on their return outside of the kiln.

Recently other methods of utilizing dust have been devised which may prove successful commercially, and the indications are that within a short time the greater portion of this former waste will be prevented.

The development of the Portland cement industry in this country and the extension of its uses have been marvellous, and the following table shows a remarkable increase in the production of Portland cement in the United States every year since 1895, when this country first reached the production of approximately 1,000,000 bbl.

Year.	Production of Portland Cement of United States. Barrels.	Production of Universal Portland Cement. Barrels.	Percentage of Universal to Total American Production of Portland Cement.
1895	990,324	.....	....
1896	1,543,023	.....	....
1897	2,677,775	.....	....
1898	3,692,284	.....	....
1899	5,652,266	.....	....
1900	8,482,020	32,443	0.39 per cent.
1901	12,711,225	164,316	1.29 per cent.
1902	17,230,644	318,710	1.85 per cent.
1903	22,342,973	462,930	2.08 per cent.
1904	26,505,881	473,294	1.78 per cent.
1905	35,264,812	1,735,343	4.92 per cent.
1906	46,463,424	2,076,000	4.55 per cent.
1907	48,785,390	2,129,000	4.36 per cent.
1908	51,072,612	4,535,000	8.89 per cent.
1909	62,508,461	5,786,000	9.27 per cent.
1910	73,500,000 gov. est.	7,001,500	9.52 per cent.

It may be of interest to note the increasing percentage of the total American production shown by Universal Portland cement, which is the only Portland cement manufactured in this country using slag as one of the raw materials. With the new plant now approaching completion the aggregate production of Universal Portland cement in the Chicago and Pittsburgh districts will amount to over one-eighth of the country's total. Expressed in weight, the output of the finished product will be over 2,000,000 gross tons per annum.

Our plants in the Chicago district will consume all the available slag that is suitable for the purpose from an aggregate of nineteen blast furnaces in the South Chicago works of the Illinois Steel Company and in the Gary works of the Indiana Steel Company.

Comparing the pig-iron production and Portland cement production of this country in figures of long tons, the percentage of Portland cement to pig iron in 1890 was six-tenths of 1 per cent, in 1900 10 3-10 per cent, and in 1910 47 per cent. The continuation of any such relative growth would mean that before 1920 the tonnage of Portland cement would considerably exceed

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that of pig iron. I would hesitate, however, to predict that such would be the case.

Portland cement is defined by the United States Government as the product obtained from the heating or calcining up to incipient fusion of intimate mixtures, either natural or artificial, of argillaceous with calcareous substances, the calcined product to contain at least one and seven-tenths times as much of lime, by weight, as of the materials which give the lime its hydraulic properties, and to be finely pulverized after said calcination, and thereafter additions or substitutions for the purpose only of regulating certain properties of technical importance to be allowable, not exceeding 2 per cent of the calcined product.

From this definition it will be seen that the raw material for Portland cement is not limited to any particular form of material. It may be made from any combination of materials that together furnish the proper elements. In this country Portland cement is manufactured from a number of raw materials, which, with a few exceptions, may be classed under four heads:

First.—Argillaceous limestone (cement rock) and pure limestone.

Second.—Clay or shale and limestone.

Third.—Clay or shale and marl.

Fourth.—Slag and limestone.

In all cases the raw mixture is a combination of some form of clay and some form of lime, and in the first and fourth classifications the clay materials contain some lime. This simply reduces the proportion of lime material necessary for a proper mixture.

In the manufacture of Portland cement from slag and limestone the molten slag flowing from the furnaces is granulated by a stream of water, loaded into cars and transported to the cement plant, where it is dried in rotary driers and receives the first grinding; it is then mixed in automatic weighing machines with the proper proportion of ground and dried calcite limestone. These are then ground together and burnt to a hard clinker at a temperature of nearly 3000° Fahr. in rotary kilns, using pulverized coal for fuel.

This clinker, after seasoning, is crushed and ground and mixed with a small percentage of gypsum to regulate the setting time. The cement is ground to such fineness that 96 per cent passes through a sieve having 10,000 meshes, and 80 per cent passes a sieve with 40,000 meshes to the square inch. It is then conveyed to the stock house for storage prior to shipment.

It is necessary to use a flux in furnaces supplying slag for cement manufacture, a pure calcite limestone. The limestone burnt with the slag must also be a pure calcite stone. It is also essential that the ores be of a uniform and proper character.

Inasmuch as Lake Superior ores are noted for their remarkable uniformity of analysis, the resultant slag obtained from the use of these ores and a pure calcite limestone is more uniform in its analysis than any form of natural clay deposits used in the manufacture of Portland cement, and the variation in the proportions of the two raw materials used in the manufacture of Portland cement from slag is less than those of any other materials mentioned above.

In addition, the opportunity for analysis and selection of the proper ingredients through the use of an artificial material is a great advantage as compared to the necessitous use of natural materials just as they are found with their variations in analysis at different depths.

In the intense heat of the kiln, under the influence of the oxidizing flame, any sulphides in the slag are completely burned out.

The rotary kiln commonly used ten years ago was 60 ft. long and 6 ft. in diameter. This has gradually been increased in length and diameter until the modern kiln is 140 ft. to 150 ft. long and 8 ft. to 10 ft. in diameter, and there are a few even larger kilns in use. Kilns are usually set at an incline of ¾ in. to the foot. With the lining and contents the modern kiln weighs 150 tons, and in revolving upon two bearings presents interesting constructional features.

In the case of the plant at Buffington, Ind., using 26,000 hp. situated between South Chicago and Gary, Ind., electric power is supplied from the steel works at these points. Each piece of machinery is driven by its individual motor. The high-tension line is connected through the cement plants, and the gas engines at these two steel works, 14 miles apart, operate continually in parallel.

This enables the cement plant to draw its power from either source or from both sources at the same time, as may be desirable at any time.

It has happened that one of these works has supplied power to operate the cement plant and has also furnished additional power at the same time to the steel works at the other end of the line.

The method of manufacture above described is the standard method of manufacturing Portland cement from natural deposits, and the finished product differs in no way from other Portland cements in chemical analysis, fineness, specific gravity, color, nor in the operation in practical work.

It has no peculiarities whatever and has no limitations as to its applications.

There is no difference, from the chemist's point of view, between the manufacture of Portland cement from natural deposits, such as limestone and clay or shale, and its manufacture from limestone and slag. Slag is really a mixture of the clay from the ore with the lime content of the stone used as a flux in the furnace.

Our method of manufacture of Universal Portland cement does not embody any real invention, nor is it based on any patents.

It is simply an adaptation to an artificial raw material of the regular Portland cement process formerly applied only to natural deposits.

True Portland cement in which slag is used as one of the raw materials should not be confused with Puzzolan or so-called "slag cements," which are simply mechanical mixtures of slag and slaked lime ground together without burning. Such cements are suitable only for use underground and in moist locations.

The manufacture of Puzzolan cements ni this country has practically been abandoned.

The remarkable growth of the Portland cement industry is not equalled by any other manufactured article. This is due to the economy, durability and plasticity of cement and concrete work.

While the large engineering work, such as dams, bridges, and heavy reinforced-concrete buildings consume large quantities of cement, the bulk of consumption at the present day is in a multitude of small uses. It takes an average shipment of only 5 bbl. a day to take care of the average customer of a large cement company.

For example, there is a steady increase in the application of cement to new uses on the farm, such as silos, fence posts, barn floors, feeding floors, watering troughs, corn cribs, etc. There, as elsewhere, concrete is rapidly displacing all forms of wood construction, this process being hastened by the continually advancing cost of lumber.

Beautiful effects are now being obtained in concrete surface finishes and its use in decorative work is advancing rapidly.

The use of Portland cement will continue to increase until the campaign of education of the small user has reached its finality.

Most of the problems of utilization of wastes or by-products of the blast furnace have been solved, and these solutions, in addition to being highly profitable, are powerful factors toward the conservation of our natural resources. Portland cement manufactured from slag, to a large extent, replaces wood; the waste gases displace coal, and reclamation of the flue dust conserves the deposits of iron ore.

The Problems of Tungsten Concentration.--II.

By H. C. Parmelee

Treatment of Old Tailings

The Cardinal mill of the Primos Mining and Milling Company is a remodeled gold mill, formerly known as the Boulder County mill. The tailings impounded during several years' run of tungsten ore before the canvas plant was installed are now being retreated at the rate of 20 tons per day. Naturally the grade of the material is not uniform owing to irregularities in its deposition and in the grade of the ores originally treated.

Ten stamps are used to break up the pulp and prepare a uniform feed for the concentrators. There is really little work for the stamps to do, and the operation is hard on them as there is little bed of ore to resist excessive wear of shoes and dies. The pulp is discharged through a 20-mesh screen and flows directly to one Wilfley table from which a fairly coarse concentrate is obtained. All of the tailings are returned by a Frenier pump to a Callow cone which thickens the pulp for tube mill feed. No screen analyses are available to show the work of the tube mill, but an inspection of the canvas plant would indicate that the sand is not ground fine enough, and that a heavier load of pebbles should be carried and the pulp further thickened to reduce the ratio of water to solids in the tube feed.

The tube mill discharge is pumped to a V-shaped settler, from which pulp is drawn by several syphons to a distributing box for three Frue vanners. All vanners thus receive the same feed and no attempt is made to classify the pulp. The vanner tailings are treated on the canvas plant which consists of 3600 sq. ft. of canvas arranged on stationary tables.

At the head of each set of tables is an inclined launder carrying the pulp to distributing boxes. One double box is provided for each pair of tables and wooden plugs are used to stop the flow of pulp to any box. Adjustable dividing strips are placed in the launder to equalize the flow of pulp to the several boxes. From the box the pulp is further distributed across the head of the table by means of several launders of varying length, and the flow over the table is still further equalized by strips of wood notched in a saw-tooth manner. A main service pipe for wash water extends along the upper end of the tables, with a valve and lateral pipe for each table. Two parallel launders, one for tailing and one for concentrate, extend along the foot of the tables and wooden aprons are so hinged that by shifting their position the pulp flowing off the table may be diverted into one launder or the other.

Another plant treating old tailings is that of Smith and Ardourell on Beaver Creek. The tailings from earlier work at the Wolf Tongue mill are hauled two miles and presumably treated at a profit. The average grade is reported to be about 2 per cent WO_3. Twenty tons a day are treated at this mill.

From the bins the pulp is conveyed by spiral conveyor to a shaking screen for the removal of stones and rubbish. It is then classified, the coarse grade being treated on a Wilfley table and the two finer grades on Monell slime concentrators. The Wilfley sand tailing is rejected and the middling reground in a Denver quartz mill. The Wilfley slime tailing and the Monell tailings are run over canvas. There are four canvas tables, each 4 ft. by 18 ft., built with a fall of 1 in. per foot in length.

Two grades of concentrate are recovered at this plant, instead of one as at the mill just described. The Monell slimer, which will be referred to in more detail later, will deliver two grades of mineral, one high and one low. The high grade from the Monell is combined with the Wilfley concentrate making a product running from 40-50 per cent WO_3. The Monell low-grade concentrate carries 30-40 per cent, and the canvas concentrate 10 per cent WO_3. If the Wilfley and Monell tables are running well the canvas plant makes a poor recovery and at times the concentrate is freely contaminated with pyrite and grains of gangue and scarcely worth saving.

When the present campaign on tailings is finished the mill will be fitted with crushing apparatus and adapted to treat crude ore. It appears probable that stamps will be installed, and that the capacity of the mill will be about 5 tons of ore per day as compared with 20 tons of tailings at present.

Treatment of Ore.

Before describing the other mills in detail it may be well to speak of the Monell slimer, as it is used in almost all of the plants and is regarded with favor. It is a slime concentrator of the traveling belt type with a shaking movement in the direction of belt travel. Its principal features are the unusual length and slow travel of the belt, and the large flat settling area afforded for the collection of fine mineral. The belt is 5 ft. wide and the working surface is 23 ft. long. The rate of belt travel is adjustable from 3 to 27 in. per minute, and is usually set at 18 in. The number of vibrations varies from 160 to 180 per minute. Pulp is fed at one side of the belt and tailing discharged at the other. The concentrate is collected in a box at one end in the usual manner.

The feed is first received in a classifying box set about 2 ft. above the deck of the table and near one end. Pipes from the several compartments of the box deliver the pulp to distributing launders arranged along the side of the table. The coarse grade is delivered near the end of the table where it forms a bed for the collection of the finer grades that are delivered farther on. Suitable wash water boxes are provided near the discharge end to wash off the sand and leave a clean concentrate.

The belt travels on strips of redwood laid lengthwise of the table. The upper or feed edge is slightly raised to direct the flow of pulp across the belt, and the lower or discharge side is inclined downward to aid in discharging the tailing. These inclinations at the edges of the belt extend for a few inches only, and the large area between them is perfectly flat, thus affording an excellent settling surface.

Owing to the manner of distributing the feed on the belt a bed of high grade mineral forms on its upper half, and a less well defined streak of lower grade mineral on the lower half. When the belt passes through the collecting box the mineral accumulates in corresponding beds of high and low grade material and the attendant keeps the two classes separate when cleaning the box. The illustration, Fig. 1, shows a Monell slime machine in the mill of the Tungsten M. & M. Co. on Beaver Creek. The bedding of high and low grade mineral is plainly shown on the end of the belt passing down to the concentrate box. The illustration also shows a Monell concentrating table which is of unusual design. The concentrates are drawn off through slots in the deck of the table and conducted by launders to boxes at the end.

The mill of the Tungsten M. & M. Co. has a capacity of about 5 tons in 24 hours. The ore is reduced by a jaw crusher to ½-in. size and fed to a Denver quartz mill fitted with 16-mesh diagonal-slot punched screen. The discharge is then passed through a 20-mesh screen, the oversize of which is treated on one Monell concentrator. The undersize is roughly classified, the coarse portions being sent to a second Monell table and the fine to a Monell slimer. There is no canvas plant at this mill and the tailings of all concentrators are run to waste. Two grades of mineral are recovered but no data can be given as to percentage recovery as there is no systematic chemical control of operations.

The Alton mill is a new plant modeled along modern lines, but as it has not been in steady operation nothing can be given but the plan of treatment. The ore is first crushed in a Samson No. 4 crusher, elevated to a bin from which it is delivered by a plunger feeder to a pair of 14-in. by 30-in. rolls. The roll product passes through a trommel fitted with 12-mesh screen, and the entire oversize is treated without further sizing on a double compartment Harz jig, having the same screen as the trommel. The undersize of the trommel and the hutch product of the jig are combined and treated directly on a Wilfley table.

Coarse concentrate is drawn from the side gates of the jig and the tailing is reground in a Denver quartz mill fitted with 20-mesh punched screen. The Wilfley tailings also are reground in the same machine. The reground pulp is pumped to an 8-ft. Callow cone, the underflow of which is divided between a Card table and a Monell slimer. The Card and Monell tailings are combined and treated on two canvas tables of 12-oz. duck, 6 ft. wide and 26 ft. long.

FIG. 1.—SHOWING MONELL SLIME CONCENTRATOR IN MILL OF TUNGSTEN M. & M. CO.

The manager of this plant expressed the opinion that the jig would make from 25 per cent to 40 per cent of the total recovery, according to the grade and physical condition of the ore. This belief is based on the treatment of 20 tons of ore when the mill was first put in commission.

The Wolf Tongue Mining Company whose mill at Nederland is shown in Fig. 2, is one of the two large companies that have been pioneers in the tungsten industry. The mill is an old remodeled stamp mill formerly used on gold and silver ores. As a result it is not as well arranged as a new plant would be, nor is its operation as satisfactory as that of a more modern plant. Nevertheless it shows very well the possibilities of coarse crushing and jigging tungsten ore, and it is probable that should the company erect a new plant it would conform closely to modern practice in ore dressing.

The company is at present treating custom ore which on an average will carry about 7-8 per cent WO_3. Each lot is sampled and bought on comparison of assays. The ore is first crushed in a Blake crusher and then in rolls set ½ in. apart Power is applied to but one of the rolls which makes 90 r.p.m., the other running free. The roll product is elevated to a Vezin sampler which cuts out a fraction of the lot. This is further reduced by the tenth shovel method, and the portion thus obtained split on Jones riffles. The sample is then ground in a coffee mill, finished on the bucking board and divided into portions for the buyer and seller.

The reject from the Vezin sampler falls on a belt conveyor running to a trommel fitted with 16-mesh screen. The entire oversize is jigged, and the jig tailing is dewatered and sent to stamps for regrinding. The trommel undersize and the hutch product of the jig are combined with the reground tailing and classified for table treatment. The slime from the dewatered jig tailing flows directly to the canvas plant.

Ten 750-lb. stamps are used as regrinders of jig tailings. This battery formed part of an old equipment and probably is used only because it is convenient. The pulp is discharged through 12-mesh ton-cap screen and combined with the undersize of the 16-mesh trommel and the hutch product of the jigs. This mixed pulp is then classified in Flood classifiers and treated on four Card tables. These tables make a fairly coarse high grade concentrate which is dried and sacked for shipment. The middlings and tailings are reground in another stamp battery through 50-mesh screen and concentrated on a fifth table.

The slime tailing from the fourth and fifth tables and the overflow from the fourth or last Flood classifier are collected in a distributing box and sent to three Monell slimers. A lower grade concentrate is recovered from these and their tailings are pumped to the canvas plant. On account of the preliminary coarse concentration, less reliance is placed on the canvas plant in this mill than in some others. It contains only 768 sq. ft. of canvas arranged in two tiers of tables, eight in a tier, each 4 ft. wide and 12 ft. long. A cement foundation was put in for the canvas and it appears to be perfectly satisfactory provided a firm foundation is made for the concrete work. The surface is smooth and there are none of the irregularities sometimes noticed in board surfaces where the boards have been laid either too close together and buckled, or too far apart and failed to close up on becoming thoroughly wet. Twelve ounce canvas is used and the tables have the usual fall of 1 in. per foot in length. Pulp is allowed to flow for three or four hours, after which the usual method of washing and cleaning is followed.

Cement settling basins for fine concentrates, with a cocoa matting filter at the outlet, are placed in the lower part of this mill. These basins have proved valuable in saving fine floating mineral from the canvas plant and from the concentrate boxes of the Monell slimers.

The Lakewood Mill.

In marked contrast to the principal methods employed in the plant just described are those practiced at the Lakewood mill of the Primos Mining & Milling Co. Those in charge of this mill and responsible for the methods employed are likewise pioneers in the milling of tungsten ore, and it is interesting to compare their methods with those of the other old company. The Lakewood mill is new and a model of construction as regards arrangement, convenience, cleanliness and facilities for saving time and labor. The illustrations, Figs. 3, 4 and 5, will tend to bear out these assertions.

At the head of the mill is ample bin room for the numerous

FIG. 2.—THE WOLF TONGUE MILL, NEDERLAND, COLO.

small lots of ore shipped by leasers when the company is in the market for custom ore. Separate crushing and sampling apparatus is installed for this work, consisting of jaw crusher, rolls and pipe sampler. The ore is bought outright on comparison of assays, and the lot is held until satisfactory settlement is made.

Ore from the company's mines is dumped into a large bin from which it is drawn over 2-in. grizzlies, the fine passing to the crushed ore bin. The coarse is reduced in a 10-in. by

16-in. Blake crusher and combined with the undersize of the grizzlies. The crushed ore bin is of 120-ton capacity, or about three times the daily capacity of the mill. Tappet-driven Challenge feeders draw the ore from this bin to the stamps through which all ore passes before concentration begins.

The battery frames are of heavy timber construction and the mortar foundation is built up of heavy timbers 18 ft. long set on end on a concrete foundation.

Concentration is divided into two principal departments; first, the customary table work with Wilfley tables and Frue vanners, and second, the recovery of fine mineral on the canvas plant. All of the concentrating tables are arranged on one floor below the stamps, and the canvas in three tiers below the tables. Concentrates from the Wilfley and Frue machines are combined. The product assays over 60 per cent WO_3.

The pulp from each battery of five stamps flows directly without classification to a Wilfley table. A clean concentrate is cut out and the balance is sent by a Frenier pump to a spitzkasten which serves the first set of five vanners. The vanner tailings are combined and pumped to a large V-shaped box. The pulp is delivered to the box at one end through a pipe

FIG. 3.—THE LAKEWOOD MILL OF THE PRIMOS MINING AND MILLING CO., LAKEWOOD, COL.

extending some distance beneath the surface, thus giving the coarse particles opportunity to settle. There are no partitions in this box, but as the pulp flows from one end to the other there is a rough classification due to the settling of the particles with the retardation of the stream.

Six spigot products are drawn from this box; the first or coarsest being thickened, reground in a tube mill and returned, and the other five treated on Frue vanners. The tube mill is shown in Fig. 5 in relation to the V-box and the Frue vanners. No screen analyses are available, but the statement is made that 85 per cent of the tube product will pass a 200-mesh screen. Pebble consumption is 50 lb. per day, but no figures can be given as to quantity of pulp ground. The vanner tailings and overflow from the V-box flow to the canvas plant.

The Lakewood mill has the most extensive canvas plant in the district, comprising 10,320 sq. ft. of cloth arranged in three tiers of tables. An idea of the area covered can be gathered from Fig. 3, showing the mill exterior. With the exception of the concentrating table floor, the entire lower portion of the mill is devoted to the canvas plant.

The construction of the distributing and collecting launders and other features of the plant is practically the same as described for the Cardinal mill of this company. The tables are cleaned once each shift and all canvas concentrates are collected in settling boxes on the lower floor. This is practically on a level with the drying house shown at the right of the mill in Fig. 3, where the concentrates are dried on steam tables and sacked for shipment.

The labor employed consists of six men on the day shift and three at night. During the day there is one man at the Blake crusher, two at the stamps and tables, two on the canvas and one in the drying and sacking room. At night there is but one man on the stamps and tables and two on the canvas.

The mill is steam heated and electric lighted. Electric power is received at 13,000 volts from the lines of the Central Colorado Power Co., and stepped down to 440 volts. Motors are

FIG. 4.—VIEW OF STAMP BATTERY AND WILFLEY TABLES IN LAKEWOOD MILL.

distributed as follows: 15 hp, crusher; 35 hp, 10 stamps; 5 hp, 2 Wilfley tables and Frenier pump; 10 hp, 11 vanners and 2 Frenier pumps; 35 hp, tube mill.

Water is supplied from North Boulder Creek and delivered at the mill storage tanks either by a high line ditch, or by a Worthington 3-in. turbine pump direct connected to a 15-hp

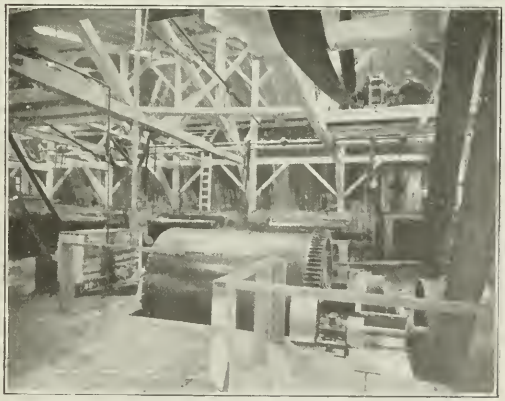


FIG. 5.—VIEW OF TUBE MILL WITH LARGE SETTLER AND VANNERS IN BACKGROUND, LAKEWOOD MILL.

motor. The pipe line from the pump to the tank house is 700 ft. long and rises 85 ft. An unusually large quantity of water is used amounting to 250 gal. per minute for the treatment of 40 tons of ore in 24 hours.

All of the Boulder County tungsten mills are fortunate in having a supply of clear water which is of advantage in all ore dressing plants, but particularly where so much fine mineral has to be recovered by means of the canvas table.

The Development of Economical Ore Dressing Systems.*

By Frank E. Shepard.

In the quest for mineral values there appear to be several phases of development, one leading into the other. The discoveries of many of our large mining sections have resulted from the placer workings in which the values are found in gravels on or near the surface of the ground. The primitive methods with pick, shovel, and hand rocker often served to develop rich deposits. Then followed the more highly developed hydraulic mining systems involving greater expenditure of capital but bringing greatly increased returns. This trail of values leads to some source and so the patient prospector finally reaches an outcrop or vein which requires deeper workings with a more elaborate plant.

The search is first for high-grade ores, for the difficulties of mine development, transportation, and treatment are great and expenses in proportion. Bonanzas, however, are rare, and then must come the patient search for ways and means of developing better methods of mining, milling, and smelting for the recovery of values from lower-grade ores. Refractory ores are found which call for more advanced methods of smelting or the grade of ore is such that milling systems must be devised for concentrating into values sufficient for commercial delivery to smelting or refining points.

While the general trend of ore values has been from high grade to lower grade, there has also been a sombre accompaniment of decreasing market prices for the metals calling for serious study in all metallurgical systems. Ten years ago, manufacturers of machinery for ore reduction processes, in the Rocky Mountain section, were devoting much attention to the details in smelting machinery for the treatment of high-grade ores. To-day there is less call for smelting systems and the demand appears for improved ore dressing machinery for the recovery of values from lower and lower grade ores. This change from higher to lower grade ore conditions has taken place generally throughout the fields of gold, silver, lead, zinc and copper ores.

The stamp mill, with its amalgamating plates and bumping tables for the recovery of 50 per cent to 70 per cent of gold and silver values in the earlier period, is being succeeded by the concentration and cyanide plants producing recoveries of 90 per cent and over.

Lead-iron-zinc ores are presenting more and more difficult problems, and with the mining of lower grade ores of these metals comes the demand for better methods of crushing, screening, hydraulic classification and concentration, and in this particular line of ore treatment appears the great development in electrical processes of mineral separation.

The successful mining and milling of the great porphyry copper ore deposits of a grade as low as 2 per cent and under has brought about radical changes in milling methods and concentrating mills of 10,000 tons daily capacity have been successfully developed.

The tremendous pace in modern steel production has produced such demands on the iron ore supply that commercial possibilities have been found in the lower-grade iron ore deposits of Michigan and concentration plants have been installed for the treatment of 20,000 tons of ore daily.

The call for economy and reclamation of values in the Cripple Creek District has resulted in treating the dumps formed by the discarded ores of a former period by means of modern concentration and the cyanide systems at the remarkably low cost of $1.50 per ton.

Business men making investments in mining enterprises have found that it pays to be guided by specialists in this line of work. So many mistakes are made by the mining companies rushing through the building of a mill before the mine is sufficiently developed. The professional fee of the competent geologist and mining engineer is well earned, and thorough examination and sampling of the mine should be completed to furnish sufficient data from which to design and develop a suitable mill system.

Having provided a representative sample of the ore to be treated, there are laboratories throughout the country where tests and experiments may be conducted and results obtained from which an economical system of ore treatment may be planned. The various manufacturers of mining and milling machinery offer engineering advice in the development of systems of ore treatment, and while this may be competent and the result of good practical experience, such advice depends upon data furnished by the mining company itself.

Large mining companies appreciate the value of preliminary investigations of ore conditions in a mine, and the manufacturer has competent advice from which to design and construct suitable machinery and equipment for the ore dressing system; but in the smaller mining organizations, the building of a mill oftentimes appears to be more important than the development of ore bodies and the manufacturer is called upon to furnish machinery to treat ores which may change radically in further development of the mine.

In extensive mining operations the plan is followed of erecting mills to test the practical application of the mill system determined upon, and this proves of great value in solving difficult problems of ore treatment.

In the smaller mining companies, often composed of business men without any experience in mining and milling methods, it is seldom found that sufficient funds have been devoted to these important preliminary tests and instead of employing competent technical advice some one of the directors of the company who "has a liking for machinery" assumes charge of affairs and the resulting mining and milling composition ends in minor chords. However small the proposed mining or milling installation, it is alway advisable to secure the advice of the mining or metallurgical engineer who studies the entire field of operation and the economic conditions relating to mine and mill, makes suitable examination of mine, takes sufficient samples of the ore, and after preliminary tests is prepared to submit general plans and specifications for a suitable mill system.

While the manufacturers of mining and milling machinery are by custom expected to prepare plans for milling systems, on account of the stress in business competition, such is not an economic condition, and it is better practice to have one engineer thoroughly advised on all technical matters from inception to the complete milling plant.

The manufacturer has the very important part to perform of attending to the many details of mill installation and it is his affair to keep up to the minute in all the improvements which aid in reducing costs and increasing recovery of values from the ores.

Mill design is a profession in itself. It is not necessary for the consulting, mining or metallurgical engineer of the mining company to be an expert in the details of mill construction as the machinery manufacturers make a study of these problems. The detail drawings of the mill should be intrusted to an engineer who has not only a very thorough knowledge of mill machinery details but an intimate knowledge of mill men and practice.

In the selection of mill machinery the new inexperienced mining company usually buys the cheapest grade, while the mill superintendent of long experience buys the best machinery that can be built. In a small mill treating 50 tons in 24 hours of an ore having a value of $15 per ton and assuming the mill recovers 80 per cent. of the value, a delay in operation of but one hour means a loss of $25. It is manifestly poor economy to save a few hundred dollars in the original investment at the expense of several thousand dollars in delays and repairs.

Ore crushers were formerly equipped with hard cast-iron jaw

*Presented before the Congress of Technology at the Fiftieth Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.

plates which were required to be removed every few weeks, whereas now these crushers are provided with the best crucible-steel jaw plates which last for several months. Crushing rolls were formerly equipped with cast-iron shells which gave short service and soon became grooved; now the best forged-steel, machine-finished shells are used with the result of longer service and better product.

Elevating and conveying apparatus has reached a high state of development, all tending to greatly reduce costs of handling materials. By means of these excellent modern devices the entire mill system is automatic from beginning to end, save for the adjustments of machinery.

One of the greatest if not the greatest improvement in the wet concentration of ores has been the important development in the preparation of ore pulps previous to jigging and table work as a result of the investigation of Dr. Robert H. Richards of the Massachusetts Institute of Technology.

Dr. Richards' devices for hydraulic classification of ore pulps, previous to table concentration, have produced savings in ore values over former methods amounting to hundreds of thousands of dollars annually.

If we refer to our small mill before mentioned having 50 tons capacity daily and recovering 80 per cent of values in $15 ore, an improvement in recovery of only 5 per cent will mean an additional annual saving of over $12,000. Such improvements have actually been accomplished in practical mill systems using Dr. Richards' systems of hydraulic classification.

The tube mill is an important element in modern mill systems and was adapted from cement mill practice. It takes the place of the more complicated crushing machines involving the frequent renewal of shoes, dies, screens, and various parts in roller or chilian mills and presents a simpler and more economical means for the fine crushing of ores. The development of the tube mill in connection with the stamp mill in South African practice is extraordinary; the combination effecting an increase in tons crushed per stamp from five tons, before the tube mill was introduced, to 20 tons per stamp when assisted by the tube mill.

The better understanding of the preparation of pulps previous to table treatment as well as the better understanding of the concentrating tables themselves has brought about great improvement in mill recoveries.

The development of electromagnetic and electrostatic concentrating machinery has established practical separation of iron and zinc minerals which before was most difficult or impossible.

Most interesting use has been made of the property of film tension in liquids for the purpose of separating mineral values from the attending gangue. In general the sulphides of the metals may be floated on the liquid while the gangue sinks. While the system is limited to certain conditions, it opens a fascinating field and one which may develop into great possibilities.

Even greater progress could be accomplished were it possible to obtain the earnest co-operation of mill operators. In some cases splendid triumphs have been accomplished because new devices have been given the "square deal" by intelligent and progressive mill operators. On the other hand, many a mill system is undeveloped on account of personal prejudice and lack of initiative.

Failures in mill systems are often due to neglect in making tests to determine the points where losses occur or the costs of various operations. Changes suddenly occur in the ore and if not promptly followed up with required changes in the mill system will result in serious losses.

One of the modern requirements is thorough sampling and the best mills make use of automatic sampling throughout the system.

It is most important to have the right spirit prevail among the members of the mill crew. Good team work tells here as well as in the game, and a fair attitude and attention to improved methods of milling means also thousands of dollars to the mining company.

We often see machinery advertised as "fool-proof." If the mill system is to be run by fools then we need the fool-proof machinery, but if we cannot advance beyond the stage of fool-proof machinery then we might as well give up our cherished ideas of additional recoveries in ore values. For the economical recovery of values in low-grade ores we must expect greater complication in mill systems, but if we can save $25,000 by the addition of some machine, even though not classified as "fool-proof," is not the expenditure warranted of $2,500 in wages to some millman who will be a friend to the machine?

The man who stands by the machine has the opportunity of discovering many points, possibly improvements, regarding the operation or results of that machine. If the personal equation of that man is plus and he receives proper recognition by the management, the result is a substantial increment to favorable mill devolpment. Mills are usually located in remote sites, difficult of access and in many cases at altitudes of 10,000 and 12,000 feet above seat level. It requires some financial incentive to induce good men to work constantly under these conditions and liberal wages must be offered to establish a good order of intelligence in the mill crew.

The study of mill systems has in some cases shown a high order of inventive ability among mill superintendents in the development of devices for increasing the recovery of ore values, or reducing the expense of operation. There is, however, a great amount of blind prejudice to overcome when an attempt is made to introduce improved methods in the milling systems. This is a strange condition, for one would think that any millman would be favorable to any device which would show improved results even at the expense of some extra attention, but the fact remains that it requires a bitter fight to establish new systems.

A very interesting psychological fact exists in mill conditions. If a new device or system be introduced by some one of the mill attendants and he has the care of the device, that device is certain to work successfully. Should a system be introduced by the manufacturer, even though such a system may have been proven a success, all manner of prejudice and resistance appears and a well-merited system is classed as a failure.

If our graduates of technical schools could be induced to serve a few years of apprenticeship in the splendid experience of these mill systems and devote their technical training toward assistance in the development of improved devices, instead of obstructionary measures, there would be good progress made for better ore dressing systems.

With the successful treatment of low-grade ores there must be better study of conditions, better skill in operation and a constant effort for improvement. The modern mill calls for a higher order of materials of construction; concrete and steel are now being used when a few years ago such use was designated as extravagance. Better machinery is being used and the use of higher grade materials throughout the mill construction is considered a necessity.

The opportunity for improved results in ore dressing systems is great. When we know that in the operation of 10,000-ton copper ore concentrators, the loss on every ton of ore treated is 30 per cent to 40 per cent; that in precious metal wet concentrating mills the loss is from 20 per cent to 30 per cent, the time is now at hand when the data from long lines of investigation assisting the engineer and the builder, added to the practical application of the honest millman, should produce some telling results.

The trail of the gold seeker leads along difficult paths and through discouraging obstacles. Constant search and persistent effort will be the procedure as long as there remains a quest for precious metals—but the problems call for work and work develops the man.

"Let not him that seeketh cease from his search until he find, and when he finds he shall wonder; wondering, he shall reach the Kingdom, and when he reaches the Kingdom he shall have rest."

Denver Engineering Works, Denver, Colo.

Notes on the Analysis of Tungsten Ores.

By Hugh F. Watts.

Insomuch as tungsten can be readily separated from other elements by precipitation as yellow oxide or as mercurous tungstate, and as these precipitates can be easily converted into weighable tungstic oxide, few methods depending on other reactions have been proposed for its gravimetric determination in ores.

Volumetric methods have not come into general use principally for the reason that most of the time consumed in making the assay is in getting the ore decomposed and the tungsten separated from iron and silica. Solution and titration of the tungstic acid would not, therefore, present any advantages over weighing.

An interesting method described by Ekeley and Kendall (*West. Chem. & Met., Jan.*, 1908) consists in dissolving the tungstic acid from the silica by means of a measured volume of standard ethylamine and titrating back the excess with standard oxalic acid using phenolphthalein as indicator. The method has given excellent results.

Other volumetric methods depend on the principle that tungstic acid can be dissolved in standard sodium hydrate and the excess titrated with standard acid.

For the purpose of analysis, tungsten ores can be decomposed by fusion with alkaline carbonates; by boiling in solution of sodium hydroxide; by digestion in a mixture of hydrochloric and hydrofluoric acids, or by digesting in aqua regia.

When decomposition has been effected by fusion or digestion with alkalis, it is essential that the tungsten be precipitated as mercurous tungstate. A method given in many text-books depending on the separation of the yellow oxide from solutions of alkaline tungstates by evaporation with hydrochloric acid is unreliable, as it seems to be quite impossible to recover all the tungsten in this manner even when the evaporation and boiling with acid is repeated many times.

When the separation is made by mercurous nitrate, it should, of course, be remembered that phosphorus, vanadium, molybdenum, etc., are also precipitated. Moreover, even though these interfering elements are absent, as is generally the case, the voluminous precipitate of mercurous tungstate is apt to include impurities which cannot be easily found and accounted for, so that the results are apt to be a little high. Nevertheless, this is an excellent and much used method and is probably the only one that can be used on some ores.

Hutchin and Tonks (Inst. of Min. & Met., London, May, 1909) have described a method of especial use for battery pulp and tailings, the decomposition being effected by hot caustic alkali solution. The charge taken may be from 5 grams upward, which is digested in a porcelain dish with 25 cc of a 25 per cent solution of caustic soda. From this point the assay is finished as though the decomposition had been made by fusion. The method is not applicable to ores containing scheelite.

A mixture of equal parts of hydrochloric and hydrofluoric acids will decompose tungsten ores, sometimes to an almost complete solution. If now an excess of hydrochloric acid is added and the hydrofluoric acid expelled by evaporation, the yellow oxide will be precipitated on the addition of water. (Method of O. P. Fritchle.)

Conflicting statements can be found regarding the solubility of tungstic acid in hydrochloric acid. The unignited precipitate is certainly soluble to considerable extent in strong hydrochloric acid. Upon dilution with several volumes of water the tungsten is completely precipitated on standing. If subjected to filtration with the proper precautions in washing and the filtrate is perfectly clear and bright, no tungsten can be found therein or at the most in but very slight amounts.

Tungstic acid cannot be washed with pure water as it forms a pseudo-solution and causes the filtrate to be turbid. The wash water must contain acid or a solution of some salt. Similarly, in separating tungstic acid from silica with ammonia, much silica is carried through in a turbid stream when ammonia water is used alone. The presence of a little ammonium chloride prevents this in a great measure, but not to the extent, however, of making a complete separation, as silica is appreciably soluble in ammonia. The use of ammonia is, nevertheless, a most convenient means of separating all the tungstic acid from the bulk of the silicious residue.

When it is expected to use ammonia in the separation, care must be taken not to carry the assay to hard dryness as the tungstic acid is thus rendered difficultly soluble. Even prolonged heating on the water bath makes its solution uncertain.

Tungstic acid and silica may be separated by evaporation with hydrofluoric acid, the addition of sulphuric acid being unnecessary.

A Bunsen or Teclu burner can be safely used to ignite tungstic acid for weighing. The use of a blast lamp is inadmissible since tungstic acid is slowly but steadily volatilized at such temperatures. (Wells and Metzger, *J. Am. Chem. Soc.*, Vol. 23, No. 5.)

When working with material so intractable as tungsten ores, whatever method of analysis is adopted, it is absolutely essential that the sample be finely agate-ground.

Having regard for some of the foregoing principles, the methods in practice are carried on about as follows:

Fusion Method.

From ½ to 1 gram of the sample is thoroughly fused in a platinum crucible with 4 or 5 grams of sodium carbonate. The melt is dissolved in hot water, a little sodium peroxide added, and the iron and manganese separated by filtration. The tungsten passes into the filtrate as sodium tungstate.

Acidify the filtrate slightly with nitric acid, make slightly alkaline with ammonia, boil and filter from the usually slight precipitate. To the filtrate add a few drops of methyl orange and titrate with nitric acid, contained in a burette, until just acid. Heat and add an excess of a clear solution of mercurous nitrate. Now add a few drops of a strong solution of sodium carbonate to precipitate a little mercurous carbonate. Remove from the heat and allow to settle for a few minutes. Filter on a 12-cm filter paper and wash well with hot water.

Without drying the precipitate further than can be done on the filter pump, transfer it with the filter to a platinum crucible. Char the paper at a low heat and finally ignite, in a good hood, over the flame of a Teclu burner. Cool the crucible, moisten the contents well with hydrofluoric acid, dry, reignite and weigh as tungstic oxide.

Aqua Regia Method.

Usually 1 gram of ore is used for analysis. If extremely high grade or a concentrate, ½ gram is sufficient, while for mill tailings, etc., 2 grams or more may be taken. For high-grade ores 50 cc aqua regia are used, for low grade 30 cc. The decompositions are made in 4-oz. covered beakers. The degree of heat should be such that there is a gentle action of the acid on the mineral, but the solution should not actually boil. The assay should be occasionally well stirred up to prevent caking. When the solution has evaporated to 10 cc or 15 cc it is removed from the hot plate and diluted with 50 cc hot water and allowed to stand for half an hour. Decant the clear solution through a good filter paper. The residue settles readily and can be decanted close. Wash the residue twice more by decantation, using each time 50 cc hot water, which has been acidulated with hydrochloric acid, keeping as much of the residue in the beaker as possible.

Add about 20 cc ammonia solution to the beaker to dissolve

the tungstic acid and decant the solution through the filter. The ammonia wash solution is made by mixing 200 cc strong ammonia with 1000 cc distilled water and adding 10 cc hydrochloric acid to form a little ammonium chloride. The residue of silica is now brought on the filter and washed thoroughly with the ammonia solution, which is most conveniently used in a wash bottle.

The resulting silica should be white; if it shows any particles of undecomposed mineral, as may sometimes occur in the high-grade ores, from the protecting action of the separated tungstic acid, instead of bringing on the filter, wash two or three times by decantation and treat again with aqua regia, using 15 cc or 20 cc, which will generally complete the work in a short time. One trial is all that is usually necessary with the ordinary run of ores.

The ammoniacal solution is now evaporated to dryness in a weighed platinum dish, ignited gently at first to drive off ammonium salts and finally at the full power of the burner. Cool, moisten with hydrofluoric acid and again evaporate to dryness and ignite. Weigh as tungstic acid.

The aqua regia method is the one used in the writer's laboratory for the valuation of ores from the Boulder district and has been found to work well on any of our western ores. Two time-saving devices are used to facilitate the work, a McKenna mechanical agate grinder, which is a most efficient machine, and a special air bath for evaporating the ammoniacal solutions, constructed so that each dish has its own receptacle. Each receptacle is heated from beneath with a small Bunsen burner and overhead is furnished with a glass tube through which a current of air is made to impinge on the surface of the liquid. When removing a dish the tube may be turned to one side out of the way.

The time that this arrangement saves by hastening the evaporation is very material, and insomuch as the evaporation can be started while the washing is still in progress, this part of the operation is really very short. The air is furnished from the blast arrangement of a Richard's filter pump.

Tungsten ores are paid for by the unit of tungstic acid, consequently the weight derived from the assay gives at once the figure without calculation further than reduction to percentage. If metallic tungsten is wanted the factor is 0.7931.

A simple qualitative test can be made by boiling a little of the finely ground mineral with 10 cc strong sulphuric acid in a test tube. When cold the liquid assumes a deep blue color. The addition of tin or other reducing agent is needless.

The papers cited in these notes will be found of value to any one interested in tungsten analysis.

Boulder, Colo.

Progress in Mexican Metallurgy During the Century of Independence.*

By A. Grothe.

On reaching the patio we find a scene vastly different from that of the morning. A large cavalcade has arrived and the dismounted riders are busy examining the numbered ore heaps. It is the day appointed for the ore sale or "rescate" and the visitors are the owners or managers of the many reduction works in the valley near town, all intent upon purchasing ore for their "haciendas." Notebook in hand they proceed from heap to heap, judging the weight by the eye and the contents by panning in a small lacquered gourd (jicara). They are wonderfully expert at the operation (tentadura) when dealing with ore of which they have experience.

The art of assaying is hardly practised as yet (any more than that of mine surveying) and in consequence grievous mistakes occur occasionally; as, for instance, when some years later ore was found in abandoned workings in the upper levels of Valenciana which had the look of compact iron pyrites and gave no indication of silver in the tentadura. Nevertheless it was found by fire assay to contain from 22 to 24 marcos of silver per "montón" (3.5 to 4 kg per ton). Still these mistakes are the exception, and ordinarily the estimate of weight and contents is astonishingly correct.

When all are ready with their calculations the manager of the mine (administrador) takes his position near one of the heaps and invites bids for this number. The competing buyers approach him one by one and make their bids in whispers. The proprietor of the mine or his representative do the same as a protection against possible collusion among the rescatadores. When after the auctioneer's solemn: "No hay quien diga más?" no further offer is forthcoming, the name of the highest bidder and the price to be paid are announced, and the group moves on to the next montón until all are sold. In this manner thousands of dollars worth of ore are sold every week. Losses are made, and great bargains secured, and it seems that this gambling nature of the method makes it so attractive to those interested that it has survived in Guanajuato to until the end of the last century, when the San Cayetano Company still sold the production of its mines in this manner. And, crude as the system was, perhaps no great injustice resulted to either buyer or seller.

The writer had the ore offered for sale by those mines in 1888 carefully weighed and assayed and found that at the end of the year the computation of the value on this correct basis differed only 2 per cent from the price realized, the latter being the higher. He thereupon published his figures on the day before the rescate and sold the ore on the basis thereof. This put an end to the picturesque but risky proceeding of former times.

At the conclusion of the rescate the mules which have to convey the ore to the reduction works swarm in, and the arrieros proceed to load the ore in leather sacks, while their masters sit down to a sumptuous lunch, after which the tables are cleared for a friendly game of cards, which is often prolonged far into the night, in the course of which many of the purchased heaps of ore change hands.

Not being possessed of the gambling spirit, we shall rather follow the ore to the reduction works.

At the hacienda it is weighed, without, however, a determination of the moisture being made. This and other inaccuracies make the returns as to results obtained not quite so trustworthy as might be desirable. The coarse ore (grueso) is then crushed to pea size in crude wooden stamps or equally primitive roller mills with stone runners, and, after being mixed with the smalls, is fed to arrastres, where it is ground to impalpable slime under addition of water.

The arrastre is a circular stone tub of about 9 ft. diameter half sunk below the floor and provided with a bottom of hard basalt or porphyry, the stones being placed on end. The grinding stones are large pieces of the same material suspended by chains from cross-bars fixed to a vertical shaft, to which a mule imports a rotary motion. Eighty or more arrastres are placed in a row in a large shed or galera. The bottom of the arrastre is covered with a thin layer of silver or copper amalgam to which the fine particles of metallic silver and gold contained in the ore adhere.

Every morning the arrastres are emptied of the ground pulp, which goes to the "patio" for further treatment. The capacity of an arrastre is from 7 to 8 cargas (1100 to 1300 kg) per day, according to the degree of softness of the ore. Every two to three months a "raspa" is made; that is, the amalgam is removed from the bottom and joints. The scrapings are thinned

*In the days of rapid progress in the twentieth century it is difficult to recall or imagine some of the conditions which existed a century ago, or the crude methods then employed for the valuation and treatment of ore. At a recent meeting of the *Concurso Cientifico y Artistico del Centenario*, held in Mexico on Feb. 25, 1911, Mr. Albert Grothe represented the Mexican Institute of Mining and Metallurgy, and read a paper showing the progress of metallurgy in Mexico during the past century. In a delightfully interesting manner he contrasts the conditions existing at the Mellado and Rayas mine at Guanajuato in the year 1809 with those of the present day. His full address, of which the following is an abstract, appears in the *Bulletin* of the Mexican Institute of Mining and Metallurgy, Vol. 2, No. 2.

out with mercury, filtered and the remaining mercury distilled off in vertical iron retorts or "capellinas." The resulting bullion, which is very rich in gold, is then melted in charcoal furnaces, cast into bars and taken to the mint, where it is assayed and parted—that is, the two metals are separated, and obtained in an almost pure state. By this process about one-half of the gold contained in the ore is recovered, the other half remaining in the pulp is lost, as we shall see.

The pulp is collected in flat heaps, called "tortas," each containing about 100 tons of ore, on the flagged floor of an open yard, called the "patio," from which the name of the whole process is derived. It was invented by Bartolomé de Medina and first practised by him in Pachuca in 1557. It was admirably adapted to the available resources and the requirements of the country, and, crude as it may appear to the modern metallurgist, has been of incalculable benefit to the mining industry.

The proper proportion of common salt is then added to the torta and thoroughly mixed with the pulp (repaso) by the action of twenty-four horses held on lines by the "tortero," round whom they circle, while he himself continually changes his position so that all parts of the torta are eventually reached. Before 1793 this work was done by hand.

The next operation consists in the admixture of "magistral," an impure sulphate of copper obtained by the calcination of copper pyrites containing variable quantities of iron. Lastly the torta is incorporated—that is, mercury is spread over its surface in a fine spray produced by filtration through porous cloth. After each addition of ingredients, and afterwards every other day, a repaso is given.

In from twenty to thirty days, according to the size of the torta and the weather, the chemical action is complete and the torta is said to be "rendida." The silver and gold have combined with the mercury to fluid amalgam, from which the pulp is washed by a liberal addition of water and stirring in large cylindrical vats. It is now only necessary to filter the excess of mercury off the solid amalgam, distill the latter in the manner described before the gold amalgam, and melt.

The product is very pure. Its fineness is determined in the mint and expressed in thousands of the gross weight. Impurities are deducted and it should be noted that if gold is present in less proportion than three-one-thousandths it is not only not paid for, but even deducted from the silver as an impurity. Small wonder that Mexican pesos of this period from the gold-bearing districts are subjected, at a good profit, to parting in London and Paris.

The patio process is a purely empirical one and requires great practice on the part of those who carry it out. It would lead us too far were we to attempt a detailed description of all the operations in connection with it, or to mention the various vicissitudes to which a torta may become subject, and the means for diagnosing and correcting them. The only guide for the "azoguero" is that afforded by frequent tentaduras.[1]

From time to time improvements, based upon chance discoveries, have been introduced and local modifications demanded by peculiarities of the ore have been made, but the history of the process is free from the often absurd nostrums frequently used during the earlier times of Californian panamalgamation. Altogether Medina's discovery was an admirable one and much more perfect than the mining methods in use at the beginning of the nineteenth century, which have been described above, for no other process until the advent of cyaniding has been able to oust it.

We have tried to produce an impressionist picture in broad traits of the methods employed in Mexican mining and metallurgy at the beginning of the century, in the years immediately preceding the outbreak of the revolution which ultimately liberated the country from the oppression under which it had been ground for more than two centuries and a half. It was the most productive period of Mexican mining, with which the general prosperity of the whole country had naturally kept pace.

Another discovery which gave an immense impulse to Mexican mining was the method of extraction of the precious metals from their ores by means of weak solutions of potassium cyanide. There has been much controversy over the question as to who was the real discoverer of the principle underlying this operation. We shall not enter into it here. Uncontrovertible, however, is the fact that since the introduction of the process by the McArthur-Forrest interests millions of tons of ore which formerly had to be considered as of no value can be treated profitably, in consequence of which the economic conditions of the mining industry have been radically changed and much foreign capital has been attracted.

Mexico was somewhat slow in adopting this method, possibly on account of the fact that as at first practised it was more applicable to gold than to silver ores.[2]

In our description of the process we can be much more concise than we were in that of the earlier methods, if for no other reason than that Rayas in 1808 is immeasurably farther from us than, for instance, Pachuca in 1911. Any of the modern reduction works there can be readily visited by those desiring some knowledge of this newest branch of metallurgy of the precious metals. Such a visit will afford convincing evidence of the great importance of this method of ore treatment and will show that, while Mexico entered the field long after other countries, once the start was made it has not only kept pace with them, but, certainly as regards the treatment of silver ores, is ahead of most.

Briefly, the process is carried out as follows: As in any other wet method of extraction, the ore is finely triturated in stamps, Chilean mills and tube mills, cyanide solutions being added from the beginning. The separation of sands from slimes, which a few years ago was considered indispensable to successful cyanide practice, has been superseded in all modern plants of this country by the "all-sliming" method, in which the sand is all ground as fine as possible, or, rather, as fine as necessary to obtain the best economic results.

The pulp, consisting of finely ground ore and cyanide solution, is then agitated by means of mechanical stirrers or by compressed air, the latter now being the rule in Pachuca When the cyanide has completed its chemical action on the gold and silver of the ore, these substances are found in solution as double salts with sodium cyanides. This solution must now be separated from the worthless tailings by means of vacuum or of pressure filters of various construction, of which new forms are almost daily brought out. The clear, filtered solutions are then brought into contact with zinc shavings or zinc dust, on which the precious metals are precipitated at the expense of the zinc.

The solutions, after having been brought to their original strength by the addition of fresh sodium cyanide, enter again into circulation by being returned to the grinding apparatus. The black precipitate obtained by the operation, consisting chiefly of gold, silver, finely divided zinc, lime, alumina and other impurities, is then washed in filter presses, dried, melted with suitable fluxes, and finally cast into bars of about 30 kg, which are exported. The product is not of the same purity as that from the patio process, rarely being above 950 fine, but, on the other hand, the extraction is above 90 per cent of the silver and 98 per cent of the gold, and the process requires only as many days as formerly weeks, and the cost is less than one-fourth.

A description of the mechanical details of the appliances used in the cyanide process would fill volumes and not be pertinent

[1] The old system of tentaduras as a guide in the patio process was gradually replaced by assays of the amalgam, proposed in 1875 or 1876 by Manuel Maria Contreras, M. E. The art of making correct tentaduras has well-nigh been lost among the younger generation of engineers.

[2] In 1894 the McArthur-Forrest Company installed a movable plant for the treatment of tailings in the State of Sonora which they shifted from one point to another. This was followed by stationary plants in Guanajuato, El Oro, Pachuca and other mining centers. In Pachuca the new plants followed each other rapidly. The last mill which has been reformed is that of Purisima Grande, where centuries ago the system of Bartolomé de Medina had originated and where it was practised until very recently.

to the purpose of this paper, which is to draw attention to the great and beneficent change in methods since the time when Mexico obtained its independence, especially during the last thirty years, when continued peace made it possible to devote all the available energy to the development of the industry.

It must be noted that since the introduction of cyaniding all efforts have been directed toward improvements on mechanical lines, in which certainly wonderful progress has been made. Our knowledge of its chemistry has, however, not advanced at the same rate and is still very much as it was when McArthur-Forrest first advocated and practised the principle involved, with the important exception, however, that it was found that in the treatment of silver ores the extremely weak solutions used on gold have hardly any effect, but have to be considerably strengthened, a fact which was not known at that time. Before this was known successful treatment of silver ores by cyanide was considered impracticable.

Various other chemicals in combination with cyanide have been proposed and tried, as they were supposed to accelerate or intensify the action, but the largest number of them have no effect or are only of use with certain classes of ore. Mention should, however, be made of Clancy's proposal to add to the straight cyanide solution calcium cyanamide, an alkaline sulphocyanide and an alkaline iodide, and passing a low-voltage current through the agitated pulp. The process has not as yet been fully tried on our Mexican ores, refractory on account of containing part of the silver in combination with manganese.

If successful on these, many ores not amenable to straight cyanide treatment could be profitably worked in this manner. According to the experiments of the inventor even in ordinary ores a saving of cyanide is obtained as the solutions are regenerated at the expense of a less costly substance and a trifling expenditure of electrical power. If the claims made for this process are substantiated by results on a working scale, new prospects for the industry will doubtless be opened.

We approach the end of our task, which was to indicate in broad lines the difference between the methods in use at the beginning of the century of independence and those now practised. Some of the manifest faults of our sketch are due to the disproportion between the space at our disposal and the great mass of material which imperatively seemed to demand consideration. This caused a constant conflict between the desire for completeness and the necessity for brevity, which perforce made the work fragmentary. We have dwelt more minutely on the early part of the period under consideration than on more recent occurrences. The knowledge of these early times is rapidly being obliterated; in none of the splendid and valuable treatises on old Mexican mining history is it presented in a connected form, being generally overlaid by a wealth of statistics and detail, which to the non-technical reader is not calculated to increase clearness.

We are safe in concluding with the statement that, especially during the period of peace and security of the last thirty years, the progress has been enormous and in expressing the hope and conviction that under a continuance of these conditions the industry will gain in importance at the rate justified by the large mineral wealth of the country.

Asbestos of Commerce.

The asbestos of commerce includes fibrous minerals of several species, but all may be included under amphibole and serpentine. Fibrous amphibole is properly called asbestos, fibrous serpentine is chrysotile, and the especially fine silky form of both amphibole and serpentine is known as amianthus, but in the trade all are included under the general term asbestos. Like auriferous quartz and the ores of other precious metals, asbestos generally forms veins, but in some places where it is of low grade it makes up the whole mass of the country rock. The three types of asbestos, cross, slip and mass fiber, are distinguished by the form of aggregation.

Treatment of the Anode Sludge in Electrolytic Refineries.

From lecture notes prepared for the School of Mines, Columbia University, by Edward F. Kern, Ph.D.

Anode sludge is the residue of the anode, formed during the electrolytic refining of metals. It consists mainly of the associated impurities which stand lower in the E.M.F. series than the metal which is being refined.

Analyses of copper anode sludge vary between:

15.0 to 55.0% Cu (as Cu: Cu_2S: $CuSO_4$: Cu_2Se: Cu_2Te: etc.)
2.0 to 8% Sb (as Sb_2O_3: $(SbO)_2SO_4$: $Cu_2O.Sb_2O_6$: etc.)
1.5 to 6.0% As (as $As_2O_3.H_2O$: $Cu_2O.As_2O_6$: etc.)
0.2 to 8.0% Bi (as Bi_2O_3: $(BiO)_2SO_4$).
0.5 to 6.0% Pb (as $PbSO_4$).
0.5 to 12.0% S (as Cu_2S).
0.5 to 1.5% Fe (as $FeO.FeSO_4$).
0.1 to 2.5% Se (as Ag_2Se: Cu_2Se).
0.1 to 3.5% Te (as Ag_2Te: Cu_2Te).
5.0 to 50.0% Ag (as Ag: Ag_2Se: Ag_2Te).—1,500 to 15,000 oz/T.)
0.017 to 0.70% Au (as Au) = (5 to 10 oz/T).

Typical analyses of sludge from Montana copper:

40.% Ag (12,000 oz/T): 2.% Au (600 oz/T): 25% Cu:
5.% Se & Te: 10% As + Sb: 18% $PbSO_4$, S etc.

Analyses of lead anode sludge vary between:

6. to 18% Pb (as Pb, PbSe, PbTe, oxide).
5. to 25% Cu (as Cu, Cu_2Se, Cu_2Te, oxide)
5. to 70% Sb (as Sb or Sb_2O_3).
2. to 50% As (as As or As_2O_3).
0. to 20% Bi (as Bi or Bi_2O_3).
0. to 2% Sn (as Sn or SnO_2).

} The OXIDES formed during drying of the SLUDGE.

0. to 3% Se (as Cu_2Se, Ag_2Se, etc.).
0. to 4% Te (as Cu_2Te, Ag_2Te, etc.).
10. to 40.% Ag (as Ag, Ag_2Se, Ag_2Te, etc.) = (300 to 12,000 oz/T).
0. to 1% Au (as Au) = (0 to 300 oz/T).

Typical analyses of sludge from trail, B.C., lead:

32% Ag (9,600 oz/T): 0.3% Au (90 oz/T): 9.0% Cu:
27% Sb: 7.% As: 14.% Pb: 11.% moisture, O., etc., and
19.% Ag (5,700 oz/T): 0.1% Au (30 oz/T): 11.% Cu:
31.% Sb: 9.% As: 16.% Pb: 14.% moisture, O. etc.

A characteristic of lead anode sludge is that the constituents are present principally as metals or oxides, whereas the constituents of copper anode sludge are present principally as basic sulphates, sulphates and oxides.

A reason that basic sulphates are the predominant constituents of copper anode sludge is that the principal impurities (Pb, Sb and Bi) of crude copper stand above copper in the E.M.F. series, so are dissolved along with copper, but forming stable sulphates and basic sulphates they are precipitated and collect in the sludge as such.

The impurities of base bullion (crude lead) stand below lead in the E.M.F. series, they, therefore, are not attacked, but remain as metals, forming the sludge. The oxides form during drying.

Preparing Anode Sludge for Treatment.

The sludge, on being received at the "silver house," is first poured through a copper screen (about 10-20 mesh) in order to remove any nodules, or crystals, of metal. The screenings are washed and returned to the refinery, and melted down with the next lot of metal to be cast into anodes.

In case the material is copper anode sludge, after it is screened, it is poured upon a filter of muslin and filtered by suction or put through a filter-press to remove excess of electrolyte, which is returned to the refinery. The filtered sludge is washed with water, well drained, placed in sheet steel pans and dried in an oven. When dry, it is sampled and the sample sent to the assay office. The dry sludge is then treated for metal content by one of the following methods.

If the material is lead anode sludge, after it is screened, it is well washed with hot water by decantation, then filtered by suction on a muslin filter. The solution and wash water is evaporated in copper pans to about 1.15 Sp.Gr. and returned to the refinery. The washed sludge is placed in pans and dried and partially oxidized in an oven, and finally treated by one of the following methods.

Treatments.

A. Direct Cupellation: (Method followed when copper refinery is run in connection with the Parkes' or Pattinson's desilverizing plant). Used for treatment of copper anode sludge: The dry sludge placed in heavy paper bags holding 10 to 15 lb. and charged on the molten metal in a cupelling furnace. The Cu, Pb and Bi are oxidized and collect with the litharge on the surface of the metal as scoria, which is skimmed off and sent to smelting furnace, where added along with regular furnace charge. The As, Sb, S, Se & Te are oxidized and volatilized, and partially enter the scoria. Difficult to remove all of the Se & Te from the resulting Doré bullion.

The Doré bullion, assaying 900+ fine, is cast into suitable slabs, and parted either with boiling concentrated H_2SO_4, or else by electrolysis.

B. Fusion Methods.—For treatment of copper and lead anode sludge: The washed and dried sludge is mixed with a flux of soda ash (crude Na_2CO_3) and small amount of KNO_3 and smelted in a basic lined (magnesite) reverberatory furnace, producing a slag which contains the Sb, As, Se, Te, S and most of the Cu and Bi. The slag is sent to the ore smelting furnace and added along with regular charge. The bullion remaining on the hearth is cupelled to Doré bullion (alloy of Ag, Au, Cu, etc.), and the resulting scoria of Bi_2O_3, PbO, Cu_2O, etc., is skimmed and sent to the smelting furnace. The Doré, assaying 900 to 975 fine, is cast into suitable plates and parted either with hot conc. H_2SO_4 or by electrolysis.

C. Fractional Scorification in Basic Lined Reverberatory Furnaces.—For sludge which contains small amount of copper, such as electrolytic lead refining sludge. The sludge, after washing and drying, is scorified (oxidizing atmosphere above melting point) in the "antimony slagging" furnace till the arsenic and antimony are mostly removed. The scoria is skimmed and subsequently treated in another furnace for the production of hard-lead. The metal is withdrawn from the scorifying furnace and placed in a second basic lined furnace and rescorified for the removal of the bismuth as Bi_2O_3. When most of the bismuth is scorified, the scoria is completely skimmed and subsequently treated for the bismuth by chemical leaching methods. The metal remaining in the furnace is withdrawn and finally rescorified in a third basic lined furnace to remove the remaining lead and most of the copper, leaving Doré bullion. The lead-copper oxide scoria is smelted along with the regular charges in a lead blast furnace, recovering the copper as matte and the lead as base bullion. The Doré bullion is cast into thin plates, and parted, either with concentrated H_2SO_4 or electrolytically.

When high grade bullion is treated by either of the above methods, precautions must be taken to *collect the flue dust.* This is done by passing the gases through sheet-steel flues 100 to 350 ft. long, then through a series of dust chambers and finally catching the finely suspended particles by filtering the gases through long flannel bags, suction draft being used, or else passing into spray-towers, or under water.

The flue-dust from these furnaces is returned to the smelting furnaces, or else treated the same as sludge. If much As_2O_3 present in the flue dust, it is recovered by subliming twice in a muffle furnace or in a reverberatory furnace using coke fuel. The residue is mixed with next treatment of sludge.

In some plants, where the treatment is carried out by either A. or C., the scoria removed during the latter part of the treatment runs high in silver, so is kept separate from that which is produced during the first part of the operation. The first scorias, in this case, are sent to the smelting furnace, and that which runs high in silver is added along with the next charge of sludge to the furnace, and its silver content collected.

D. Direct Parting with Boiling Conc. H_2SO_4.—Not used at present, as much acid is lost due to formation of SO_2. (Not suitable for sludge containing much As or Sb).

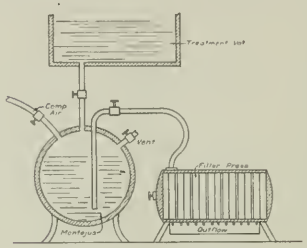
$Cu + 2H_2SO_4 = CuSO_4 + SO_2 + 2H_2O.$

$Ag_2 + 2H_2SO_4 = Ag_2SO_4 + SO_2 + 2H_2O.$

$M + 2H_2SO_4 = MSO_4 + SO_2 + 2H_2O.$

This treatment is similar to the parting of Doré bullion with boiling conc. H_2SO_4 in iron kettles.

E. Mix the washed and dried sludge with a calculated amount of conc. sulphuric acid (sufficient to dissolve the copper, antimony, etc., but not enough to dissolve the silver), forming a thick mud. It is then heated in iron kettles till SO_2 is expelled, and silver begins to dissolve. A small amount of fresh sludge is added to precipitate the silver, then the residue is leached with water to remove the sulphates. The solution is treated by the usual method of crystallization, or by passing over iron, for recovery of the metals. The residue after leaching is scorified to Doré bullion, which is finally parted. The scoria is smelted in a lead smelting furnace along with regular charge.



FILTER PRESS AND MONTEJUS.

The cement metal, formed by precipitation of the sulphate solutions with iron, is smelted for its metal content.

F. Treatment with Hot Dil. H_2SO_4, Air and KNO_3.—The washed sludge is placed in a lead lined vat containing dilute H_2SO_4 (1 acid : 2 + water), and heated to boiling, and agitated by means of injected steam and air, and hard-lead stirrers. A small amount of KNO_3 (saltpeter) is added at intervals in order to assist in the oxidation and more rapid solution of the copper, etc. This treatment lasts from 20 to 30 hours, during which time most all the copper and some Ag is dissolved, and Bi and Sb changed to basic sulphates, and Pb to $PbSO_4$. The Ag in solution is precipitated by suspending strips of copper in the tank at the end of treatment, and collecting it with the residue of Au, Ag, $PbSO_4$, BiO_2, SO_4, $Sb_2O_3SO_4$, etc., or else by adding a small amount of fresh sludge.

The Cabell Whitehead process is a modification of the above treatment, using Ag_2SO_4 residues from the parting kettles in place of KNO_3 to get the Cu, Bi, etc., in solution. Process formerly used at plants where the Doré was parted with boiling conc H_2SO_4.

Reaction.—$Cu + Ag_2SO_4 = CuSO_4 + 2Ag.$

$2Bi + 3Ag_2SO_4 = Bi_2(SO_4)_3 + 6Ag.$

$M + Ag_2SO_4 = MSO_4 + 2Ag.$

In case of Bi and Sb, their sulphates are decomposed, forming insoluble basic sulphates ($Bi_2O_3 . SO_4$ and $Sb_2O_3 . SO_4$).

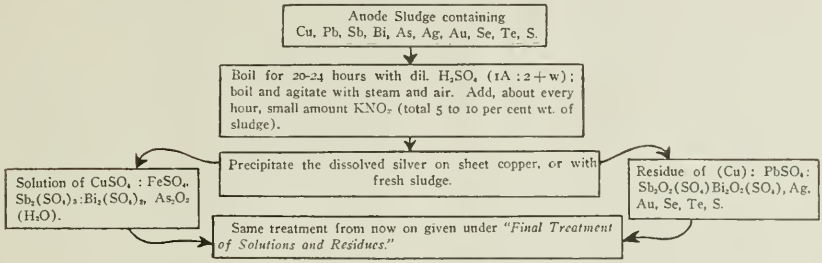
The use of the Ag_2SO_4 residue from the H_2SO_4 parting kettle reduces the time of treatment six to eight hours. These residues consist principally of anhydrous Ag_2SO_4 and $CuSO_4$, which form on the side of the kettle during the parting of Doré bullion.

The vats in which the treatments are carried out are made of heavy pine boards lined with sheet lead. Dilute H_2SO_4 solutions do not dissolve metallic lead after it is coated with a thin layer of $PbSO_4$.

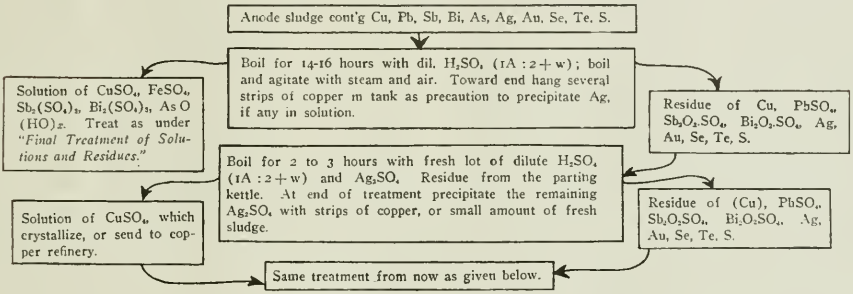
The vats used for evaporating or for boiling *dilute sulphuric acid* solutions are made of hard sheet lead (containing 4 to 6 per cent Sb), the heating being done by means of hard lead coils through which steam is passing.

Tree of Treatment of Anode Sludge with Dilute H_2SO_4 (F).

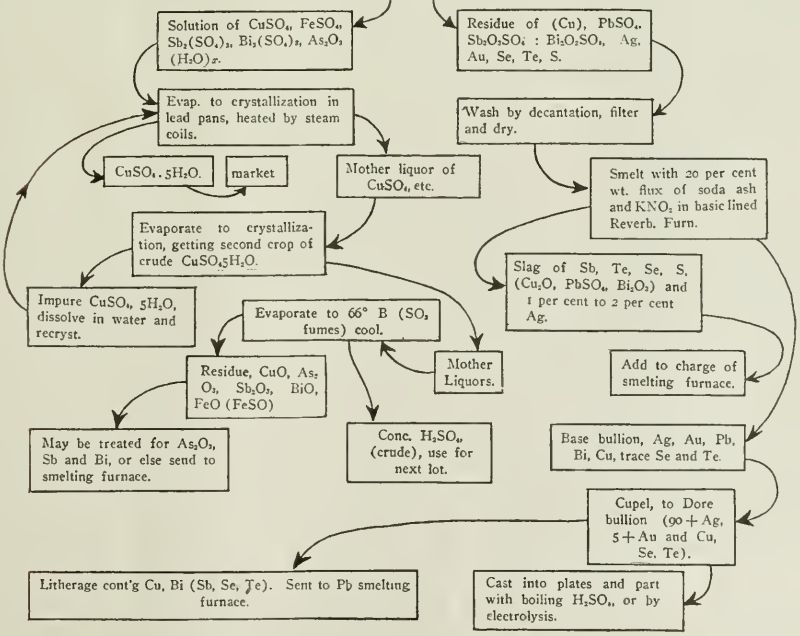
1. *Use of KNO_3 as Oxidizing Agent.—*


Anode Sludge containing Cu, Pb, Sb, Bi, As, Ag, Au, Se, Te, S.
Boil for 20-24 hours with dil. H_2SO_4 (1A : 2 + w); boil and agitate with steam and air. Add, about every hour, small amount KNO_3 (total 5 to 10 per cent wt. of sludge).
Precipitate the dissolved silver on sheet copper, or with fresh sludge.
Solution of $CuSO_4$: $FeSO_4$, $Sb_2(SO_4)_3$: $Bi_2(SO_4)_3$, As_2O_3 (H_2O).
Residue of (Cu): $PbSO_4$: $Sb_2O_2(SO_4)$ $Bi_2O_2(SO_4)$, Ag, Au, Se, Te, S.
Same treatment from now on given under "Final Treatment of Solutions and Residues."

2. *Cobell Whitehead Process—Use of Ag_2SO_4 as Oxidizing Reagent.* (F).


Anode sludge cont'g Cu, Pb, Sb, Bi, As, Ag, Au, Se, Te, S.
Boil for 14-16 hours with dil. H_2SO_4 (1A : 2 + w); boil and agitate with steam and air. Toward end hang several strips of copper in tank as precaution to precipitate Ag, if any in solution.
Solution of $CuSO_4$, $FeSO_4$, $Sb_2(SO_4)_3$, $Bi_2(SO_4)_3$, As O (HO)$_x$. Treat as under "Final Treatment of Solutions and Residues."
Residue of Cu, $PbSO_4$, $Sb_2O_2SO_4$, $Bi_2O_2SO_4$, Ag, Au, Se, Te, S.
Boil for 2 to 3 hours with fresh lot of dilute H_2SO_4 (1A : 2 + w) and Ag_2SO_4. Residue from the parting kettle. At end of treatment precipitate the remaining Ag_2SO_4 with strips of copper, or small amount of fresh sludge.
Solution of $CuSO_4$, which crystallize, or send to copper refinery.
Residue of (Cu), $PbSO_4$, $Sb_2O_2SO_4$, $Bi_2O_2SO_4$, Ag, Au, Se, Te, S.
Same treatment from now as given below.

Final Treatment of Solutions and Residues.


Solution of $CuSO_4$, $FeSO_4$, $Sb_2(SO_4)_3$, $Bi_2(SO_4)_3$, As_2O_3 (H_2O)$_x$.
Residue of (Cu), $PbSO_4$, $Sb_2O_2SO_4$: $Bi_2O_2SO_4$, Ag, Au, Se, Te, S.
Evap. to crystallization in lead pans, heated by steam coils.
Wash by decantation, filter and dry.
$CuSO_4.5H_2O$.
market
Mother liquor of $CuSO_4$, etc.
Smelt with 20 per cent wt. flux of soda ash and KNO_3 in basic lined Reverb. Furn.
Evaporate to crystallization, getting second crop of crude $CuSO_4.5H_2O$.
Slag of Sb, Te, Se, S, (Cu_2O, $PbSO_4$, Bi_2O_3) and 1 per cent to 2 per cent Ag.
Impure $CuSO_4.5H_2O$, dissolve in water and recryst.
Evaporate to 66° B (SO_3 fumes) cool.
Mother Liquors.
Add to charge of smelting furnace.
Residue, CuO, As_2O_3, Sb_2O_3, BiO, FeO (FeSO)
May be treated for As_2O_3, Sb and Bi, or else send to smelting furnace.
Conc. H_2SO_4, (crude), use for next lot.
Base bullion, Ag, Au, Pb, Bi, Cu, trace Se and Te.
Cupel, to Dore bullion (90 + Ag, 5 + Au and Cu, Se, Te).
Litherage cont'g Cu, Bi (Sb, Se, Te). Sent to Pb smelting furnace.
Cast into plates and part with boiling H_2SO_4, or by electrolysis.

The boiling of *concentrated sulphuric acid* is done in cast iron or cast steel pans, heated by flame underneath.

The *filter presses* used for filtering sulphuric acid, or sulphate solutions, are made of hard lead (containing 8 per cent Sb), the filtering medium being closely woven cloth. The solutions are forced through the filters by either pumps or montejus, which are made of iron lined with hard lead. A montejus is an air-tight siphon vessel, used for lifting liquids by means of compressed air. The solutions are run into the montejus by gravity and afterward forced out through a pipe extending to the bottom, the supply pipe being closed. This apparatus is more suitable than a pump, as it has no working parts which are exposed to the acid solutions.

The *crystallizing vats* for $CuSO_4 . 5H_2O$ are made of wood and lined with lead. Strips of lead are suspended in the vat and form the nucleus for the crystals of $CuSO_4 . 5H_2O$. The hot saturated solutions are run into the vat and allowed to cool, giving a crop of crystals, which, if pure, are collected, washed, dried and stored. If the crystals are impure they are dissolved and recrystallized. The final mother liquors are passed over scrap iron to recover their copper content and the solutions run to waste. The cement copper is added to the copper furnace.

The *scorifying furnaces,* in which the acid-treated sludge is brought to Doré bullion, have hearths 5.5 to 5 ft. wide and 6 to 8 ft. long, capacity of 60,000 to 150,000 Troy ounces of Doré per charge. The hearths are made of either magnesite or chromite bricks, which are not corroded by the scoria. The furnaces are usually fired by oil and blast. Ordinarily two charges of Doré bullion are completed and cast per week.

A comparatively cheap, but still very satisfactory, material for making hearths of scorifying and cupelling furnaces is made of dolomite concrete, which is prepared by mixing about four parts of crushed dolomite and one part Portland cement. This is thoroughly mixed with sufficient water to give a good "setting" mass, then tamped into place, allowed to "set," and thoroughly dried by a fire of gradually increasing intensity.

The workmen, before they are allowed to leave the "silver house," are required to change their clothes and wash. The waste wash water from the sinks is run into settling tanks where the slimes collect. The slime is removed from the settling tank at stated intervals and is treated along with the anode sludge. The old cloths are burnt, and the ash treated as sludge.

Notes on Chemistry and Metallurgy in Great Britain.

(Continuation, from our last issue, of the report of the Iron and Steel Institute.)

Mr. J. E. Stead's paper on "Notes on the Welding-Up of Blowholes and Cavities in Steel Ingots" was next read.

Professor Arnold spoke of the remarkable effect of a small proportion of aluminium in preventing blowholes. For many years it had been an annual experiment at Sheffield University to cast two ingots of the same weight, one without aluminium and the other with 0.01 per cent of aluminium, and the latter was entirely free from blowholes.

Mr. S. H. Ridsdale said that hollowness in forgings was often caused by working at too low a temperature in order to save re-heating. He had seen blooms which showed splits in the forged ends and middle, but were proved, by polishing and etching, to be quite sound in the unforged parts which, under proper conditions, yielded thoroughly sound forgings. He could not concur in Mr. Stead's statement that blowholes were least objectionable when well within the surface. Rails with a very solid exterior almost always had a spongy interior, while if the surface were spongy and shelly the inside was almost invariably sound.

Dr. W. Rosenhain remarked that the fact of a hole not reopening was not proof that it had been welded up. The time during which pressure was exerted on the steel was an important factor. There was an inward diffusion of air apart from pressure. He could not accept as proved the author's statement that a coating of iron oxide on the interior of a blowhole was a bar to welding, because he had evidence that where the skin of oxide could be broken through welding might occur.

Mr. W. H. Ellis referred to a case of an internal cavity in a large 30-ton fluid-compressed ingot. Apparently when the walls attained a certain thickness a cavity formed. He had forged down such an ingot, cut it into lengths and found that it welded satisfactorily. He believed that the reason was that the interior of the cavity was free from oxide.

A paper by Messrs. E. F. Law, W. H. Merett and W. Pollard Digby, "Some Studies of Welds," followed. Opening the discussion, Mr. A. E. Tucker said he was anxious that it should not go forth that defects such as were detailed in the paper were insuperable or even serious in practice. Gas welding had already been largely adopted with complete practical success. He produced a sample to illustrate what might be done by cutting steel with the oxy-hydrogen flame in discovering defects in the body of the metal which seemed undiscoverable in any other way. The two surfaces were very interesting. The first, cut with pure oxygen, was but little inferior to one obtained with a rough milling cutter, while the second, cut with a 96 per cent oxygen, was ragged and heavily oxidized.

Professor Turner said the microscope afforded an easy means of determining which process was best adapted for the production of a particular kind of welding. There were defective weldings by the acetylene process or by the resistance process, but that only meant that the operations were not properly carried out. He believed that those welds by modern processes when properly effected were perfectly satisfactory, as was shown by the tensile strength and working properties. At the same time the authors were doing a public service in drawing attention to certain possibilities of failure.

Sir Thomas Wrightson remarked that only certain metals could be welded. The melting of two metals together was not the same thing as welding. Many years ago he had made experiments at the Mint with Roberts-Austen which demonstrated that in the case of iron there was a distinct fall of temperature at the moment of solidification, when the pressure came on, and not an evolution of heat. He mentioned this as an explanation of the welding properties of certain metals.

Mr. H. Le Chatelier (Bessemer Medallist) said the inevitable defects of welds were all known and it would never be possible to entirely obviate them. The most pressing problem of the day was to find some means of ascertaining the quality of a weld without at the same time destroying it. Possibly the electric, magnetic or acoustic properties of the material might afford an index of even internal defects.

Dr. J. E. Stead quite agreed with the authors that it was possible to tell the method of welding by the microscope. His own experience had often confirmed the fact.

The following papers were discussed together: "The Action of Aqueous Solutions of Single and Mixed Electrolytes Upon Iron," by Dr. J. Newton Friend; "The Corrosion of Metals," by Mr. P. Longmuir, and "The Influence of Impurities on the Corrosion of Iron," by Mr. J. Cobb.

Professor Arnold remarked that one interesting point in Mr. Longmuir's paper was the mechanical deterioration of metals as a consequence of corrosion by sulphurous or sulphuric acids. This was a matter deserving attention.

Mr. E. H. Saniter fully appreciated the value of Mr. Longmuir's insistence on the importance of studying the effects of atmospheric corrosion as well as local corrosion set up by electrolytic action. He considered that in many cases laboratory tests on corrosion were quite misleading.

Prof. H. C. H. Carpenter read a paper on "The Growth of Cast Irons after Repeated Heatings." This was a continuation of his paper on the same subject last year.

Mr. J. E. Stead opened the discussion and said he had examined a stove pipe which had been heated for several years at a constant temperature of about 750° C., and the microscopical, physical and chemical results obtained led to the important conclusions which might modify some of Dr. Carpenter's tentative conclusions. Mr. Stead exhibited micrographs of the structure of the pipe before and after heating. Gas penetrated the metal very easily, but the pipe had not been heated above 750° C. and all the pearlite in the area of ferrite was globulized, while the central parts of the dark areas consisted of laminæ of graphite. When broken the cleavage was through the graphite, and, although about 25 per cent of slag was present, the fracture of the pipe was just the same as that of the original pig. He was of opinion that the shape of the sample was a factor in the longitudinal expansion. If the bars had been round different results would have ensued. Dr. Carpenter naturally assumed that silicon steel was more easily oxidized, but steel containing no silicon had been proved to oxidize to the greater extent.

Professor Arnold said that Dr. Carpenter classed as steel a material which was something between steel and pig iron. The material specified with 2.5 per cent of carbon was a special steel which commanded a high price in Sheffield. He suggested that the material Dr. Carpenter proposed to substitute was not cast iron, but a hard and probably brittle steel.

Mr. Saniter was not able to accept Table V. An expansion of 11 per cent in vacuo of one alloy could not be altogether attributed to dissolved gases.

Professor Turner highly appreciated Dr. Carpenter's work and congratulated him on being able to indicate the composition of an alloy which would not expand on heating. An alloy containing 2.66 per cent of carbon with 1.5 per cent of manganese should run quite well. He did not agree with Professor Arnold's remarks as to Sheffield steel, Professor Arnold called it steel, but he called it cast iron for the addition of a little phosphorus gave it almost exactly the composition of pig. The irons experimented on were very complex materials and could be heated to such temperatures as would cause absorption of the pearlite and diffusion and absorption of phosphorus. Some volume changes seemed to result from alteration in the silicon-iron-carbon compounds.

Mr. Edwards considered the author's explanation of growth the only one reconcilable with observed facts. Oxidation could not take place under those conditions. There might be alloys of various compositions evolving different gases, but growth might be due to the nature of the gas and not to its volume.

Mr. Adamson agreed in the author's view of the effect of manganese and phosphorus on growth. He had proved its correctness several years back with castings required to stand high temperatures in working. Dr. Carpenter was, doubtless, right in his contention as to the influence of sulphur under those conditions of test, but when sulphur in the form of acid was brought into contact with iron castings, such as gas retorts, a deposit resembling pyrites resulted, with destruction of the iron. Consequently the less sulphur such iron contained the longer was the working life of the casting. There has not yet been brought to light any proof that such destructive action of sulphur was connected with the growth of cast iron.

In the discussion on the paper by Professor A. McWilliam and E. J. Barnes, "The Influence of 0.2 per cent. Vanadium on Steels of Varying Carbon Content," Mr. J. Kent Smith said vanadium toughened steel by scavenging from it the suboxides and nitrides. The products were fusible and formed with the vanadium a solid solution in ferrite. The great strength of vanadium steel was due to the formation of complex carbides. With regard to cast iron he was able to confidently assert that the scavenging action followed on the use of vanadium. Notable advantages of vanadium cast-iron were high resistance to abrasion and freedom from porosity.

The Metallurgical Future of India.

The development of the natural resources of India and their resulting industries is, and must necessarily be, more a matter of judicious government encouragement than in countries where private enterprise is more in accordance with political conditions. The existence of various intricate systems of established rights renders interference with existing customs a process of grave importance, and one which cannot always be entrusted to a commercial body with more prospect of permanent success than can the public safety be handed over to the care of the half-educated amateur imported politician.

Previous to 1899 the complications caused by a fluctuating exchange made the extension of mining concessions impracticable, the financial return being too uncertain to form a steady inducement to private enterprise. The regulation of the coinage question which came into effect in that year, however, permitted the Indian government to take the matter up and frame regulations which offered more scope for private enterprise.

Some idea of local conditions, and the care which must be exercised in making alteration, however slight, in the usages of centuries, can be gathered from the comprehensive and instructive paper recently read before the Indian section of the Society of Arts by Prof. Sir Thomas H. Holland, formerly director of the Geological Survey of India. Amply qualified as he is to deal with the question from a scientific point of view, Sir Thomas is compelled by his subject to treat occasionally of the social or political aspect of the matter, as in the paragraph on relative safety of European and Indian mines. The superiority of the Bengal mines he attributes, in a great measure, to the fact that the Bengal collier retains, so far, a sense of discipline "that has now been all but destroyed by our education authorities in this country." Possibly a consequence of controlling instruction by those who are most in need of it.

While so many of the finest intelligences are thus engaged in directing the affairs of a portion of the empire which does not govern itself automatically, there is every hope that the mineral resources of the country will develop in conjunction with its other activities.

Taking the products in the order given them by Sir Thomas Holland, coal naturally leads in importance. We learn that during the past 25 years the production of coal has risen from 1¼ million to 12 million tons a year, and still increases, the best being that obtained from the Giridih field. This field is estimated to contain about 70 million tons. The author of the paper considers that a considerable impetus could be given to the industry in Central India by the construction of a broad gage railway extension from Nagpur to Itaisi, which would tap also a rich cotton-producing country, as well as the principal manganese mines.

The coal fields of Assam are at present worked principally for the local market, expense of transport cutting them off from Calcutta, as well as the quality of the coal, which is friable and liable to spontaneous combustion, and thus not easily carried for long distances.

The development of this industry depends very largely upon railway extension, both as providing a market and a means of transport. The available foreign markets, says this paper, are comparatively small, the principal consumption being due to locomotive wants. It seems possible, however, that a market for shipping might be cultivated where the quality of coal permits. At present, and since 1900, the exports of coal largely exceed the imports, which latter have been generally limited to small quantities to meet local and temporary variations in the demand for special coals. The exports have, on an average, been over half a million tons per annum.

With regard to petroleum the difficulties due to ancient customs for a considerable time hindered the development. When

these were overcome the output increased rapidly, rising during the past ten years from 37¾ million gallons to over 233½ million gallons (1909). Certain of the fields, however, show signs of exhaustion, the Twingon, in Upper Burma, for instance. On the other hand, three promising properties remain to be developed in Burma, the Khodaung, Singu and one on the other side of the Irrawaddy.

At present the indigenous production is insufficient to meet the country's requirements in kerosene and petrol, of which 90 million gallons are imported annually.

The production of iron is at present low in the peninsula, only one company operating with favorable results (the Bengal Iron & Steel Company), although another (the Tata Iron & Steel Company) has now been floated with a capital of £1,500,000 sterling to exploit the hematite ore deposits of the Mombhanj State in Orissa. The new works will be within forty-five miles of the principal ore supplies and 180 miles from the collieries. The ore in the Mombhanj is rich, carrying over 60 per cent of iron, and exceptionally easy to work, standing in hills above the general plain. In a period when rich ores were in demand they might thus be worth the attention of European furnaces.

Data of other fields do not seem available, although iron is known to exist, in small quantities at any rate, throughout the peninsula.

Turning to minerals raised chiefly or entirely for export the author finds that gold is only mined in Mysore and at Hutti in the Nizam's dominions. The principal opening for development appears to be on the upper reaches of the Irrawaddy, where dredging operations for alluvial gold have been carried on for the past nine years. The production in 1909 was valued at £32,730.

Manganese is a flourishing industry and appears capable of considerable and rapid expansion whenever the market provides the necessary encouragement, as shown by the output in 1907, which expanded from 571,500 tons to 902,291 under this stimulus. The falling off in steel manufacture and lower prices have since caused the output to drop to 600,000 tons.

In mica the country has a valuable product, which only needs control by organized limited liability companies to become of first-class importance. Even under present circumstances the production is equal to 61.78 per cent of the quantity produced by the three principal mica-bearing countries—India, Canada and the United States. The present methods of winning are hampered by the disadvantages consequent on divided effort and may be regarded as mere prospecting. Numerous deposits still lie untouched.

The further development of India in this connection may bring forward copper as a potential product. Copper ores are known to occur, and, in the past, have been considerably worked in the Nellore district of the Madras Presidency, in Rajputana, in Chota Nagpur, and at various places along the lower Himalayas from Kulu in the northwest to Sikkim and Bhutan in the east. Operations are now in progress by the Cape Copper Company to test the probable value of a copper-bearing country in Bengal. Prospecting operations are also in progress in the Darjeeling district and adjoining parts of the Sikkim State.

Lead, zinc and silver ores are known to exist and are being prospected on a considerable scale in upper Burma. Pitchblende and wolfram are also found, the latter in some quantity. Corundum is widely distributed in various forms, but it is possible that, in face of competition from natural and artificial abrasives in the countries of principal consumption, it would not pay at present to develop the Indian mineral for export to Europe and America.

The principal requisite for further development in the production of minerals in this country appears to be means of transport, a question which will have the consideration of those interested. The opening up of any particular field is almost invariably stimulated, in the first instance, by a brisk market. Unfortunately the period of high prices is very apt to terminate just as the preparations for opening up a new mine are ended, and it is in this connection that mercantile foresight becomes as valuable to the country as to the individual.

The continually urged objection to investment of political unrest the author does not consider grave. The results are temporarily sensational without having any widespread or permanent significance, like the political speeches of some home enthusiasts, while they serve to rouse to a stricter sense of duty those whose patriotism is not so case-hardened as to render them incapable of belief in sedition.

Recent Metallurgical and Chemical Patents.

Gold and Silver.

Silica Sponge Filters.—In two recent United States patents Mr. E. G. Spilsbury, of New York City, describes the use of a commercial product known as silica sponge for the purpose of filtering liquids or gases.

The treatment of gold and silver ore by the cyanide process involves methods of agitation and filtration which the inventor believes can be advantageously performed by the medium of a porous septum of mineral composition, such as silica sponge.

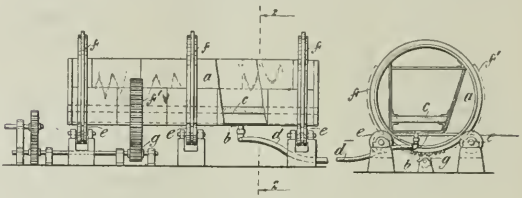


FIG. 1—SILICA SPONGE FILTER IN CYANIDE PROCESS.

As shown in Fig. 1, he proposes to place a septum of porous material *c* in a revolvable tank *a*, operated as shown in the figure. Beneath the septum is the bottom of the tank *b*, which is connected with a source of vacuum, compressed air and water by means of the flexible tube *d*.

The process of treating a charge of ore would be as follows: The ore having been crushed to the requisite degree of fineness and mixed with the cyanide solution would be charged into the tank *a*. For agitation of the pump compressed air would be admitted through the tube *d*, and when that operation was completed the air would be disconnected and vacuum applied through the same tube. This would draw off the pregnant solution from the ore, which could then be washed by water introduced through the tube *d* and filtered again as before. In this way the porous septum of silica sponge would act as a means of securing thorough agitation and aeration as well as filtration. When it is desired to discharge the filter cake the tank is rotated and the cake allowed to fall off by gravity, assisted by compressed air or water. (994,281, June 6, 1911.)

In applying the silica sponge to the filtration of gases, such as blast furnace gases from the smelting of lead and copper ores, the inventor proposes to use it much after the manner of using cotton and woolen bags in bag-houses. The mineral substance, however, will have the advantage that it will not be affected by the heat or acid nature of the fumes. The construction of a building for filtration of gases would be much the same as used in bag-houses, with a large chamber below and the filtering medium above, except that the bags would be replaced by a cell structure composed of a series of vertical cells alternately open and closed at the top, those cells which are closed at the top being open at the bottom, as shown in Fig. 2, which is a vertical section of the upper part of the filtering arrangement. The advantage of this arrangement is that the furnace gases will pass upward into those cells which are open at the bottom, filter through the porous walls of the

cell and escape to the air through the cells which are open at the top. When it is desired to clean the cells of the accumulated solid cake of fume, air can be forced in a reverse direction to the flow of furnace gas and cause the adhering cake to fall from the cell walls and down into the chamber beneath. This is an interesting modification of the present bag-house system and may have a bearing on the future operation of smelters in relation to agricultural interests which object to the discharge of noxious gases and fume. (994,282, June 6, 1911.)

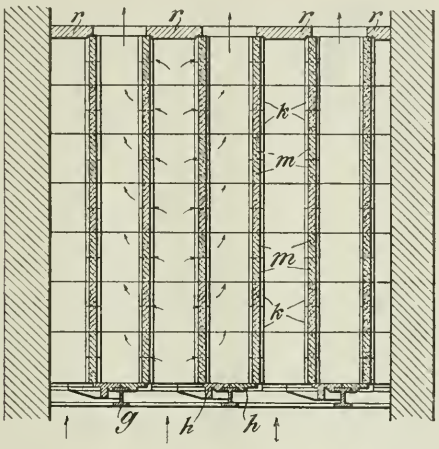


FIG. 2—FILTRATION OF GASES WITH SILICA SPONGE FILTER.

Rotary Filter.—A modification of the type of revolving filter commonly used in the United States has been patented by MR. ALEXANDER J. ARBUCKLE, of Johannesburg, Transvaal, South Africa. In the Oliver and Portland revolving filters used in this country the filtering medium is on the outside of the drum only, but in the Arbuckle filter it is proposed to make use of both the outside and inside surfaces of the drum for filtering purposes. The drum is divided into sections which, by means of a three-port valve located in the hub of the drum, are brought successively into connection with a vacuum for forming the cake, a wash and vacuum for removing the retained values and compressed air for discharging the cake. For this purpose three non-communicating spaces are provided in the hub, one for each process mentioned. Consequently, as the drum revolves in a tank of ore pulp, each section successively forms a cake on both its surfaces, which are later washed and discharged. (994,235, June 6, 1911.)

Air Lift Agitator.—In Fig. 3 is shown an improvement in air lift agitators for use in the cyanide process, patented by LEON P. HILLS of Tuolumne, Cal. The object of the invention is to provide means for bringing all the pulp in a tank into circulation, even after the ore may have settled and packed around the opening of the lift, and for securing a quiet overflow of clear solution in case of continuous treatment. The operation of the tank will be understood by reference to the figure, in which 33 is a launder and 34 a pipe for continuously introducing pulp and solution respectively. An air pipe 19 extends downward through the inner central tube, terminating in a U at the bottom. The central lift 12 is surrounded by a movable tube 20 which is operated by the hand wheel 25, this outer movable tube being shorter than the inner tube. A circular baffle 30 is placed in the upper part of the tank, as shown. The tank being full of a mixture to be agitated, the outer tube 20 is lowered into contact with the walls of the tank at the bottom, and air is admitted at the bottom of the inner tube. This will cause a circulation of the nearly clear solution, which will rise through the inner tube, spill over the apron 14 at the top and return through the annular space between inner and outer tubes, as shown at 29. When this circulation is thoroughly established the outer tube is raised gradually to admit the solid pulp at the bottom of the tank into the inner tube, and in this way the entire mass will ultimately come into circulation. The baffle 30 acts as a guide to prevent the pulp from flowing to the outer edge of the tank and directs it downward, thus preserving a quiet zone between the baffle and the wall of the tank where solution more or less free from pulp

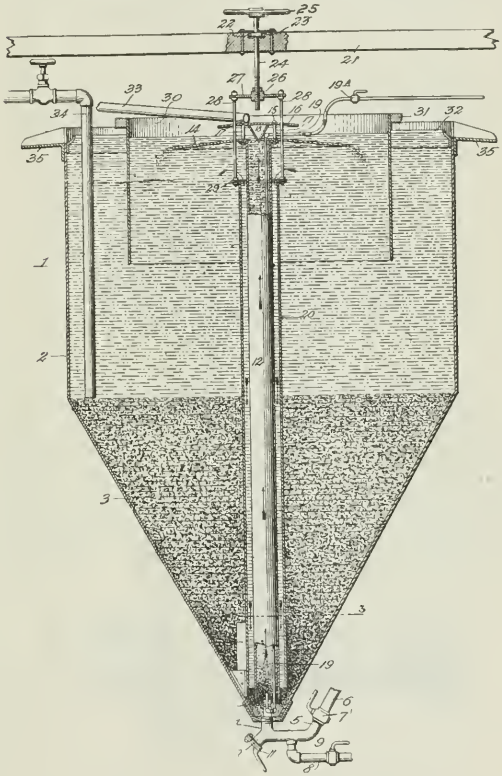


FIG. 3—AGITATOR.

can be drawn off through the circular launder 35. At the bottom of the tank a valve is provided for sluicing out the settled pulp when its values have been extracted by agitation. (994,679, June 6, 1911.)

Lead and Copper.

Fume Neutralization.—The system of fume neutralization as used by the U. S. Smelting Company at Salt Lake City, Utah, is the subject of a patent issued to MR. CHARLES B. SPRAGUE, of Salt Lake City.

Owing to the acid nature of the gases given off in the roasting of ores in converter and reverberatory roasting, it is impossible to filter those gases through bags in the manner customary with lead smelters. A system of neutralization of the sulphuric acid contained has been found possible, after which the gases can be filtered and discharged into the air, free from noxious substances, either solid or gaseous.

For neutralizing the acid in gases from converter roasters of the Huntington-Heberlein type, which are not as highly charged with sulphuric acid as those from reverberatory roasters, zinc oxide has been found efficient. Powdered lime, how-

ever, is preferred for neutralizing the gases from reverberatory roasters. In operation a furnace is built adjacent to a flue leading from the converter roasters, in which a mixture of zinc ore and coal slack is burned under a low blast. The zinc oxide produced is conducted into the roaster flue, where it mingles with the roaster gases and neutralizes the acid contained. When powdered lime is used, as on reverberatory roaster gases, it is introduced by means of a fan and eccentric feeder. The gases pass through the fan, and the feeder is set to deliver the proper quantity of lime into the gas.

Flues leading from both converter and reverberatory roasters may be united after the separate treatments have been applied and the combined gases filtered through bags as usual, the corrosive action of the gases having been overcome. (992,391, May 16, 1911.)

Roasting Furnace.—Improvements in the manner of attaching and detaching rabbles and rabble arms in ore roasters of the superimposed-hearth type have been made by Mr.

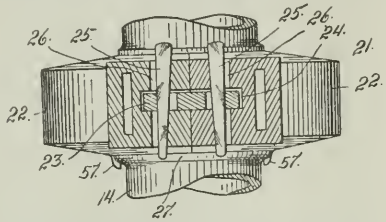


FIG. 4—DETAILS OF ROASTING FURNACE.

Chauncey E. Dewey, of Denver, Col. Figs. 4, 5 and 6 illustrate the invention. Referring to Fig. 4, the vertical central tube which carries the rabble arms in a furnace of the type described is represented by 14. The hub which carries the arms is composed of two twin members 22 which are connected by locking plates 23. As shown in Fig. 5, the hub is equipped with sockets adapted to receive the cone-shaped extremities of the rabble arms, which are fastened in the sockets by bayonet

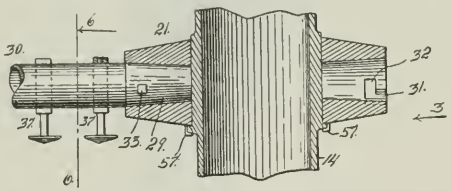


FIG. 5—DETAILS OF ROASTING FURNACE.

joints, so that when the furnace is in operation the movement of the rabble through the ore tends to hold it tightly in place. The rabbles themselves, which are shown in Fig. 6, are adapted for easy attachment or detachment. They are held in place by pins 34, which engage the perforations 35 made in the upper curved end of the rabble or plow. By means of the perforated lug 40 any plow may be removed from the rabble arm when necessary.

Alloys.

Ferrovanadium, Molybdenum and Tungsten.—The difficulty of preparing the ferro alloys of vanadium, tungsten and molybdenum usually has been that the reducing agent used remained in the resulting alloy in too large quantity. Mr. Gideon Boericke, of Philadelphia, Pa., has patented a process of reducing these alloys, which he claims eliminates the trouble experienced before. In carrying out his invention he uses a suitable form of the metals, such as vanadate of iron, which he mixes with manganese metal or alloy and a flux such as common sand. When the process of reduction is completed the products are ferrovanadium and a slag composed chiefly of the oxides of silicon and manganese. (993,338, 994,055, May 30, 1911.)

Purification of Magnesium and Alloys.—In the ordinary manufacture and refining of magnesium and its alloys more or less metallic chloride is retained in the metal, which subsequently has a deleterious action and causes oxidation of the metal in moist air. Berthold Hoffmann and Robert Suchy, of Griesheim-on-the-Main, Germany, have discovered that it is possible to readily and completely free magnesium and its alloys from this chloride by passing through the molten metal hydrogen or other reducing gas, which will drive off all traces of chloride at temperatures which are but little higher than the melting point of the chloride.

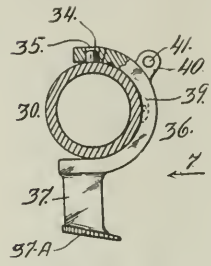


FIG. 6—DETAILS OF ROASTING FURNACE.

The process may be carried out as follows: About 8 kg of the metal are charged into a crucible fitted with a pipe through which hydrogen gas may be passed to the bottom. The current of gas is passed through the fluid metal for five or six hours at a temperature of 800° to 900° C. (993,373, May 30, 1911.)

Pyrophoric Alloy.—Metallic alloys which give sparks when scratched with a hard substance have hitherto been made by fusing together the metals of the rare earths, especially cerium, with heavy metals like iron. Adolf Huber, of Berlin, Germany, has discovered that aluminium can be substituted for iron and that the alloy thus resulting must contain not more than 50 per cent of the aluminium or other earthy metal. When the percentage of the cerium metals falls below 50 per cent the property of spontaneous ignition of the separated particles almost totally disappears. The preferable percentage of aluminium to be added is 12 per cent, at which proportion the alloy can be cast without disintegrating on cooling. The lower specific gravity of the aluminum-cerium alloy is an advantage, as is also the lower melting point. (993,908, May 30, 1911.)

Electroplating.

Cleaning and Polishing Silver Plate.—Silver plate is cleaned and polished, according to Mr. A. M. Kohler, of Brixton, London, England, by immersing in a sodium hydroxide solution in presence of aluminium or an aluminium alloy. When the articles are immersed in the solution and are in contact with aluminium, galvanic action is set up, the aluminium acting as anode and the silver articles as cathode. The anodic action is the formation of sodium hydroxide; the cathodic action is the reduction of the oxide which forms the tarnish on the silver to metal. Instead of metallic aluminium an alloy consisting of 80 parts aluminium, 10 barium, 8 calcium, 1 sodium and 1 silicon and impurities may be used. If the articles to be cleaned are greasy or very much soiled they may be immersed in a hot solution of Na_2CO_3 before placing in the solution as above described. (983,727, Feb. 7, 1911.)

Detinning.

Treatment of Old Tin Cans.—Preparatory to the removal of the tin from the iron old tin cans, etc., must be freed from everything that is neither tin nor iron. Dr. Karl Goldschmidt and Dr. Joseph Weber, of Essen, Germany, subject the tin cans to the following treatment: First, removal of the adhering organic substances by treatment with a hot 3 per cent caustic soda solution for five to fifteen minutes. Second, removal of the solder and the caoutchouc ring (which in folding the edges of the tins is placed in the fold to make the same tight) by sudden exposure of the tin cans to a temperature of from 600° C. to 800° C., the solder being melted off in five to seven minutes and the caoutchouc being destroyed at the same time. Excessive heating must be avoided, as this would cause the tin to alloy with the iron. (13,123, June 28, 1910.)

SYNOPSIS.

Gold and Silver.

Cyaniding at the Waihi Mine.—In the *Transactions* of the Australasian Inst. Min. Eng., Mr. E. G. Banks, metallurgist of the company, describes the method of treatment of an ore containing sulphides. All products are cyanided, but the sulphides are separated and treated separately from the balance of the ore.

Following rock breakers and stamps, the ore is ground in tube mills. The tubes are 18 ft. x 4 ft. 9 in., and have cast-iron liners in the shape of ribs 4 in. wide spaced at 15 in. centers. When new the ribs stand 3¼ in. above the body of the liner. They have a life of at least 18 months and represent a cost of a trifle over one cent per ton of stamp output. The consumption of pebbles is 2.6 lb. per ton of stamp output, representing a cost of about one and three-quarters cents per ton. The proportion of water to ore is kept at about 1 to 1, and each tube has a capacity of 110 to 120 tons of stamp product (10 mesh), although it grinds from 280 to 340 tons on account of the oversize returned from the sizing boxes.

Concentration follows tube milling, which is unusual practice, but justified by the author on the score that if the coarsest sulphide particles are removed it matters not if very fine sulphides escape to the cyanide plant. The sulphide concentrates are treated as follows: Grinding is accomplished in tube mills, and the coarsest of the tube product returned for further grinding. The ground pulp is thickened to 0.7 parts of solution to 1 of ore and agitated from 8 to 10 days by means of air in conical bottom tanks. Cyanide solution (NaCN) contains 0.45 per cent. of the salt, and the consumption is 16 lb. per ton, which represents 0.25 lb. per ton of original ore. The pulp is filter-pressed and the solution precipitated in zinc boxes.

The slime is thickened in V boxes and later by vacuum filtration, and is then mixed with solution of 1.2 per cent. KCN, and agitated by air in tall tanks for five days. The latest tanks built are of reinforced concrete 55 ft. high and 13 ft. diameter, erected at a cost of about $1,064, which is less than for similar sized steel tanks. No center pipe is used, slightly better results having been obtained without it. Gold is extracted in 24 to 30 hours, but a profitable extraction of silver continues up to five days. The consumption of cyanide is 1.66 lb. per ton of slime.

Vacuum filters are used to recover the pregnant solution, and zinc dust is used for precipitation. The extraction is 89 per cent. of the gold and 71 per cent. of the silver.

For several years the sulphuric acid process was used for refining, but was discarded two years ago in favor of the electrolytic method. The Doré bars run about 12 per cent. gold and 87 per cent. silver, and are separated in the stationary cathode type of cell. The output of 30 cells is about 5000 to 6000 oz. in two shifts (16 hours). The gold recovered runs 996 to 999 fine. The cost of refining is about 0.72 cents per oz. of Doré. The total cost of ore treatment is about $1.30 per ton.

Air Lift Agitation.—Writing on the subject of air lift agitation of slime pulp, Mr. Robert Allen presents some interesting data in the March, 1911, issue of the *Journal* of the Chemical, Metallurgical & Mining Society of South Africa. Referring to F. C. Brown's development of the so-called Pachuca tank, he says that the following points were found essential: The higher the tank compared with the diameter, the less the power required; the consumption of cyanide was less in tall than in shallow tanks owing to the fact that the necessary quantity of compressed air is less per ton of ore in the former than in the latter; although good agitation could be effected with slimy material in shallow tanks, yet as the coarseness or density of the material, or the specific gravity of the pulp increased, the ratio of height to diameter should be increased; the diameter of the central air lift pipe should be 1¼ in. for each foot of diameter of the tank.

On the score of excessive consumption of cyanide the author cites comparative results from the Loreto mill and the Hacienda San Francisco at Pachuca. In the former they use 30-ft. mechanically agitated tanks, with a charge of 57 tons in a thin pulp, while in the latter with Pachuca tanks agitated by air they work on charges of 112 tons of thick pulp. The cyanide consumption in the former mill is greater than in the latter and its power consumption is at least ten times as great.

Other experiments showed that Brown or Pachuca tanks consumed only 2.6 and 2.4 lb. cyanide on the same pulp on which mechanical agitators consumed 3.1 lb. cyanide per ton.

Lead.

Smelting in the Ore Hearth.—An interesting comparison of results obtained in smelting non-argentiferous galena in Scotch and Jumbo hearths at Granby, Mo., is given in the May, 1911, *Bulletin* of the A. I. M. E. by J. J. Brown, Jr., formerly connected with the Granby Mining & Smelting Company.

The ore hearth was at first a very wasteful means of reducing galena because no provision was made to recover the fume. Later this type of furnace was regarded with favor because it could be used to make a large percentage of fume or pigment, which was collected in the Lewis and Bartlett bag process. In fact, the production of metallic lead was kept down by hot blast, etc. Recently, however, a modification of the old Scotch hearth has been adopted for the reduction of lead, with a minor quantity of fume which is collected in bags, and slag which is reduced in blast furnaces.

The Granby company, however, having no blast furnace, was obliged to make as complete a recovery and leave as little lead in the slag as possible. About five years ago the system was as follows: The furnaces were water-backed Scotch hearths 2 ft. wide, using charcoal as fuel, and treating 14,000 of charge in about 14 hours, working two shifts of four men each. The men were required to convert 70 per cent. of the galena charged. 50 per cent. of the white fume, and 40 per cent. of the blue fume into pig lead. A premium of one cent per pound was given for better work than that required, but a similar penalty was charged for poorer work. Under this system about 85 per cent. of the lead in the charge was recovered, and about 2 per cent. went into blue slag, while the balance was recovered as fume and resmelted.

The author believed that better work could be done in Jumbo hearths and the result of his experiments was that all the Scotch hearths were remodeled into Jumbos. The percentages required of the operators were changed to 65 per cent. for galena, 60 per cent. for white fume, and 50 per cent. for blue fume contained in the charge, with the same premium or penalty as before. Cheaper fuel was used (bituminous coal instead of charcoal), and in a six months' run it was found that there was an advantage in favor of the Jumbo hearth of $0.6854 per ton of material smelted, or more than $1,200 per hearth per year.

The Jumbo hearths smelted a charge in less time than the Scotch hearths, the difference being about 4 to 5 hours per day. This is due to the larger fire area of the Jumbo hearth; the larger the hearth the more material can be smelted at one time provided the workmen can do the extra work entailed. A higher extraction was found on the 4-ft. than on the 5-ft. hearth, because the latter had too much fire to be worked advantageously. Pea-size ore is found to be the best.

A great objection to the ore hearth is that of small capacity. The author states that the only way to overcome this is to make the furnace mechanical in its operation instead of having it operated by hand. He states that according to designs of his own, in a patented furnace (888,582, May 26, 1908), the hearth may be from 10 ft. to 20 ft. wide, and worked by half the number of men.

Tin.

Anaylsis of Tin Ore.—Disregarding the fire assay for tin, which is regarded as inferior to volumetric methods, R. J.

Morgan gives his preferred method of estimating tin in an ore in the *Australian Mining Standard* of March 8, 1911. Six to eight grams of stick NaOH are heated in an iron crucible to expel moisture. About 5 grams of the finely powdered ore are added to the crucible after the NaOH has cooled and set. The crucible is then covered and gently heated to dull redness until fusion is complete. Allow the melt to cool, and then extract it with water in a casserole. Keep the washings as low as possible, but use enough weak HCl to wash the crucible and cover free from any adhering melt. Then cover the casserole and add HCl until effervescence ceases, and add 20 cc. strong HCl in excess. Heat to boiling and transfer to a suitable flask. There should be no undecomposed residue, save possibly some gelatinous silica.

Now add pure iron, in the shape of three horseshoe nails, to the solution and boil for 15 minutes. Filter through glass wool, washing with about 50 cc. water. To the filtrate add about 5 g. finely powdered pure antimony, heat to boiling and boil for 5 min. Cool as rapidly as possible in an atmosphere of CO_2. This can be done by dropping a small piece of marble in the flask, but a continuous source of CO_2 conducted through the flask is better. When cold the solution is titrated with standard iodine solution.

Zinc.

Flotation of Minerals.—In the *Australian Mining Standard* for April 5, 12, 20 and 27, 1911, Kenneth A. Mickle presents a very exhaustive statement of the conditions affecting the flotation of minerals in different solutions. Details are given of different experiments on many minerals and solutions under different conditions. It is impossible to present the details of the work, but the following conclusions will be interesting.

All minerals absorb gases.

Sulphides appear to naturally absorb CO_2.

The consequence of this is—

(1) Particles are not wholly wetted when immersed in water.

(2) Particles also tend to float when sprinkled on the surface.

(3) When immersed in water and heated, the air or gases dissolved in water will collect on the particles, and float or tend to float them to the surface.

(4) Saturated solutions of gases in water evolve gas which collects on all particles.

The Addition of Acids.—Metallic particles, such as sulphides and metals, when immersed in dilute acid solution, are not wholly wetted, but particles of rock materials become more readily wetted, and give off their adsorbed gas. When different dry minerals are sprinkled on the surface of dilute cold acid the tendency for them all is to float as in water. The minerals are more readily wetted by hot acid solutions than cold.

The Addition of Alkalis.—When different mineral particles are immersed in an alkaline solution, they tend to part with their adsorbed gases, and will not float even when heated. When sprinkled dry on the surface of the solution they remain for a few seconds only and then break through.

This is due to the property possessed by minerals of collecting gases on their surface in such quantity as to diminish the density of the product, to less than that of the liquid. By calculation from the decrease in density it is found that the amount collected on dry sulphides is sometimes half the volume necessary to float them at ordinary temperature. Given a starting point, any gas generated in or introduced into the solution will tend to collect on the surface of the particles. If acid is added, the gases will leave the gangue particles and collect on the sulphides; the latter, therefore, tend to float, and the former to remain under the surface of the liquid. When substances in the solution, such as carbonates, are attacked by acids, the gases evolved saturate the liquid, and commence to grow on the sulphides. Sulphuretted hydrogen, steam, air, and any other gases present either in the solution, introduced into it, or evolved from the minerals present, assist floatation, provided the sulphide particles have dry points on them for the gases to become attached to. Some sulphides naturally absorb gases, and such sulphides readily float. The bulk of Broken Hill ores may be taken as examples. Other ores require preliminary treatment, so as to alter their surface to some extent. This may be done in certain cases by merely heating to 250 deg. C., or immersing in some solution which will attack the sulphides, such as nitric acid or chlorine solution. Sulphides which condense or absorb gases well will gather together in a solution as flocculent masses, while those that do not remain individually distinct, like grains of sand.

In certain cases sulphides will float in acid solution at ordinary temperature, but for the most part with dilute acid solution flotation starts at about 80 deg. C. The explanation put forward that flotation at that temperature is due to certain carbonates, such as rhodochrosite, siderite, smithsonite being attacked and giving off CO_2, does not appear to be correct, for under reduced pressure these minerals are attacked, but flotation does not take place. The presence of silicic acid and sulphur, as stated by De Bavay as necessary for a coagulum, is not necessary in many cases of successful flotation. When minerals are treated with dilute acids, and then exposed to the air, certain sulphides absorb gases on their surface. If they are alternately wetted and exposed to the air they will become entrapped by the surface film of water which they are brought in contact with. The appearance of the film is the same as that of a bubble of gas under the surface of a solution with sulphides attached. The sulphides in both cases adhere to the surface film of the bubble or free surface of the liquid, but are mainly below the surface itself. The gas obtained in all cases from experimental work and also from samples obtained from various flotation plants was CO_2, with varying quantities of nitrogen and oxygen. There is no doubt that most of this gas is generated from gangue particles in the ore, and being more soluble in the liquid than the other gases, collects readily on the surface of the sulphides.

In all experiments with sulphides carbon dioxide was obtained on exhaustion. This gas must have been present either in the form of a compound easily decomposed by heating or by reducing the pressure, or as a gas condensed on the surfaces of the sulphides, and there retained with greater force than other gases present. It was found that the carbon dioxide was the last gas to come off.

The Effect of Oils.—The effect of oils has long been known in protecting minerals from being wetted by water owing to the oiling of their surfaces, and also the affinity of oils for metals and many metallic substances. Nearly all the minerals tried could be coated with oil either in the presence or absence of water. This seems to be connected with the absorption of gases on the particles themselves, for if the particles are deprived of their gas, then as a rule they do not become coated with it. It would therefore appear that the oiling of particles is dependent on their gaseous attachments. If the particles are oiled to a minute extent, the gases previously absorbed are retained, and if a solution is heated more gas will become attached to the particles, and flotation will ensue. An oil float may be obtained without acids. Nearly all minerals will float on water after being oiled.

The flotation is apparently due partly to the lesser density of the oil and the presence of gases absorbed on the minerals or entangled in the oil. If a large quantity of viscous oil is used gases become entangled in it, and the product is oily. In course of time the particles will drop away from it, carrying down attached globules of oil. When a lesser amount of oil is used the product becomes a coherent mass like putty, showing no apparent tendency to float. On further decreasing the quantity of oil, the product becomes firmer, and breaks up into rounded pellets. With a still less quantity of oil, when shaken up with water in presence of air the product becomes flocculent, and increases in volume. This flocculation is due to the absorption of gases. With very minute quantities of oil the flocculation becomes more pronounced, and the tendency to

float becomes greater. These results may be obtained with gangue and metallic substances. Gases collect more readily on oiled than on unoiled particles.

The Effects of Acids on Oiled Particles.—The addition of acid causes a more selective action. If oiled gangue and metallic particles are immersed in water, and acid is added, then the oil will tend to leave the gangue particles and become attached to the sulphides. The action of the oil in this respect is similar to that of the gases attached to different minerals, and probably is due to the same cause.

If an ore is treated with an acid and the acid is removed, then if the particles are oiled, the oil tends to become attached to the sulphides only. This is probably due to the removal of the gas from the gangue, which prevents oiling of the same. If acid and oil are added together, the action of the oil is selective, as in the previous case. The presence of oil on their surface does not prevent the minerals being attacked by acids.

The amount of acid required for selective flotation depends on the composition of the ore, the temperature and the pressure existing at the time. Generally it may be stated that an increase of acid will give a purer sulphide product, but flotation will take place at a lower temperature, and with a lesser amount of acid, with oil and acid than with acid alone. A selective flotation can be obtained in some cases by treatment with a chlorine solution previous to the addition of oil, and without the addition of acid.

Some of the light volatile oils cause flotation in an acid solution at a lower temperature than the heavier and thicker oils. With some Broken Hill tailings a flotation takes place at ordinary temperature with sulphuric acid and a volatile oil.

Effects of Oils and Alkalis.—Flotation can be obtained in alkaline solutions when oil is used, but in every case tried the action was non-selective. On making the solution acid the action becomes selective, only the sulphides floating.

The flotation of mineral particles appears then to depend on the particles having gas attached to them becoming entangled, or caught, by the surface film of a liquid. This film may be (1) the free surface of a liquid, (2) the surface of a bubble of gas under the surface of a liquid, or (3) the surface of a bubble of gas above the surface of a liquid. Once the particles are entangled in this film they are sustained there. That a considerable weight may be supported by this surface film is shown by the experiment of floating fine-grained mineral on the surface film of water.

The Purchase of Fuel Upon a Heat Unit Basis, in Germany and Switzerland.

By John B. C. Kershaw.

Considerable attention has been given in the United States to the methods of purchasing fuel upon a heat-unit basis, and the action of the government in arranging for the application of this system of purchase to all government contracts has given a great impetus to the movement for the use of scientific methods in the purchase and control of fuel supplies.

In view of the widespread and general interest in the subject in the United States the writer has thought that the following details of the progress of a similar movement, in Germany and Switzerland, would be found interesting.

In *Germany* considerable progress has been made during the last two years. In their report for the year 1909 the committee of the Hamburg Smoke Abatement Society state that an increasing proportion of the members of the society are taking samples and having tests made of their fuel deliveries. Over 1000 fuel tests were made at the official testing laboratories of the society in Hamburg in 1909, and it is expected this total will be largely exceeded by the figures for 1910 and 1911. The chief engineer of the Hamburg Smoke Abatement Society (Herr Nies) is firmly convinced of the value of chemical and calorific tests for checking the quality of the deliveries of fuel. This society, which had at the end of 1909 365 members and over 1300 boiler and other furnaces under its control, has arranged to undertake the regular sampling of their fuel supplies for those members who desire to be relieved of this work. This is a great step in advance, for to obtain a really fair average sample of coal or slack demands more skill and knowledge of sampling than is generally found in those who are trusted with this work, and the slow progress of the movement for more scientific control is partly due to this difficulty.

The actual testing of the fuel sample in Hamburg is undertaken by an independent fuel expert, Dr. Aufhäuser, who is head of the Thermo-chemische Prüfungs und Versuchs Austalt, or in plain English, of the Thermochemical Testing and Research Laboratory in Hamburg. The following are the details of the basis upon which the fuel is valued:

An average sample of the fuel is taken from the first or trial delivery after discharge at the consumer's works, and this is carefully tested at the official testing laboratory for moisture, ash and calorific value. The latter value is calculated on the assumption that the products of combustion of the fuel pass away in the state of vapor at 212° Fahr.—and with the aid of this result the calorific value of the combustible portion of the fuel is also calculated.

The regular samples of fuel taken to check the quality of the further deliveries are only tested for the percentage of water and ash; and from the results of these tests the calorific value is calculated by the aid of the figures obtained from the first or guarantee test. Should this calculated value deviate in either direction from the guaranteed value by more than the difference allowed by the term of the contract, an addition to, or deduction from, the price follows.

In those cases where doubt exists as to the correctness of the results, which in some cases may vary on account of a change in the calorific value of the combustible matter of the coal, the sample is submitted to all three tests, and if a different calorific value is found by the direct test, a correction is made accordingly.

It will be seen that this method of checking the fuel deliveries differs in two respects from that practiced in this country. In the first place the guaranteed calorific value of the fuel is based upon the "combustible" matter in the same, and is calculated for gasses escaping at 212° Fahr.; and in the second place, the regular samples of the fuel are only tested for moisture and ash, and the calorimeter tests are only repeated in cases of dispute.

As regards these modifications there is no doubt much to be said for the change. A fuel consists, from the practical man's point of view, of combustible and incombustible matter, the latter being made up of water and ash. If the fuel comes always from the same seam in the same pit the calorific value of the combustible portion of the coal ought to be a constant and not a variable value. The variables are the moisture and ash, and by estimating the percentage of these in each sample, and then calculating the value of the remaining combustible by aid of the value obtained in the first test, one can obtain the calorific value of the fuel actually delivered, with a minimum of time and expense.

But, this system of valuation ought to be checked, say, monthly or quarterly, by repeat tests with the calorimeter of the actual calorific value of the fuel, and the latter figure should be used until another determination is made. This will guard against errors due to the slow changes in character and quality, which undoubtedly occur in the working of some seams of coal, especially when the thickness is diminishing. It is also advisable to have some guarantee that the deliveries of fuel are from the mine and seam of coal stipulated in the contract. Mine managers often ship other coal than that bought, unless a penalty is arranged for such conduct.

As regards the assumption that the exit gases from the combustion of the fuel are always cooled to 212° Fahr, and the correction of the calorific value accordingly, this method of expressing the value is supposed to have the advantage that it

renders the calculations of the boiler efficiencies more simple, and that one can more easily judge from the calorimeter test results the actual steam-raising capacity of the fuel. As a rule, English and American calorific test results are based upon the escape of the gases at 15° C. and the value obtained in English, American and German calorific tests of fuel are, therefore, not comparable, until allowance has been made for this difference in the method of calculation. But no boiler is so efficient as a heat abstractor that the exit gases escape at a temperature of 212° Fahr. The German test results, therefore, still require correcting for the difference between 212° Fahr. and 560° Fahr., or 600° Fahr., which is generally the temperature of the escaping gases, and the simplifying of the calculations required when estimating boiler efficiencies is thus more apparent than real.

Below are a group of test results calculated on this basis obtained at the official fuel testing laboratory of the Hamburg Smoke Abatement Society and taken from the last annual report of the society:

	Percentage of			Calorific values in centigrade units.	
	Moisture.	Ash.	Combustible.	Raw coal.	Combustible.
Durham coal	1.6	2.2	96.2	7951	8273
	4.6	15.8	79.6	6339	7999
	2.6	6.9	90.5	7350	8139
Northumberland coal	1.4	13.4	85.2	6907	8115
	5.4	8.7	85.9	6840	8001
	12.1	5.5	82.4	6413	7871
Yorkshire coal	3.5	7.3	89.2	7145	8034
	6.4	4.8	88.8	6987	7912
	3.7	3.3	93.0	7526	8116

The above calorific tests require multiplying by the factor 1.8 to obtain the corresponding values in British thermal units, and it will be seen that there is a margin of 402 Centigrade units (723 British thermal units) between the highest and lowest results when based on pure coal or combustible. This difference proves that even after the two variables, moisture and ash, have been eliminated from the coal, considerable variation exists in the calorific value of the coal mined in different collieries and seams in the North of England and that the Hamburg system of valuation, if adopted, must be used with discretion and common sense.

As regards the progress of scientific methods of controlling fuel supplies in *Switzerland* the state authorities in that country established in 1907 at a cost of $10,000 a well-equipped fuel testing laboratory at Zurich, and have since 1908 based all contracts for the supply of fuel to the State railways upon calorific tests.

The principle employed here is to fix an average heat value for the fuel being delivered (in this case German and French briquetted fuel) and to deduct from or add to the price a proportionate sum for every 50 heat units (90 British thermal units) by which the calorific value as delivered differs from this standard. An illustrated description of the State fuel testing laboratory at Zurich appeared in the issue of the *Engineer* for June 12, 1908, and it was there stated that over 3000 samples of fuel had been tested in the first year of its existence.

Most of these represented German fuels, for Switzerland has to import all her coal, at an annual cost of $15,000,000. The remaining samples were of Belgian, French and English fuels.

The director of the laboratory at Zurich, Professor Coustam, has stated that this extensive examination of imported fuels in Switzerland, both for the State and private consumers, has tended to enlighten the buyers as to the qualities and economic value of the fuels from the various countries and collieries and has also contributed toward the classification of fuels according to their heating power or value.

In a recent letter to the author of the present article Professor Coustam confirms this view, and asserts that fuel testing in Switzerland is centralized at the State Fuel Testing Laboratory where practically all fuel for railway and gas-making purposes is tested regularly—i. e., a sample being taken from every twentieth car-load of coal as it arrives. In 1909 4000 samples and in 1910 4400 samples were tested calorimetrically, besides about 1000 samples of gas-coal tested for moisture, ash, volatile matter and sulphur.

All parties concerned (producers *and* consumers) appear to be satisfied with the methods of fuel testing, the former agreeing that they are thereby protected from unjustified complaints. The results of the tests are compiled monthly, and are communicated to the sellers and buyers, the monthly mean serving as basis for payment.

Liverpool, England.

Tests for Fibrous Insulating Materials.

Next to mechanical strength, the most important factor in determining the value of different types of electrical generating machinery is insulation. The size and life of dynamos and motors depends not so much on the actual current-carrying capacity of their conductors as on commutation, and the resistance of the insulating materials employed to the disintegrating action of heat, in the first place, and humidity in the second.

A dry heat need obviously only be limited in intensity by the point of combustibility of the substances employed, and it has naturally always been the aim of electrical manufacturers to employ substances as incombustible as possible. A test for inflammability under dry heat is easily contrived, and has not therefore been considered so important as one under the more onerous conditions of heat and moisture simultaneously applied.

Tests under these circumstances and the data gathered form the subject of a paper presented to the British Institution of Civil Engineers by Mr. W. Pollard Digby. This paper more particularly alludes to tests on the three well-known substances, "Press-spahn," "Leatheroid" and "Empire" cloth.

The test employed was to expose strips of the substance to be investigated, of known area and weight, to the action of a vapor bath having a temperature of 167° F. (75° C.) for periods aggregating 1 hour and 40 minutes, the weight and insulation resistance per square inch being taken every 20 minutes. The results of the test go to show how widely thin "Empire" cloth (between 4 and 5 mils thick) varies in its properties. Increases in thickness for this substance were obtained from 1.7 to 11.1 per cent., and increases in weight of from 5.9 to 7.8 per cent., the insulation resistance at the end of the test varying from 10.1 to 11.2 megohms. Thicker material (from 6 to 6.9 mils) increases in weight of from 2.6 to 6.6 per cent., were noted, and the insulation resistances on three samples were as high as 5,000 megohms per square inch. These tests showed the value of using a single thickness of, say, 12 or 14 mil material, rather than an equivalent thickness of two or more sheets of thinner material.

Press-spahn or fuller-board, unvarnished, gave, on the whole, poor results. The more satisfactory samples varied from 1.2 to 40 megohms, and other results were as low as .0001 of a megohm. The thickness seemed to make little difference. The conclusion come to was that "Press-spahn" was unsuited for any but small low voltage machines.

Leatheroid gave more satisfactory results, the poorer registering a resistance after test of 10.8, 7.7 and 4.9 megohms per square inch, while six out of nine samples gave 40 megohms per square inch, with increase in thickness not exceeding 3.4 per cent. The thicker samples (over 10 mils), however, were not so uniformly good, only three out of twenty having insulation resistances of more than 1,000 megohms per square inch. As a potential specification for this substance, the author permits the use of leatheroid, which, after the vapor bath test at 167° F. for 100 minutes, has an average insulation-resistance of over 20 megohms per square inch and an increase in weight and thickness not exceeding 5 per cent. This substance compares very favorably with press-spahn, as, for any given

increase in weight and thickness the moisture absorbed lowers the insulation resistance of leatheroid to a much less degree than that of press-spahn.

The function of these materials (press-spahn and leatheroid) is generally described as mechanical, i.e., they are used to protect the actual insulating sheaths of molded mica or woven material from abrasion in the slots. At the same time it is obviously advantageous that the most durable and non-hygroscopic material should be used.

Data on dielectric strength were only available on one specimen of leatheroid, which was tested by a manufacturing firm. The volts per mil thickness were here found to increase up to 15 mils in undried, and 21 mils in dried leatheroid, after which they apparently fell off. The highest recorded was 475 volts (on 8 mil material), and the lowest 229 volts on 21 mil.

As a result of thirty-two tests it was found that "leatheroid" might contain up to 5.5 per cent., and "Empire" cloth up to 2.96 per cent. of moisture and volatile matter, the figures for vulcanized fibre ranging up to 2 per cent.

The paper concludes with a suggestion that a formula expressing the minimum permissible resistance for motors and generators after heat-run should read: $R = Kv \div s$, where v = voltage of machine, s = estimated net iron-area of one slot, multiplied by the number of slots, and K = a constant (not less than 15). This formula would differentiate between high-speed machines with small surface and low-speed generators of large diameter.

NOTES.

The Linde Air Products Company, of Buffalo, N. Y., have just shipped a complete oxy-acetylene cutting plant arranged for six operators to the War Department for use in conjunction with the salvage of the U. S. S. Maine in Havana harbor. Much of the work involved is the cutting of 9-in. armor plate in addition to the usual structural shapes. It is interesting to note that similar apparatus to this was used with signal success in connection with the removal of the debris of the Quebec bridge.

Dings electromagnetic separators are built in low, medium and high intensity types designed for different kinds of work. The low intensity type is useful for separating metallic iron from non-magnetic material. Those of medium intensity are used for the separation of roasted sulphide ores, magnetic iron ores, etc., while the high intensity machines are applied mainly to the separation of wolframite, monazite, rutile and minerals with very feebly magnetic constituents.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in March, 1911, was 68. The total tonnage milled during that period was 2,005,526 tons of ore; 9911 stamps were in operation with an average duty of 7.62 tons per 24 hours. Tube mills in commission numbered 209. The yield for the month was 676,065 fine ounces gold. This report shows an increase of about 200,000 tons of ore milled and an addition of 47 tube mills in commission as compared with one year ago.

In the Annual Report of Phelps, Dodge & Co. appears a brief statement of the experiments on dust settling at the Copper Queen Smelter. During 1910 these experiments were carried to conclusion, and it was shown beyond question that it is possible to settle a large part of the finest dust in a comparatively short but wide chamber, in which the velocity of gases is reduced to between two and two and a half feet per second. The dust chambers are, therefore, being reconstructed by sections. The first section will handle the smoke from two blast furnaces. It is 1300 square feet in area, and it is thought this will be sufficient to settle most of the fine dust. In order to reduce the amount of gases passing through the main chamber a separate flue is being built for the converters. A calcining and reverberatory smelting plant is being built to treat fine ores and flue dust. This will be completed by the end of this year. It will consist of two reverberatories with boilers built in the furnace flues to utilize waste heat; six McDougall roasting furnaces; dust chambers, flues and stack.

Filter leaves or clarifying leaves may be bound with iron wire covered with P. & B. paint instead of the usual method of sewing them together at intervals. The wire is bound around the frame outside of the cloth and then fastened with staples passing over the wire on one side of the leaf and clinching over a wire on the opposite side. The scheme may have some advantage in cases where the threads used in sewing tend to break when reverse pressure is applied to discharge the cake.

Ferric iron may be precipitated quantitatively by a 6 per cent solution of nitrosophenylhydroxylamine, according to Biltz and Hodtke in *Zeit. Anorg. Chem.*, 1910, 66, 426-431. An excess of acid does not interfere with the precipitation, and iron may be separated from nickel, chromium and aluminium in this way.

A new classifier designed by G. A. Overstrom makes use of a rising current of water in successive sorting columns for the purpose of separating the specifically heavier particles into certain grades. The slime is removed first, and in each succeeding sorting column the rising current of water is so adjusted that successively finer grades will be raised out of the pulp, while the heavier settle to the next sorting column. The classifiers may be arranged in succession to receive and treat the specifically heaviest grade from the next preceding classifier. The system requires little clear water, as the overflow from each classifier is practically clear and is used in the succeeding machine.

The cost of silver production at Cobalt, Ontario, Canada, is indicated in the fourth annual report of the La Rose Consolidated Mines Company. During the seven months covered by the report a total of 2,569,905 oz. silver were produced at a cost of 19.11 cents per ounce. Of this quantity 401,221 oz. were produced by concentration, representing a recovery of 74.61 per cent. As a result of the large quantity of cobalt offered from this district the market is overstocked, and all payments for this metal have been discontinued.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore Treatment *(Continued).*

405,824, June 25, 1889, Turner D. Bottome, of Hoosick, New York, assignor to John B. Tibbits, of same place.

Gold and silver. The ore is reduced to powder and blown by steam into an atmosphere of chlorin and atomized water, the metal being thereby converted into chlorid. The chlorinated ore falls through sprays of sodium thiosulfate, to dissolve out the chlorids. The undissolved residue is filtered out and the solution is electrolyzed between electrodes of carbon and copper, the gold and silver being deposited on the cathodes.

407,386, July 23, 1889, and 410,228, Sept. 3, 1889, Jacob C. Wiswell, of Medford, assignor by Direct and Mesne Assignments, to the Electric Gold and Silver Chlorination Company, of Boston, Massachusetts.

Produces mercuric chlorid solutions for separating gold and silver from ores. A sodium chlorid solution, to which is added ammonium chlorid or hydrochloric or sulphuric acid, is electrolyzed in a tank of wood between anodes of carbon and cathodes of zinc. The resulting solution is delivered into a second electrolytic cell containing an anode of mercury and a cathode of carbon. The mercuric solution produced in this cell is employed to free the precious metals from the gangue.

415,576, Nov. 19, 1889, Werner Siemens, of Berlin, Germany, assignor to Siemens & Halske, of same place.

Copper, zinc, etc. Lixiviates the ore with a solution of ferric

sulfate, chlorid or cyanid, which dissolves the metal and is reduced to the ferrous form. Electrolyzes the resulting solution at the cathode, depositing the non-ferrous metal; then reoxidizes the residual solution at the anode to the ferric form. The operation may be carried out in one cell or in a number of cells arranged in cascade. The single cell is divided into two compartments by a cup-shaped diaphragm. The cup contains an anode of carbon, platinum or lead, and receives one leg of a syphon, which extends nearly to its bottom. The outer compartment contains a cathode-plate and a liquid supply-tube. The cup commnnicates with the outer compartment through a hole near its top. In the modification employing a series of cells, the anode and cathode compartments of adjacent cells are connected by syphons. To extract copper, the ore, for example, pulverized pyrites, is roasted until the iron therein is entirely oxidized and is lixiviated with ferric sulfate or chlorid, the resulting solution containing cupric and ferrous salts. Some free acid is added to increase the conductivity and the liquid is electrolyzed at the cathode, the copper being deposited. The solution, becoming lighter, rises in the cathode compartment and, when but one cell is used, flows through the opening at the top of the cathode-cup into the anode compartment. Here the ferrous salt in solution is reoxidized into the ferric form. In the series-cell construction, the two solutions pass successively through the cathode and anode compartments of all the cells, to secure complete deposition of the copper and oxidation of the iron. The depleted liquid from the last cathode compartment is returned to the first anode compartment. The potential difference used is 0.7 volts, and the cathode current-density 25 amperes per square meter. Zinc is similarly extracted from slightly roasted blende, the necessary potential-difference being nearly double that for copper.

415,738, November 26, 1889, Hermann A. Seegall, of Berlin, Germany.

Copper, zinc, tin. Lixiviates the ores first with ferric chlorid and then with a solution capable of dissolving the residual cuprous chlorid in the ore, for example a halogen salt or acid, or sodium hyposulfite. The second solution, plus ferrous chlorid, may be electrolyzed in an open cell, or it may be electrolyzed at the cathode in a diaphragm cell with ferrous chlorid in the anode compartment. Examples: 1. Lixiviates copper and zinc sulfid, or an alloy of the two, with ferric chlorid slightly acidulated with hydrochloric acid, until the iron is reduced to the ferrous form, the zinc dissolved as chlorid and the copper in the residue converted into cuprous chlorid and a little spongy copper. The copper is filtered and electrolyzed to recover the zinc. Copper cuttings or filings are added to the residue and it is lixiviated with ferric chlorid, slightly acidulated with hydrochloric acid and containing calcium or soldium chlorid to dissolve the cuprous chlorid. The solution is filtered to remove the freed sulfur and is electrolyzed to recover the copper. 2. Copper tin scrap is lixiviated with ferric chlorid slightly acidulated with hydrochloric acid. The tin dissolves and the solution is filtered off and electrolyzed to recover it. The residual cuprous chlorid is recovered as before. A series of electrolytic cells may be employed, the solutions passing successively through the several cathode and anode compartments, that of the last cathode compartment being returned to the first anode compartment.

418,134, Dec. 24, 1889, Henry Forbes Julian, of Johannesburgh, Transvaal, South Africa, assignor to Flavien Ernest Lezard and James Joseph Coghlan, Trustees.

Gold and silver. The ore is pulverized and agitated in a closed vessel with a solution containing bromin, chlorin or iodin or their salts. A pressure of from 60 to 80 pounds per square inch is maintained in the vessel by air or steam. When the gold has been dissolved as a haloid and the silver converted into an insoluble haloid, mercury or sodium amalgam is added to the semi-liquid mass, which is again agitated, amalgamating the gold. Or the gold in solution may be precipitated. The tailings are then run through a series of electrolytic cells, in cascade, having inclined cathode bottom-plates of amalgamated copper covered with mercury and depending anodes of carbon, platinum or lead peroxid. The remaining gold and silver are thereby dissolved and redeposited in the mercury.

452,125, May 12, 1891, Werner von Siemens, of Berlin, Germany, assignor to Siemens & Halske, of same place.

An apparatus for effecting the step of initial lixiviation in the process of his patent 415,576 (supra). The vessel is a long trough, or a series of connected troughs arranged zigzag, containing parallel shafts carrying paddles and a heating pipe.

PERSONAL

Mr. H. Foster Bain has returned to San Francisco from a trip to Colorado and will soon go to Alaska.

Mr. Thomas A. Dickson, formerly with the Colorado Iron Works Company, Denver, has gone to Goldfield, Nev., to take charge of the refinery of the Goldfield Consolidated Mines Company.

Mr. C. A. Filteau has returned to Denver from San Jose de Gracia, Sinaloa, Mexico, where he has been in charge of cyanide operations of Cia. Minera Jesus Maria y Anexas.

Mr. Franklin Guiterman, general manager of the American Smelting & Refining Company in Colorado, has left for several months' stay in Europe. He will study the progress in electric smelting with particular reference to zinc reduction.

Mr. Charles Janin, of San Francisco, was in Denver recently on a professional trip which took him through Arizona and Colorado.

Mr. William F. Kett has been appointed manager of the Mountain Copper Company, California, in place of Lewis T. Wright, who has resigned.

Mr. L. H. Norton, of Silverton, Colo., was in Denver recently on professional business.

Mr. Cooper Shapley has been appointed superintendent of the Oro Grande Company at La Luz, Guanajuato, Mexico.

Mr. George M. Taylor has been appointed general manager of the milling department of the Portland Gold Mining Company, Colorado, vice J. B. Daniels, deceased.

Mr. J. C. Welch has resigned his position as smelter superintendent of the East Butte Copper Mining Company, Butte, Mont.

Mr. Lewis T. Wright has retired from the management of the Mountain Copper Company, California, and will take up his residence in London, at least for a time.

BOOK REVIEWS.

Die Metallurgie des Zinns.—Mit spezielle Berücksichtigung der Electrometallurgie. (Band XXIX. Monographien über angewandte Elecktrochemie.) By Dr. Hans Mennicke, chemical engineer. 8vo., 196 pages, 40 illustrations, price 10 marks, bound 11.50 marks (retail price in New York $3.75). Halle a. S.: Wilhelm Knapp.

The ordinary processes of extraction of tin from ore are but briefly treated; Schnabel's "Handbook of Metallurgy" is more satisfactory on this part of the subject. The chapter on obtaining tin from tin skimmings, dross, and other chemical wastes is interesting and contains much new information. A 43-page chapter on the detinning of tin-scrap is probably the first publication of a description of some commercial plants. A 25-page chapter on the refining of impure tin, chemically and electrolytically, is most acceptable. Several short chapters follow on electroplating tin, production and consumption of tin; finally, there are 40 pages containing the full specifications and claims of important patents concerned with the metallurgy of tin. The whole is a useful and valuable contribution to the scanty metallurgical literature of an important and interesting metal.

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Sintering Fine Iron-Bearing Materials by Blast-Roasting Machines.

Mr. James Gailey, who needs no introduction to our readers, has presented an exceedingly interesting paper before the June meeting of the American Institute of Mining Engineers on the sintering of flue dust on the Dwight-Lloyd system. As is well known this machine consists of a moving train of perforated iron pallets passing over a suction-box and that on this is placed the ore which is desulphurized by the blast principle of roasting. The machine resembles in appearance a short belt-conveyor.

This machine has been used successfully for the roasting of lead and copper ores. Mr. Gailey describes some experiments looking toward its application in the sintering of flue-dust and ore from the recently discovered mines of Mayari, Cuba. This proposal is to replace the sintering of the ore in long nodulizing kilns. The experiments described are favorable to the proposal. Both ores were delivered from the sintering machine with a large proportion that was coarse or granular. The percentage of iron was increased in the case of the Mayari ore from 48.53 per cent to 53.97 per cent and in the case of the flue-dust from 46.06 per cent to 57.90 per cent. The first increase is mainly due to the burning out of the coke while the second is due to the dehydration of the ore. All the pieces of the ore are described to have consumed a cellular structure like coke which is peculiarly desirable for the blast-furnace.

Experiments were made with magnetic concentrates and it is remarkable to note that the sulphur was reduced in the sintering from 1.17 per cent to 0.006 per cent. In the case of the magnetic concentrates and the Cuban ore, seven per cent of coal (presumably bituminous coal) was added. The flue-dust contained enough coke to furnish heat by its combustion for the operation. Mr. Gailey estimates the power needed for a 100-ton plant at 45 hp., the cost of repairs at 6 cents per ton. The capital cost of plant is low, as a 100-ton plant only weighs 26 tons.

We believe that this is a most important common-sense application of a machine devised for one operation to a process somewhat different in purpose but similar in modus operandi. The question of flue-dust in the smelting of Mesabi ores is most important. If the ore could be screened and the objectionable fines eliminated and sintered in a cheap efficient process a lot of blast-furnace managers would sleep better. Though we do not hear much now about blowing off the tops of blast-furnaces, yet Mesabi ores are smelted by slips and falls and the smelting is hard on the nerves.

It is very apparent from this and other proposals that the metallurgical engineer who recently proposed to amortize his plant on the ground that it would not be obsolete for twenty years was far from right. "The depreciation due to

the advance in the state of the art," should be at least ten per cent. The metallurgical profession has not given this important point the due weight that the electrical profession has.

Arbitration of the Smelter Fume Question.

The recent announcement of a suit brought by ranching interests in Nevada to restrain smelting operations on the part of the Mason Valley Mines Company, at its plant which is now in course of construction at Thompson, Nev., brings forcibly to mind the necessity for devising some means of reconciling the conflicting interests of agriculture and metallurgy. Both are of fundamental importance as original sources of wealth, and as such their interests are in a measure mutual and interdependent, especially when their several operations are carried on in the same locality. No permanent good can come from suits or injunctions, nor from resort to warfare of any sort. The settlement of the problem requires calm and impartial investigation, consideration of the interests of both parties, and, above all, extreme fairness in placing restrictions on the one or granting indemnity to the other.

That smelter fume may be and has been the direct cause of damage to vegetable life does not admit of contradiction. Three principal sources of damage are recognized, viz., sulphur dioxide, sulphur trioxide and solid fume such as arsenic. The best scientific talent in the profession has been marshalled to study the nature and extent of the damage which these substances cause and the possibility of taking reasonable steps to prevent their indiscriminate discharge into the air. It has been pretty thoroughly proved that it is possible to prevent the discharge of sulphur trioxide and solid fume, but it may be asserted that in the present state of the art no reasonable and effective method has been proposed to prevent large quantities of sulphur dioxide from escaping into the air and doing possible damage.

Sulphur trioxide may be neutralized with zinc oxide and collected in bag houses as at Midvale, Utah, and Kennet, Cal. Solid fume may be settled in flues by cooling to the proper degree of temperature and causing it to impinge on baffles as at Great Falls, Mont., and other places. But sulphur dioxide presents a different problem. It is not so much the fault of chemical knowledge that this gas is not handled as well as sulphur trioxide, but rather the difficulties encountered on the mechanical and commercial side. Many proposals have been made which undoubtedly will neutralize this gas and eliminate it as a source of damage. But what about the cost of equipment and operation of these schemes? How about the difficulty of handling large quantities of flue gas without seriously interfering with the draft on the furnaces? What shall be done with some of the neutral products obtained by these various proposals, and finally, if sulphuric acid be made from the gas, what shall be done with the acid?

Here is a problem which any sane farmer must not be too insistent in forcing the smelters to solve. The latter are fully conversant with the situation, much more so than the former, and, we are proud to say, the spirit of fairness and generosity has marked the attitude of the smelters more than that of the farmers. The larger companies, against which most of the agitation has been directed, acknowledge the possibility of damage and realize their present limitations in preventing it. It is a matter of economy with them to prevent the indiscriminate discharge of fume. Furthermore, they stand ready to pay for unavoidable damage and regard such payment as one of the expenses incident to the business; but they want the damage fairly assessed with due regard to all the conditions.

There are things which can be settled only by a fight—by a fight to a finish. It seems almost foolish to deny this. But can any sane man think seriously of a fight to a finish between agricultural and metallurgical interests? We believe that the solution of their problem lies in arbitration and that arbitration will prove in this restricted field immensely more successful than in the evasive and spectacular field of international politics. We decry suits and injunctions as being only so many expensive obstacles in the way of progress. The Anaconda Copper Mining Company and the United States government have set a good example in appointing a technical commission to consider the question as far as it relates to their several interests. This commission has arbitrary power to decide whether material damage is done to the government, and what steps, if any, the Anaconda company should take to prevent such damage. The arrangement is eminently satisfactory and is regarded on all sides as the only fair solution of the problem and one which should be adopted instead of resorting to the courts. Suits are expensive and leave one party dissatisfied. Injunctions only paralyze industry and frequently reflect injury on those who obtain them. Arbitration will be cheaper in every sense of the word, will possibly satisfy all concerned, and will surely reflect credit on the intelligence of both parties to the controversy.

The Metal Markets.

As we have pointed out several times, the increase of the productive capacity of the various metallurgical plants of the country in the past ten years was so rapid as to overrun the consumptive capacity of the country. From 1900 to 1907 there was almost without halt an increase of from 7 per cent to 10 per cent per year in the production of iron, steel, lead, copper, zinc, and the minor metals and this increase was compounded. At the present time we believe that these several basic industries are recovering from the period of mental unsettlement that occurred after the period from June, 1909, to July, 1910, when the wheels of commerce seemed to be rushing again. At present the production is going directly into consumption. As this consumption is on a hand-to-mouth basis, it is hard to see how it can be much reduced. As this situation is quite widespread throughout all industries, it would seem that the metal prices have about touched rock-bottom and the general tendency in all should be upward. There would be little doubt about this upward tendency were it not that the volume of all crops except that of cotton seems bound to be below the average of the past four years.

Sober-minded people while they may not exactly prefer a period of liquidation and retrenchment do not dread it, for that is the period when everybody works more and spends less, and when the entire spirit of inflation and speculation is curbed and chastened. This conservative policy of producers and the everywhere manifest attempt to reduce fixed charges,

to eliminate useless expenses, to improve the efficiency of plant and workmen, has put the metal industry in good shape to meet any disappointment that can result from the diminished harvests.

It also can be remarked that there are a great number of far-seeing and able inventors in this country and that their endeavors are appreciated more in periods when it is a question of a maximum of plant efficiency rather than a question of a maximum of plant capacity. Their efforts will reap a golden harvest when rightly and conservatively employed. Furthermore keen observers notice among the plain people of the United States a manifest displeasure with the attempts at "trustbusting" simply for the glory or fun that may be in it and a desire for a sane regulation of these instruments of production. When all these curative influences have done their work, we shall see a more normal growth of the metal industry. In the immediate future the outlook is not particularly promising.

Western Hospitality.

After a red-hot summer day in Omaha, to wake up the next morning in the Pullman sleeper in Colorado and see through the windows in the distance the snow-clad peaks of the Rocky Mountains, is a sensation which has given a thrill to many a traveler from the East. The stately range of peaks in the glory of their eternal snow offers the first greeting to the stranger who feels that he is entering God's country. And when on leaving the train in Denver he turns toward the city, there is the big "Welcome" Arch—that pretty and touching symbol of Western hospitality.

It has been truly said that nobody who has not seen the West and the Pacific Coast can have a full and proper conception of this country and its people. It is true, the East is mighty, and New York City is really a wonderful sight with its feverish commercial activity concentrated in such a limited space that its buildings must grow to scrape the sky. Yet the East and all its activity would be unthinkable if it were not backed up by the unlimited natural resources of our big and beautiful West. There is everything big and great in the West—the giant trees in the virgin forests, the orange groves, the wheatfields, the mines, the concentrators, the smelters. And the very men themselves—generous, cheerful, big-hearted, enthusiastic, and of a never-dying optimism. How could they have built up again a new and more beautiful San Francisco after the calamity of only a few years ago, if it was not so? Nature, bountiful and generous, is reflected in the big-hearted optimistic generosity of those empire builders of the West. And its natural expression is Western hospitality which no one who once enjoyed it can ever forget.

The Western man, whether he be a metallurgist or a mining man or anything else, always gives himself as he is. He is not afraid to show his smelter or concentrating plant to the interested visitor. There is nothing about him of that anxious secretiveness which is such a distinguishing trait of some of our Eastern and European friends. While his work speaks for him, he is not afraid to speak of his failures. He believes in the do ut des. He is always an optimist, and where old methods fail he looks out for new ones, and extends to them his hospitality.

In some parts of the West where the immense strides in agriculture have been coincident with a temporary decline in mining, the latter industry has suffered a loss of prestige as a money-making business. This has been due partly to a lack of spectacular discovery or development in new mines, and the cry has gone forth that new Cripple Creeks must be found in order to stimulate the industry. That the old spirit is still alive is shown in a broader interest and greater activity in prospecting and development, more evident now than has been apparent for some years past, and undoubtedly some good will result. But in another direction and in a far more important manner the true Western pioneer spirit shows itself now in the willingness to accept and attack new methods of metallurgical treatment.

Indeed, of greater importance than the discovery of new and rich mines, is the development of improved metallurgy for the treatment of ore already known to exist and, in some cases, already mined. As a measure of conservation of our natural resources improved metallurgy is of fundamental importance and we may look to see greater results accomplished in the immediate future by solving metallurgical problems than from the discovery of new mining districts. The most active metal-mining districts in the West to-day are those in which the metallurgical problem has been solved. The "dead camp" is the one that abounds in ore that cannot be treated at a profit. The main point is that the West is fully alive to the problem of the situation, and that means the West will solve it by its own energy and by its willingness to try new methods even if not yet completely proven. The Noble electric iron ore reduction and steel plant, now going up in California, is a true expression of this new metallurgical pioneer spirit.

In general it is evident that the limits for successful ore treatment are gradually widening. Not that any new and startling process is proving a panacea for all troubles, but accessory devices and schemes applied here and there in conjunction with well known and proved processes are raising the percentage of extraction and extending the application of the processes to a wider field. It is a fact that an old slag dump of a smelter near Denver which was dismantled not so long ago is now being treated for its metal content at a good profit.

In concentration where great waste has hitherto existed, improvement is being made by the use of flotation as an accessory process. Dry chlorination holds out some hope for the treatment of complex ores; in two western plants it will soon have an excellent chance to show what is in it of commercial success. Cyanidation is being extended by the aid of additional chemicals; and from physical chemistry the phase rule is borrowed to base on it an interesting modification of cyanide practice (as described in the suggestive articles of Mr. Lowden and Mr. Rothwell in our July issue and in this issue respectively). Electrolytic schemes are in sight which may prove of value in conjunction with present processes, while the electric furnace is anxiously awaited as one means of smelting zinc ores carrying other metals. In short, Western metallurgy is ready to make use of all latest advances in chemistry, electrochemistry, and physical chemistry. The outlook is very bright and this journal will rejoice in recording how this new phase in the evolution of Western metallurgy is carried through to success by Western hospitality.

Toronto Meeting of the American Electrochemical Society.

The twentieth general meeting of the American Electrochemical Society will be held in Toronto, Ontario, Canada, from September 21 to 23 (Thursday to Saturday).

Sessions for the reading and discussion of papers will be held on the morning and afternoon of Thursday, while a smoker with entertainment by section Q is scheduled for Thursday evening.

On Friday morning there will be a session for reading and discussion of papers, probably at some golf club. In the afternoon a golf tournament, etc., will be held, and in the evening a banquet at McConkey's restaurant.

On Saturday morning an excursion will be made to the Canadian General Electric Company's plant at Peterboro, with the return trip through the locks to reach town in time to catch the 5 p. m. boat out of town.

The full program of papers will be published in our next issue. Members having further papers to present, should notify before Sept. 1st either the chairman of the committee on papers, Mr. F. A. Lidbury, Oldbury Electrochemical Company, Niagara Falls, N. Y., or the Secretary of the Society, Prof. Jos. W. Richards, Lehigh University, South Bethlehem, Pa.

The local committee for the Toronto meeting consists of the following gentlemen: Dr. W. Lash Miller, Chairman; Wallace P. Cohoe, Chas. Danforth, Saul Dushman, Thos. C. Irving, Jr., J. A. Kaemmerer, Peter Kirkegaard, Jas. W. Moffat, L. V. Redman, Thos. D. Robertson, A. O. Tate, Geo W. Watts, O. H. Wurster, and any Niagara Falls members as is deemed advisiable.

The Iron and Steel Market.

While August has been rather a dull month as to the placing of new business in iron and steel, and has seen considerable wavering of prices, the trend of production has been slightly upward on account of the fairly heavy buying in the latter part of July and on the whole the developments of the month were reassuring, for the expectation was that this year July and August would prove very unsatisfactory. It may be well to observe that the iron and steel trade in the past two months has not been in as good condition as has been widely reported. Unusual efforts have, indeed, been made in some quarters to spread the impression that the iron trade has been rapidly improving since about June 1, when as a matter of fact there has been little real improvement and the principal cause for congratulation is that what was expected to be a very trying period has been passed without serious trouble. Production, as measured by pig iron, has decreased month by month since March, whereby production in July was at the rate of about 21,500,000 tons a year, about 4,000,000 tons less than the April rate. Late in July idle furnaces began to be put in blast and this increase in number has continued through August. On account of the high humidity always prevalent in these months, the additional furnaces have not served greatly to increase production, and indeed there is reason to infer that they were put in largely to maintain, rather than to increase, the rate of output. Under the system of individual ownership which formerly prevailed, production almost invariably suffered a large decline in the midsummer months, because individual interests could not afford to blow furnaces in and out by reason of such a temporary influence. With the United States Steel Corporation the case is different. Having so many furnaces, it always has a few which are about ready to go out, while it has others which have been relined and can readily be blown in. There is no doubt, however, that August has produced somewhat more iron than July, and it is certain that stocks, both merchant stocks and those of steel works, have been drawn upon at a fairly heavy rate. The steel works stocks are now very light, while the merchant stocks, although decreasing somewhat, are still very heavy.

Prices of finished steel products evinced more irregularity in August than in July, although there were important irregularities in the former month. Practically all the important finished products except rails have been subject to cutting, and indeed there is little semblance left of the controlled market which existed for so long.

An important feature of the market as now made is that the cutting usually proceeds by very small increments. There is no slashing of prices, but rather the order is frequently obtained by a slight concession, a relatively small fraction of a dollar per ton. A definite effort is being made by selling departments to shake off the habit of deep cutting and to substitute the method which has always prevailed in the foreign market of quoting prices very closely, allowing the fraction of a shilling or mark per ton to turn the scale. Another influence is that the cost level has been rather closely approached by the successive declines which have occurred since the market reached a top point late in 1909.

Steel bars for prompt shipment are being sold regularly at 1.20 cents, Pittsburgh, when the specifications are at all desirable, and this price is sometimes shaded slightly on the most desirable orders, the regular or nominal market being 1.25 cents. Plates are being shaded regularly $1 a ton from the regular price of 1.35 cents, and occasionally the lower figure is shaded. Structural shapes are fairly firm, 1.35 cents being only occasionally shaded. Sheets are being shaded $1 a ton in most districts, and in Ohio and Chicago territory are frequently shaded $2 a ton. Tin plates can be purchased at 10 cents per box concession for early delivery.

The mills insist that little if any forward business is being booked at cut prices, and that as order books fill up the tendency to cut will gradually disappear, although it is presumed that as the effort is made to withdraw cut prices a spell of particularly dull business may be experienced. The difficulty is that the mills are not satisfied with running somewhat heavier than a month or two ago, but are individually endeavoring to work up to full operation, and that buyers will not purchase for forward delivery as long as they see that prompt material can be purchased at a concession, so that it is far from evident that the market is really lining up for an actual restoration of the nominal quotations as the real trading basis.

Pig Iron.

Nearly all the Southern producers have withdrawn the price of $10, Birmingham, and are quoting all the way from $10.25 to $11, but there is still a little iron to be picked up at the $10 figure for early delivery, and consumers show no disposition to pay any advance, so that Southern iron has been very quiet. In the North prices are not materially changed, except that in eastern Pennsylvania basic has advanced about 50 cents. We quote: f. o. b. Birmingham, $10 to $11 for No. 2 Foundry; delivered Philadelphia; basic, $14.75 No. 2 x, $15, f. o. b. valley, 90 cents higher delivered Pittsburgh; Bessemer, $15; basic, $13; No. 2 foundry, $13.50; malleable, $13.25 to $13.50.

Steel.

There has been some shading of regular billet and sheet bar prices, which are $21 for billets and $22 for sheet bars, f. o. b. maker's mill, Pittsburgh or Youngstown. Open market sales, however, have been light, consumers being well covered by contracts. Rods are $27, Pittsburgh.

Finished Products.

Regular prices on finished steel, f. o. b. Pittsburgh, except where otherwise stated, are as follows, but are subject to considerable shading, as already noted:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, 1.35 cents for tank quality.

Shapes, 1.35 cents for beams and channels, 3 to 15-inch, inclusive, zees, and angles 3x3 and larger.

Steel bars, 1.25 cents, base; iron bars, 1.25 cents to 1.30 cents, Pittsburgh or Youngstown; 1.27 cents to 1.30 cents, Philadelphia, and 1.20 cents to 1.25 cents, Chicago.

Wire nails, $1.70, base plain wire, 1.50 cents, base.

Black sheets, 28-gage, 2 cents; galvanized, 3 cents; blue annealed, 10-gage, 1.50 cents; painted corrugated roofing, $1.40 per square; galvanized, $2.55.

Tin plates, $3.70 per base box.

The Western Metallurgical Field

Indexing Metallurgical Literature.

Engineers who appreciate the value of indexing the rapidly growing literature on metallurgical subjects will welcome the work recently published in the April, 1911, *Quarterly* of the Colorado School of Mines. The article is by Robert M. Keeney, Fellow in Metallurgy at the School, and consists of an extension of the Dewey decimal system of classification applied to metallurgy, metallography and assaying. The work is exhaustive and gives a complete system of indexing the subjects mentioned.

In the Dewey system the entire field of knowledge is divided into ten principal divisions, numbered from 0 to 9. These are, 0, General Works; 1, Philosophy; 2, Religion; 3 Sociology; 4, Philology; 5, Natural Science; 6, Useful Arts; 7, Fine Arts; 8, Literature; 9, History. Each of these main divisions is further subdivided into ten sections, and the process of decimal division is repeated as often as found necessary. This makes it possible to represent the proper classification of an article or subject by means of figures as shown in the following example: Thus, 669 means Class 6 (Useful Arts), Division 6 (Chemical Technology), and Section 9 (Metallurgy). In carrying out the system, a decimal point is placed for convenience after every third figure. To illustrate further, we have 669.145.412 (Bessemer Converter Tuyeres); 669 as already explained means Metallurgy; 669.1 the metallurgy of iron and steel; 669.14 the manufacture of steel; 669.145 the Bessemer process for manufacture of steel; 669.145.4 the Bessemer converter; 669.145.41 the construction of the Bessemer converter; and 669.145.412 Bessemer converter tuyeres.

While it is clearly impossible to include all the terms used in metallurgy, metallography and assaying, enough can be given to enable the user to assign the proper number to almost any subject that may arise. The decimal system of classification is adapted to the indexing of books, notes, clippings from journals, drawings and catalogs. In indexing current literature where the number of cards becomes large, it will be found convenient to keep the card index in order of the numbers of the decimal system rather than by alphabetical order.

Engineers can use the system in three ways: First, to keep a card index of technical literature; second, to file clippings from journals, notes, drawings and catalogs, without a card index; or third, for a combination of the card index with the filing system, so that a reference to a journal from which there is no clipping at hand can be recorded with the other references. In filing clippings, an envelope 7¾ inches long by 5¼ inches wide is a convenient size.

Mexican Labor Troubles.

Following close on the heels of the revolution, which seriously hampered mining and metallurgical operations in Mexico for several months, comes a period of unrest in labor circles which also threatens to impede progress for some time. Native peon labor is dissatisfied with several conditions under which it works. First, wages are believed to be too low, and an increase of from 25 to 100 per cent is asked from the various mining and smelting companies. The peon wage ranges from one to two pesos per day. Secondly, there is objection to some arbitrary deductions which are made from wages to cover medical attendance and hospital fees. This has been an item objected to in many other places, so that it is not surprising to find it prominent in the present Mexican troubles.

The northern part of the republic is most affected. In the state of Chihuahua there are about half a dozen camps in which serious labor strikes have occurred. The Chihuahua plant of the American Smelting & Refining Company has been closed since June 27 on account of a strike among the peons. The shut-down is apparently for an indefinite period, as the company gave its American employes thirty days' notice of discharge. The camp of Parral, Santa Barbara and Santa Eulalia also are threatened with serious strikes, and as there are many important mines and mills at these centers, the outlook is not encouraging. The opinion exists among those conversant with the country, that no definite settlement of labor conditions can be expected until after the national election in the coming October. The present dissension is not altogether unlooked for as a natural consequence of the recent revolution, and the control of the situation will be one of the problems of the reconstruction period. No doubt professional agitators will try to take advantage of the situation, but it is hoped for the good of the country and the mining industry that peaceful industrial conditions will be reached this year.

Spelter Statistics for 1911.

Statistics compiled by the United States Geological Survey show that the production of spelter or metallic zinc from ore for the first six months of 1911 was 140,196 short tons, a gain of more than 5,000 tons over half the record production of 1910. Of this production, 5,135 tons was made from foreign ore. Spelter stocks were reduced from 23,232 tons to 17,788 tons. Imports remained about the same, but exports were nearly double those of half the preceding year. The apparent consumption of spelter was 135,497 tons, an increase of more than 12,000 tons over half of 1910, but about the same as in half of 1909. The average price of spelter at St. Louis for the period was 5.36 cents per pound, the London average being 0.2 cent less per pound. During the latter part of May and the first of June the average London price was about 0.1 cent higher than the corresponding St. Louis prices. Under this stimulus the May exports of spelter, zinc ore, and dross were largely increased over those of the preceding months.

Camp Bird, Ltd.

This company, which owns the Camp Bird mine in Colorado and the Santa Gertrudis mine in Mexico, has recently made some important changes in its financial policy. It appears that at the time of the purchase of the Santa Gertrudis, in the latter part of 1909, it was necessary to raise between four and five millions of dollars rather quickly or lose the opportunity to purchase the property. In order to do this the company issued debenture stock which it was to redeem at the rate of about $500,000 per annum, with interest amounting to about $150,000 per annum. Although the terms of this issue were rather stringent, nevertheless the directors felt that the mine was too good to lose, and acceded to the terms.

Later developments in the Santa Gertrudis showed that the mine promised far greater profits than were at first realized, and the directors felt that it was too valuable a property to be subject to the mortgage represented by the debentures, and that the company should take steps to redeem the debentures at once and remove the risk of losing the property through some unfortunate circumstance during the next five years. It was therefore proposed to increase the capital of the company by the issue of 750,000 shares of one pound sterling each, being preferential shares drawing 7 per cent, and sharing with the ordinary shares in any surplus remaining after paying the usual 20 per cent on the ordinary shares. By this arrangement the company would remove the possibility of losing a valuable property for a small mortgage before the debentures were paid

in full in 1917, and the only expenditure which would take precedence over dividends on the ordinary shares would be the interest on the new stock, amounting to about $225,000, instead of $650,000 as under the present arrangement.

At a meeting of the stockholders, held in London on July 18, 1911, a majority interest voted to increase the capital as above outlined.

The latest reports from the company's properties are to the effect that the Camp Bird reserves have been increased by substantial bodies of ore, and that on July 1 there was a further profit of about $350,000 in sight. Profits are expected at the rate of about $50,000 per month for the next six months. At the Santa Gertrudis the company has developed about 1,200,000 tons of ore, which is estimated to give a net profit of nearly $6,000,000. In addition to the ore blocked out there is a large tonnage of low grade ore left in the mine as filling. This will be milled with the high grade ore.

Spelter Improvements.

It is reported that the American Smelting and Refining Company is to make improvements at the East Helena, Montana, plant, which will cost about $75,000. The lead smelting business in the West is generally in a poorer condition than in years gone by, and at the same time the competition in some states is keener. The recent entrance of the International Company into the field has stimulated the business somewhat, and other companies will have to meet the competition of a new and well equipped plant. It has been reported before that the East Helena plant was to be closed, but the recent decision to improve it effectually disposes of this rumor.

Improved Market for Lead Ores.

The entrance of the International Smelting Company into the field of lead smelting is expected to stimulate the production of that metal by reason of the reduced rates at which lead ores will be treated. Recently several good contracts have been obtained by this company, and it is certain that they have been secured only at rates advantageous to the shipping companies. The output of the Hecla Mining Company, in the Coeur D'Alene district, Idaho, is to go to Tooele for the next five years. This ore formerly went to the Ohio & Colorado plant at Salida, Col. The entire ore and concentrate production of the Monarch mine, near Murray, Idaho, also has been contracted to the International company. These two sources of lead are worth a great deal to any company engaged in custom lead smelting, as the product is steady and of good quality. The effect of the activity of the International company's agents is to place the lead producer in a more favored position than he has enjoyed for some time past.

CORRESPONDENCE

Fixation of Atmospheric Nitrogen.

To the Editor of Metallurgical and Chemical Engineering:

Sir.—I should be glad to know how Mr. John Woods Beckman can write (Vol. IX, No. 7, page 340): "Recently the Birkeland and Eyde process has been abandoned and the Schönherr process of the Badische Anilin Fabrik has been installed and is extensively operated at Notodden, Norway, where the company's factory is located."

Not a single furnace of Schönherr's system is burning at Notodden, but 35,000 kw is operated there in Birkeland-Eyde furnaces.

University of Kristiania, Norway. Kr. Birkeland.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir.—Your July number contains a letter by Mr. John Woods Beckman, entitled, "The Manufacture of Air Salpeter, a Well-Paying Industry."

While the article as a whole is quite correct, the author is misinformed on some points as regards our company, and as we are the owners of the Birkeland-Eyde patents for the nitrogen fixation process and of the only nitrate manufacturing plant existing in Norway today, we beg to make the following statements:

The Birkeland-Eyde process has not been abandoned; on the contrary, the entire plant at Notodden is employing the Birkeland-Eyde process exclusively.

Our company is capitalized at Francs 41,166,250 of which Francs 4,500,360 is preferred and the remainder common stock.

Norsk Hydro-Elektrisk Kvælstofaktieselskab.
Kristiania, Norway.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir.—With respect to the third paragraph of the letter of Mr. J. W. Beckman in your July issue I wish to make the following correction:

The several electrical furnaces of the Notodden plant which were formerly of the Schönherr (Badische Anilin und Sodafabrik type) have all been replaced recently by the furnaces of the Birkeland-Eyde (Société Norvégienne de l'Azote et Forces Hydro-Electriques) type.

In the large plant at Rjukan to be built jointly by the above mentioned firms, furnaces of both types will be installed in order to definitely determine which of the two types gives the best results in commercial service.

With reference to the chemical part of the Rjukan plant it had been decided some time ago to use the absorption towers of the Badische Company. It has been decided recently, however, that absorption towers of the Birkeland and Eyde type will be used at the Notodden plant.

It may be of interest to remark that the Notodden works, of which the present capacity is from 50,000 to 60,000 horsepower, belong entirely to the Société Norvégienne de l'Azote et Forces Hydro-Electriques, using and exploiting the Birkeland & Eyde patents.

Paris, France. A. S. Garfield.

Continuous Cyanidation.

To the Editor Metallurgical and Chemical Engineering:

Sir.—I have read with great interest Mr. Rothwell's valuable article in your issue of July, 1911, in which he suggests a continuous treatment method of cyanidation. The method he describes so clearly consists of all sliming, continuous agitation —which is apparently appreciated by all cyanide men since its introduction at the Esperanza last year—and continuous decantation.

It will probably interest Mr. Rothwell very much, and also your readers, to know that the essential features of the latter were first used in practice, I think, about nine years ago by Mr. John Randall at the Highland Chief Mill near Deadwood, S. D., and that several plants are now in operation along the same lines.

Mr. Randall grasped the most important point, which is the passing of the slime and its successive use for the dilution of the increasingly rich pulp as the solution passes toward the head of the series. This makes it necessary to precipitate the solution only after it has been several times enriched, thus reducing the amount to be precipitated to a small fraction of that otherwise required. He operated, as did the Denny brothers about the same time in South Africa, with large settling cones and attained fair technical results, although troubled with building up of solids on the sides of the cones, until the property was shut down for lack of ore.

It is unfortunate that Mr. Randall did not publish his results. The only reference I have seen to this is in a review of cyanide progress, by R. W. Gordon, in your issue of January, 1911, in which Mr. Randall is given credit for pioneer work along this line.

The advent of the Dorr thickener, which rendered continuous settling to a very thick pulp feasible, made the matter more

attractive and we suggested it to friends in 1906, but through pressure of other work it was not taken up seriously until 1910, when two plants— one to recover the values entirely by decantation, the other to use continuous decantation to reduce the values to a low point ahead of an Oliver filter—were installed and are now in successful operation.

A number of articles have appeared on the subject in the last year or two, though many writers, such as Mr. F. R. McCann, in *Transactions* of the Mexican Institute of Mining and Metallurgy, July 1909, while appreciating the advantage of continuous decanting, failed to grasp the idea of the reverse movement of the barren solution and so were perplexed as to the disposal of accumulated low-grade solution.

A short article by L. B. Eames, which appeared in the *Mexican Mining Journal* for September, 1910, under the rather misleading title of "Discussion," and which for that reason has probably not attracted the notice it merits, gives a very clear description of "Continuous Decantation." Besides a flow sheet, Mr. Eames has given an algebraic method for determining, from assumed conditions, exactly what the mechanical losses should be, which will be greatly appreciated by any one contemplating the use of this method.

Mr. Rothwell's flow sheet, covering as it does almost all conditions likely to be met in practice, is worked out so well that the cyaniding fraternity will find it of great value. I would call attention to one point, however, and that is that in the flow sheet given if the strength of solution used was two lb. or over, the mechanical loss in gold and cyanide would be less if the barren solution was added to the thickener next to the last. This loss would, of course, be further reduced by the use of cyanide solution instead of water in the batteries.

The remarkable percentage of recovery of dissolved values with the precipitation of a relatively small amount of solution, which should be obtained by this method, is well shown by the following figures, calculated from the conditions assumed:

1. Value per ton assumed to be dissolved from the ore before the pulp reaches the decantation system	$4.00	$4.00	$4.00	$8.00	$8.00	$8.00
2. Moisture in pulp discharge from thickener, per cent	37.5	50	50	50	50	50
3. Tons solution precipitated to $.02 per ton of ore	1.8	3.	4.	3.	4.	3.
4. Total thickeners used, the underflow into the last being diluted only with an equal amount of water to secure the minimum loss of cyanide	4	4	4	4	4	5
5. Loss of dissolved value per ton of ore	$ .04	$.044	$.026	$.080	$.044	$.026
6. Percentage of dissolved value saved	98.9	98.9	99.3	99.0	99.4	99.6

Undoubtedly within a short time there will be data of actual operations by this method published to show how closely the theoretical results have been attained in practice.

JOHN V. N. DORR.

Denver, Colo.

Ferro Titanium.

To the Editor of Metallurgical and Chemical Engineering:

SIR.—In your issue of July, 1911, page 348, there is an article by Dr. Hans Goldschmidt, in which he speaks of alloys produced by his aluminothermic process, and contrasts them with other alloys.

Regarding Ferrotitanium, he says: "This carbon-free ferrotitanium, containing 25 per cent of titanium, is used now quite extensively as an addition to steel for making rails and rolls. This ferrotitanium made by the aluminothermic process dissolves very easily and uniformly in the molten steel. It has a small intentional content of aluminium as this increases its solubility.

"The experience of metallurgists has shown that as a rule carbon-free metals dissolve more easily than metals containing carbon, since the latter contain carbides—that is chemical compounds with carbons."

It has been my privilege to have considerable experience with ferrotitanium alloys, in both steel and iron, which has been chiefly gained with the so-called 10 per cent and 15 per cent alloys of the Titanium Alloys Manufacturing Company. Based on this experience, these alloys can be thoroughly recommended. As is well known, these domestic alloys are not carbon-free, as they contain from 6.00 to 7.50 per cent total carbon, and so would appear to come under Dr. Goldschmidt's classification of alloys containing carbides. This, however, is not the case, for almost the whole of the carbon is present in the free or graphitic condition. The carbon in chemical combination varies from not more than 0.1 to 0.25 per cent, so that the amount of titanium carbide present, if any, must, therefore, be very small. The action of titanium, is some respects seems to be a good deal like that of silicon decomposing the carbide of iron, and precipitating the carbon as graphite.

When the 10 per cent or 15 per cent alloy is added to a bath of molten steel solution quickly takes place. Part of the graphite probably unites with iron forming carbide of iron, while a part rises quickly to the surface because of its low specific gravity, and is lost in the slag.

This opinion is supported by the results obtained in practice, for the carbon in the final steel when these alloys are used is generally slightly raised, but not to the amount expected if all the graphite were absorbed. It is extremely improbable that at the temperatures worked with, any titanium carbide is formed.

The Goldschmidt alloy when analysed shows that a considerable amount of aluminium is present and most probably unavoidably. This is present only in traces in the Niagara Falls alloys. In one sense its presence may be beneficial, but in another it is harmful. It may help the solubility of the low-carbon when added to the molten bath, and in this respect its action is necessary and helpful. After fusion, most of the aluminium will be oxidized to dry powdery alumina. This will tend to slag off, together with the oxide of titanium and other slag-making materials. Its action will be to decrease the fusibility of the slag, making it liable to be held in the metal, and so tend to defeat one of the chief advantages of using ferrotitanium, namely, a final clean solid steel or iron.

G. B. WATERHOUSE.

Lackawanna Steel Co., Lackawanna, N. Y.

Expropriation of Mining Rights.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I invite the friendly criticism of your journal on my proposed new (old) legislation permitting expropriation of mining rights (even *in invitum*) by strangers to the title in the soil. Some such legislation seems demanded and essential to the proper development of the mineral resources of States like those of the Atlantic seaboard, where obtains the ancient English common-law theory that ownership of the soil carries with it the title to the centre of the earth.

Rockland, Maine. C. VEY HOLMAN.

[As noticed in our May issue, page 278, a paper was presented by Mr. Holman at the last meeting of the American Electrochemical Society on the advantages of Maine for electrochemical industries, giving details of the available water powers and mineral resources of Maine. This paper was pub-

lished in full in Vol. 19 of the Transactions of the Society, page 351. Mr. Holman's argument is that the development and exploitation of these mineral resources seems wholly impossible under the present laws, owing to the unprogressiveness or unwillingness of their nominal owners, the holders of the land below which they lie, to undertake their proper working. For this reason Mr. Holman intends to have a bill introduced at the next session of the Legislature of Maine, providing a method by which mining of ore bodies in the lands of owners who are unable or unwilling themselves to develop them can be legally undertaken by strangers to the title who are willing to assume the risks. It is interesting to note in this connection that in his address on the mining industry and the public lands, delivered on Aug. 9 at the Michigan College of Mines, the Director of the U. S. Geological Survey, Dr. George Otis Smith, recommended that there should be made possible by legislation "the separation of surface and mineral rights wherever the two estates have values that can be separately utilized."—Editor.]

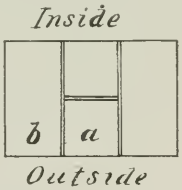
Flow of Heat Through Furnace Walls.

To the Editor of Metallurgical and Chemical Engineering:

Sir: In the recent Bulletin No. 8, by Messrs. Ray and Kreisinger, issued by the Bureau of Mines, and noticed briefly in your July issue, page 382, the conclusion is drawn from their experiments that a solid wall is a better heat insulator than one of the same total thickness having an air space not filled with any material, especially when this air space is near the hot side; also that when such an air space is filled with a loose granular material it insulates better than a solid wall.

These conclusions, if correct, seem to be quite important and valuable, although the most important one, namely, the one referring to the better insulation of loose, granulated material, is not new.

It seems to the writer, however, that the subject needs further investigation, as there are some discrepancies which seem to require an explanation. It is a well known fact, known probably to all blacksmiths, that when an iron bar which is cracked transversely is heated on one side of the crack, it may become red hot up to the crack, while it is still black on the other side. The crack may be so small as to be invisible to the eye, in the cold piece; in fact, it is believed that this phenomenon has been used to indicate the existence of invisible cracks.



This means, of course, that there is a very pronounced drop in the heat gradient at the crack, and this in turn means that the resistance to the flow of heat across the air space of the crack (or the counterpart of resistance when the heat is transmitted by radiation) is enormously greater than that of the iron. This is apparently in direct contradiction to the conclusions drawn in that bulletin. An air space is a wide crack.

A possible explanation of the discrepancy may be that the iron has such a very good thermal conductivity that the air space is poor relatively to it, and that the conductivity of fire brick is so very much poorer than iron that it is better than air. This, however, seems hardly likely, for it is evidently the air spaces in the interstices of the loose granular material that gives it its higher resistance. Moreover, the writer has an indistinct recollection having noticed that a glass rod with a crack behaved like the iron one, and the thermal resistivity of glass is of the same general order as that of refractory materials.

Moreover, the writer noticed recently in an electric furnace that the temperature of the brick at *a* in the above sketch, Fig. 1, was considerably cooler than that at *b*; the difference was so great that it was easily noticeable to the touch. This was no doubt due to the joint which separated the one from the inside of the furnace. If such a cheap and simple thing as a joint is such a good heat insulator, and this appears to be so, it would be unfortunate if the conclusions drawn in that bulletin were to be interpreted to the contrary.

It might be interesting to try heating the end of a transversely cracked fire brick or rod of insulating material to see whether it would behave like the iron bar. It would probably have to be done in the dark; moreover, the cooler end ought to be water cooled and the hot end ought to be heated quickly, for in course of time the heat would no douht find its way to the colder side by passing around the end of the crack. Mere differences of temperature alone are not measures of thermal resistances any more than voltages are measures of electrical resistances.

Fig. 2

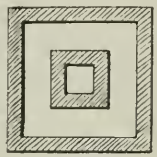


Fig. 3

If the electrical conductivity is any measure or indication of the thermal conductivity it may be of interest to note here that the writer has repeatedly drawn an alternating-current arc from the surface of a hot magnesite fire brick which formed a part of the furnace wall, the metallic charge in the furnace being the other pole of the circuit. The magnesite was scarcely visibly red.

According to the above-mentioned bulletin, filling an air space with loose granular fire brick gives that space an effective resistance which is greater than either of its two constituents, air and fire brick, would have by themselves. This seems to call for an explanation. Perhaps it lies in the resistances at the surfaces of contact of air and fire brick, as this surface is enormously greater for granular materials.

The following is another curious deduction from the formula in this bulletin which leads to apparently absurd conclusions, although they may be correct. Let Figs. 2 and 3 represent two double-walled furnaces, in the form of cubes, which are alike in every respect except that the air space within the walls in Fig. 3 is double that in Fig. 2, the thickness of each of the two parts of each of the brick walls being the same in both.

Let it be assumed for a first trial calculation that the leakage of heat is the same in both. The temperatures at the two bounding surfaces of the air space will then, of course, be the same, as the amount of heat radiated is independent of the thickness of the air space. But the cross section of the outer wall perpendicular to the flow of heat is much greater in Fig. 3 than in Fig. 2; suppose it to be twice as great in this particular case, then the drop of temperature in it would be only half as great as in Fig. 2 because the flow is the same, hence the temperature of the outside of the furnace would be considerably greater in Fig. 3 than in Fig. 2; the larger one would be the hotter. But with a higher temperature of the outside surface more heat would pass into the air; besides, the surface is now also larger, which still further increases the heat flow.

It follows, therefore, that the heat flow or leakage will not be the same in the two cases, as was assumed at the start, and that it will be *greater* for the furnace shown in Fig. 3. In other words, a larger air space insulates worse than a smaller one, which is apparently paradoxical. Yet it seems to lead to the conclusion that the thinner the air space the better the insulation; hence a joint between two abutting bricks ought to be a good insulator.

The above illustrations seem to show that it is not safe to jump to conclusions too quickly, as the problems may not be as simple as they appear at first sight.

Philadelphia, Pa. Carl Hering.

The Counter-Current of Continuous Agitation, Decantation, and Dilution Applied to the Cyanide Process

BY JOHN E. ROTHWELL.

The present practice in the treatment of a gold and silver ore by the cyanide process, when using the agitation method, requires the mixture of a barren cyanide solution with the ore at its highest value; then by the continued agitation of this mixture, dissolution of the gold and silver is accomplished and the metal transferred to the solution, the rate of dissolution proceeding more slowly as the point of chemical equilibrium is approached. Thus it is necessary to agitate the mixture for such length of time as required to obtain the highest economical transfer of the metal into solution, and in some cases it means a retreatment of the charge after the removal of the first solution, in order to obtain a satisfactory recovery of the metal content of the ore.

The ideal and logical method of continuous treatment and operation is the reverse of this method—that is, the solvent lowest in metal value and highest in free cyanide should be applied to the ore after the greater value in metal has been dissolved. This solution is then separated from the ore and applied to ore of higher value, and so back up the line of treatment until the highest grade of solution is in contact with the ore at its maximum value, after which it is separated and passed to the precipitation department, where the metal content is precipitated and recovered and the barren solution returned to the mill system to go through the work of gathering up value again in the same sequence of operations.

It is known that if cyanide solution has approached the point of chemical equilibrium on any single quantity of ore, and is then withdrawn and another solution containing free cyanide, but not necessarily barren in metal value, is brought in contact with this ore, a further dissolution of the metal takes place that it was not possible to obtain with the first solution, and that in the repetition of this sequence of operations the point of saturation is approached.

Based on this condition, which in reality is simply the application of the "phase rule," I have designed a method for carrying out these operations in a mechanical and continuous manner.

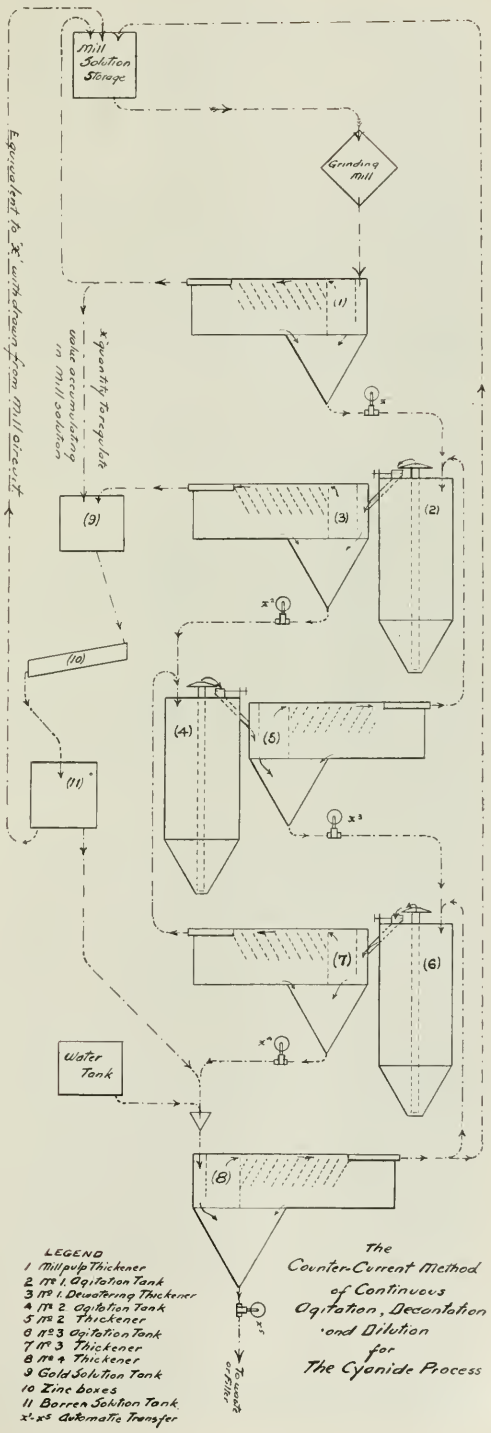
The method involves no change in the class of equipment employed in the ordinary continuous method of agitation, decantation and dilution as illustrated in a former article* of mine covering the continuous treatment of an ore, and as generally employed in mills using agitation tanks and dewatering thickeners. The only requirement is a re-arrangement of this equipment.

Reference is made to the graphic flow sheet accompanying this article, which illustrates the new arrangement, and the counter-current flow of solvent through the stream of ore. In this illustration, it will be noted that instead of placing the agitation tanks in continuous series, followed by the continuous decanting and diluting tanks, we place the agitation tank between the thickeners. The agitator then receives the thickened pulp from the thickener ahead of it, and the decanted solution from the thickener following it; thus it will be observed that the solvent is taking up value from the ore of lowest value and passing it up to the tank containing a higher value, while the ore is losing value as it passes down through each tank of the series toward the discharge.

The highest efficiency is obtained by having as thick a pulp as possible in the underflow, so that a minimum of dissolved value is contained in the moisture passing with the solids; and the greatest dilution is obtained from the counter-current flow, also the point of chemical equilibrium is removed as far as possible.

In describing the method and equipment I will not take up

*This journal, Vol. IX, No. 7, July, 1911.



the preliminary work of the preparation of the ore, as this will have to be suited to the requirements of the ore, with reference to amalgamation, concentration, fine grinding or preliminary treatment by oxidizing or other solutions to prepare the ore for the subsequent treatment in the cyanide solution.

The flow sheet shows a simple ore ground in a weak cyanide solution, and a comparatively small extraction of the metal values during the preparation.

For the operation of the counter-current method the agitation tank that can be used to the best advantage is the type in which the rising current of pulp is carried up through a central tube. At the top of this tube is a distributor, over which the flow of pulp passes, and a proportional part of this flow is cut out with a sampling vane located above the surface of the pulp in the tank. This position of the sampling vane forces the incoming flow of pulp and solution to pass from the top to the bottom of the tank before any part of it can possibly pass through the vane to the thickening tank, thus insuring a thorough mixture and agitation. The distributor may be of the rotating type or simply an inverted cone of definite diameter at the base. The sampling vane is located just below the perimeter of the cone, or in the concentric path of the rotating distributor, thus a uniform sample of the content of the tank is passed to the thickener. The adjustment feature of the vane permits of controlling the level of the pulp in the tank so the amount of outflow is equal to the inflow.

The thickener tank that works into the method is the one shown in the diagram, as it permits of a close and short connection to the agitator. This is a tank with a large cone and extended rectangular settling tank with inclined baffles that give it a very large settling area, in which the finest solids have a maximum time to settle and give a clear overflow at the weir. The weir is placed at the opposite end of the tank from the feed and the solids that settle to the flat bottom of the tank are moved to the cone by a slow-moving reciprocal rake. The large cone forms a receptacle for the storage of considerable thick pulp, thus insuring a very thick product, also a place in which any fluctuations in the flow from the grinding mill can be taken care of and the flow through the rest of the system maintained uniform.

The automatic transfer is a special type of diaphragm pump, capable of handling a very thick pulp, and may be regulated both in speed and stroke to give a definite quantity of feed in a unit of time.

In the arrangement of the series of thickeners and agitators to obtain the greatest economy in power, the tanks (2) and (3) of the series will be the lowest in the mill, the top of (4) and (5) will be about 2 ft. above the top of (2) and (3), while the top of (6) and (7) will be about 2 ft. above (4) and (5); (8) will be only high enough for the overflow to go by gravity into (6). The water tank and (11) will be located above (8); a meter placed in the outflow pipe from (11) and water tank regulates the flow of diluent into the series and registers the quantity passing. The automatic transfer takes advantage of the hydrostatic head in the thickener, thus reducing the lift necessary to discharge the pulp into the following agitator. The thickened pulp being the smallest quantity to be handled, means the minimum of power required in the system. If we figure, for example, a plant treating 100 tons a day, and the transferred pulp contains 50 per cent moisture, we then have approximately 3.26 cu. ft. flow per minute, equal to approximately 280 lb. to be raised an average height of about 6 ft. at each transfer, or 1680 ft.-lb. With the five transfers shown in the flow sheet this would be 8400 ft.-lb. Allowing 100 per cent for friction, we require very little over ½ hp for the whole transfer.

The arrangement also contemplates the overflow from (3) by gravity into (9), thence to (10) if zinc boxes are used; the barren solution is then pumped to (11). The elevation of this tank is only enough to flow back to (8) and need not exceed 30 ft. If we follow the example of 100 tons being treated, and take the quantity of solution precipitated at 3 tons for one of ore, *i. e.*, 300 tons per day, this will require, if a pump is operated continuously, 12,500 ft.-lb. per minute, which, allowing 100 per cent for friction, is about 2/3 hp.

If we take the power required to operate the air compressor for agitation in the agitation tanks at 5 hp per tank, this being about an average in the usual practice at the present time, and using three agitation tanks, we have 15 hp for this purpose. Allowing liberally for friction and contingencies, 20 hp will be ample for the whole work, or about 1/5 hp per day-ton for this department of the mill. Present practice is considerably in excess of this figure.

For the flow of the ore and counter-current flow of the solvent, reference is made to the diagram.

The following descriptive flow sheet will give the method of operation:

(1) Mill pulp thickener; receives all the flow from slime classifiers, overflow goes back to mill solution tank, except the small amount sent to (9) for precipitation if there is a tendency on the part of this mill solution to build up in value. This solution in some cases instead of being sent to (9) might be sent to (7) or (5), whichever tank should happen to contain a nearly equal amount of value per ton of solution. The underflow will be transferred in definite quantity and specific gravity to (2).

(2) No. 1 agitation tank; receives thickened pulp from (1) and counter-current solution from overflow of (5), discharges content continuously through sampling vane into (3).

(3) No. 1 dewatering thickener; receives flow through sampling vane from (2), clear overflow goes to (9), thick underflow by automatic transfer to (4).

(4) No. 2 agitation tank; receives thick plup from (3) and counter-current solution from overflow of (7), discharges content in equal quantity as received, through sampling vane, into (5).

(5) No. 2 dewatering thickener; receives flow continuously through sampling vane from (4), clear overflow goes to (2), while thickened underflow goes through automatic transfer in definite quantity and specific gravity to (6).

(6) No. 3 agitation tank; receives thick pulp from (5) and counter-current solution continuously from overflow of (8), discharges content in equal quantity as received, through sampling vane, to (7).

(7) No. 3 dewatering thickener; receives flow continuously through sampling vane from (6), clear overflow goes to (4), while thickened underflow goes through automatic tranfer in definite quantity and specific gravity to (8).

(8) No. 4 dewatering thickener; receives thickened pulp underflow from (7), mixed with barren solution from (11), plus water to make up moisture passing into the system from (1); overflow to the amount to maintain quantity in counter-current goes to (6), balance of overflow goes to mill solution storage; underflow goes by automatic transfer to waste, if sufficiently reduced in value, otherwise to an additional thickener, and this thickener would receive the flow from (11) and the water tank, instead of (8). The overflow would then go in part to (8) and the balance to the mill solution tank, while the overflow from (8) would all go to (6).

(9) Gold solution storage tank; receives overflow from (3), and delivers as required continuously to (10).

(10) Zinc boxes or zinc dust precipitating equipment; receives solution from (9), delivers barren solution to (11).

(11) Barren solution storage tank; receives solution from (10), discharges through regulating meter to (8) or subsequent thickener.

If the thickened pulp from the last dewatering thickener carries an amount of cyanide solution that would involve an excessive loss, this pulp can be washed on a pan or rotary type of vacuum filter and the cyanide solution recovered and returned to its proper place in the circuit.

The excess of cyanide can also be recovered to a high per-

centage by using two or three additional thickeners, and giving a dilution wash with water, and counter-current the wash against the solids, the overflow from the first tank going to the mill circuit to replace the amount drawn from it into the agitation circuit. A close regulation will obviate any accumulation of solution, particularly if the ore treated settles naturally to a fairly dry pulp.

There are many advantages apparent in this system as a whole over the present method of operating, the important ones of which are as follows:

The dissolved values are recovered from the system at the point farthest removed from the discharge of the tailings; thus it is impossible for a charge of pulp containing high value in solution to be thrown away by an imperfect wash or carelessness or neglect on the part of the attendant operating a filter as in the present method.

The amount of solution carried in the circuit is less than in the present method, while at the same time any unit of ore is in contact with a larger volume of solution than at present.

By proper regulation, a minimum of solution of constant maximum value is sent to the precipitation department, so that there is no occasion to carry two or more circuits for precipitation purposes.

There will be no accumulation of solutions, from which values have to be precipitated and the solution thrown away.

The power required for operating will be less and the equipment less for given tonnage capacity, thus the first cost will be less for installation per day-ton treated.

The counter-current system can be controlled in all its operation from a central point, which registers and records the exact flow at all times and sounds an alarm if any stoppage occurs or a danger point is reached that would throw the regular operation out of equilibrium.

A complete control of the process is possible by taking samples of the pulp and solutions passing, and making the necessary determinations of value considerably in advance of the time of discharge of the pulp as waste. As the mechanical operations are simple and automatic, the labor required around such a plant is a minimum, so that the personal equation plays a very small part in the whole.

The system may be applied to the treatment of ores other than gold and silver where a hydro-metallurgical method can be applied.

Denver, Colo.

New Method for the Rapid Determination of the Total Carbon in Iron, Steel and Ferro-Alloys.

By A. L. Queneau.

It is of great importance to be able to obtain rapid as well as accurate total carbon determinations in operating steel open-hearth or electric furnaces. The colorimetric method of Eggertz was formerly the quickest known, though it is not as reliable as the usual combustion method in a porcelain tube in a stream of oxygen.

The method herein described is due to M. De Noully, chief chemist of the St. Chamond Steel Works, in France. It permits of the same accuracy as the standard combustion method while giving the results in about 5 to 7 minutes. Furthermore, the apparatus used is extremely simple and can be installed independently of a regular laboratory in close proximity to the steel furnaces.

The method is applicable to the various grades of pig iron, steels and ferro-alloys; the accuracy of the results being entirely independent of the presence of nickel, chrome, tungsten, molybdenum, vanadium, titanium, sulphur or phosphorus.

Principle of the Method.—The prepared and weighed sample of the steel or ferro is burned in an atmosphere of oxygen, the combustion being started by means of a small arc flashed through the steel sample. The resulting CO_2 is absorbed in a known volume of standard NaOH solution, the excess of free NaOH being titrated by means of a standard sulphuric acid solution; phenolphthalein is used as indicator.

Solutions Used.—Alkali Solution.

NaOH	4 grams
H_2O	1000 cc.

Indicator.

Phenolphthalein	1 gram
Alcohol, 90°	100 cc.

Sulphuric Acid Solution.

H_2SO_4	4.083 grams
H_2O	to make 1000 cc. of solution

The acid solution contains 3.333 grams of SO_3 per liter; 1 cc. therefore corresponds to 1 mg. of carbon. As usual, the acid solution is standardized with a $BaCl_2$ solution.

Apparatus.—The apparatus consists of:

1. A liter Erlenmeyer flask of Jena glass, closed by a rubber stopper with 3 holes, through which pass 2 copper electrodes

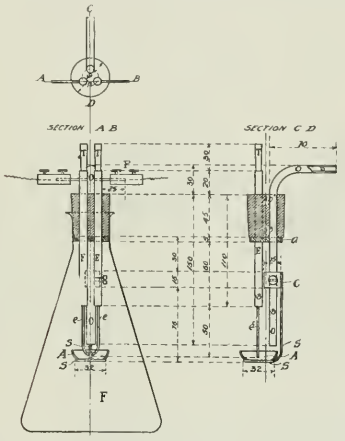


FIG. 1.—DIMENSIONS OF APPARATUS

F—Erlenmeyer Flask of 1 Liter.
B—Rubber stopper with 3 holes.
a—Asbestos disc to protect the rubber stopper from any projections during combustion.
E—Electrode supports.
e—Electrodes.
T—Fiber handles allowing to vary at will the electrode gap.
P—Current terminals.
O—Copper tube for the oxygen.
C—Adjusting sleeve with set screw.
S—Support for the asbestos boat.
A—Asbestos boat.

and a copper tube. To the copper tube is attached a copper wire; forming a loop to support an asbestos boat and a basket of copper wire cloth placed under the asbestos boat. The basket is to receive any material that may escape in case of the asbestos boat being bored by a too violent combustion of the metal sample. The copper tube terminates a short distance above the boat and serves to bring into the flask a supply of oxygen and directly over the material under analysis.

2. A glass bottle reservoir for the oxygen, which is maintained at a slight pressure, 30 to 40 cm. of water. The water is rendered slightly alkaline with KOH in order to free the oxygen of whatever traces of CO_2 may be present in the gas as supplied.

3. A small glass bottle forming a mercury trap to prevent any gas passing back from the Erlenmeyer flask in case of too intense combustion.

The asbestos boats are readily made from asbestos paper 1 or 2 mm. in thickness, from which discs, 50 mm. in diameter, are cut. These discs, slightly moistened, are pressed by means

of a bronze former, 25 mm. outside diameter, in a steel die, 30 mm. inside diameter. The boats are thoroughly dried and are also calcined in a muffle furnace to decompose any carbonates that may be present in the asbestos.

Preparation of the Sample.—Whenever possible the samples are obtained by drilling at high speed with a suitable twist drill; thin voluminous drillings are desired; dust and needle-like drillings are to be avoided if possible.

1. The hard or tempered steels are crushed and passed through 100 mesh.

2. All grades of pig iron are crushed and passed through 80 mesh.

3. All ferro-alloys are crushed and passed through 120 mesh.

For soft steels a good combustion is readily obtained without the addition of any oxidizing agents; for all other metals it is necessary to add a certain amount of PbO_2 or Bi_2O_4 to the samples, as will be shown in the table given below.

With all materials containing over 1.5 per cent of carbon in order to secure a complete combustion it is necessary to add to the sample a certain amount of fine drillings from an extra

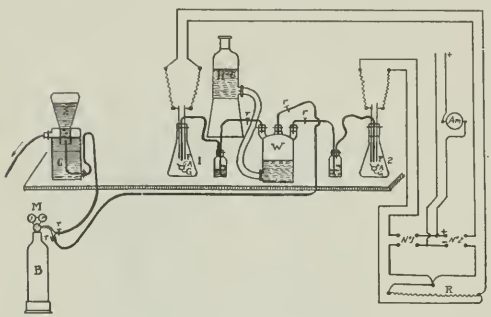


FIG. 2.—ARRANGEMENT OF APPARATUS.

A—Asbestos boat.
C—Copper wire cloth basket or clay dish to collect any oxides that may escape in case of a rupture of the asbestos boat.
F_1 & F_2—1 Liter Erlenmayer flasks.
B—Oxygen tank.
M—Reducing manometer.
C—Water tank used in filling the flasks with oxygen.
W—Wolf bottle used as oxygen reservoir.
H_2O—Water bottle to give the necessary pressure to the oxygen in W.
Hg—Mercury traps.
Am—Ammeter.
r—Cocks.
R—Electric resistance.
No. 1 & No. 2—Two-throw switch with fuses.

soft and pure steel (.05 to .07 C.), due corrections being made in the results obtained for the additional carbon introduced, the determination of the carbon contents of the extra soft steel employed having been made once for all.

Materials.	Wt. of S'ple gram.	Wt. of PbO_2 gram.	W'ght of Soft Stl. gram.
Steels with less than 1.5% C....	1 or 2.	.100 or .200[1]	
Hard steels or steels with over 1.5% C.	1.	.200	
Pig iron	.500	.200	1
Ferro-chromes with 6 to 10% C.	.250	.200	2
Ferro-chromes with 1 to 5% C..	.500	.200	1
Ferromanganese	.250	.200	1
	or .500	.200	2
Ferrosilicons, 10 to 90%.....	.500	.200	1
Ferro-molybdenum	.500	.500[2]	1
Ferro-tungsten	.500	.100[3]	1

[1]With soft steels it is not necessary to add any PbO_2 unless the borings are coarse and irregular in size.
[2]With less PbO_2 some molybdic acid is volatilized and is absorbed by the NaOH solution.
[3]With more than .100 of PbO_2 the combustion is too intense and the asbestos boat becomes bored thereby.

If any extra soft steel borings are required these drillings are spread on the bottom of the boat and on top of the layer thus formed is placed the mixture of the pig iron or ferro with the lead peroxide.

The above table will enable one to obtain the complete combustion of all products, even gray iron containing a large amount of graphite.

For the analysis of all products with more than 1.5 per cent of C it is recommended that the NaOH solution be diluted to 500 cc. in order to make the end reaction more sensitive and also to bring back the pink color of discoloration by the addition of a known quantity of the NaOH solution. With high carbon contents it is also well to take 200 cc. of the NaOH solution, instead of 50 cc., in order that there may be a large excess of free alkali to absorb the last traces of the formed CO_2.

The results are not affected in the least by the presence of sulphur. A test was made with samples containing .80 per cent of S. In the same manner the presence of phosphorus is not detrimental; phosphoretic pig with 2 per cent P gave accurate results with this method. The sulphates and phosphates formed remain *in toto* in the asbestos boat, from which they may be collected and then determinations made.

Manipulations.—The asbestos boat containing the sample to be analyzed is placed upon its support, the height of the boat being so adjusted that its edge level is from 5 to 10 mm. from the level of the end of the supporting copper tube. The copper electrodes are placed with the free extremities buried in the sample and about half way therein. The rubber stopper with its adjuncts is placed in the neck of the Erlenmayer flask, a careful closure of it being made without disturbing the boat or its contents. The flask has been previously filled with oxygen gas and has also received 40 or 50 cc. of the NaOH solution. The various connections are now made and the cock *r* is opened, thus establishing communication with the oxygen reserveir *W*; the electric current is then made to pass across the gap of the electrodes by closing the double-throw switch. The resistance *R* is such as to permit, with a given source of electric current, the passage of 12 or 15 amp when the electrodes are in short-circuit.

As soon as the combustion has begun the current is cut off, the metal continuing to burn until complete combustion has taken place.

In the event of the combustion not starting properly the gap between the electrode points is changed by means of the fiber handles *T*, while the current is sent in repeatedly by operating the switch.

If the combustion is too rapid the cock *r* is closed, cutting off the access of the oxygen.

The combustion being complete, the cock *r* is closed, the flask is disconnected and it is cooled by plunging it in water; when cooled it is rapidly opened, the stopper with the boat removed and a plain stopper is substituted. The flask and its contents are thoroughly agitated for 10 seconds and the free NaOH is titrated by means of the sulphuric acid solution. As a check on the alkali contents of the NaOH solution a fresh sample is now titrated and the difference of the two titrations, the same volumes of the alkali solution having been taken, corresponds to the NaOH absorbed by the CO_2 formed. If the acid solution contains exactly 3333 grams of SO_3 per liter, 1 cc. of it corresponds to 1 mg. of C.

By taking the difference of the two NaOH titrations it is seen that no correction need be made for any carbonation which is bound to occur in the NaOH solution.*

The Arizona Copper Company, Ltd., produced during the half year ending March 31, 1911, 9,133 tons of Bessemer copper. The quantity of copper bearing material, ore and concentrate, smelted during the same period was 80,643 tons. The copper yield was 11.32 per cent on the gross tonnage treated.

Bulletin de la Société de l'Industrie Minerale, November, 1910; July, 1911.

Refining of Silver and Gold Alloys

By EDWARD F. KERN, PH.D.,
From Lecture Notes Prepared for the School of Mines of Columbia University.

One of the final products which is obtained by the treatment of anode sludge by the cupellation of retort-bullion from the Parkes desilverization process, by the cupellation of the final "mother-metal" of the Pattinson process, and by the treatment of silver-gold ores, is an alloy composed chiefly of silver and gold, and associated impurities such as copper, selenium, tellurium, and traces of lead, bismuth, etc.

If the alloy contains an appreciable amount of one or more of the following metals: lead, bismuth, antimony, arsenic, selenium and tellurium, it should be purified as completely as possible, as these metals interfere with and increase the cost of separating and purifying the silver and gold. This is accomplished by either oxidation in the cupellation furnace, or else by melting the alloy with an oxidizing flux, such as potassium nitrate.

If the alloy contains over 75 per cent of silver, it is known as doré bullion. It is refined and the gold recovered by either the acid-parting process or the electrolytic-parting processes.

If the gold content of the alloy is over 25 per cent, the direct treatment by either of these processes cannot be economically accomplished. When this is the case, the alloy is melted with a sufficient amount of impure silver (or doré bullion) so as to dilute the gold content of the resulting alloy below 25 per cent.

In order to accomplish rapid parting of the silver from the gold, the doré bullion is cast into thin plates of suitable size and shape for treatment. The thinner the plates, the more rapid the parting is accomplished.

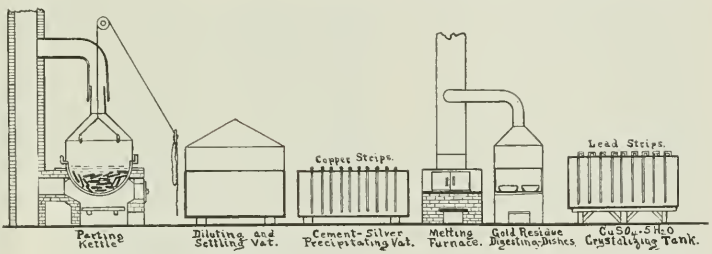


FIG. 1.—SECTIONAL SKETCH OF ACID PARTING PLANT.

Parting of Doré Bullion with Boiling Concentrated Sulphuric Acid.

The doré bullion, as thin plates containing 75 to 98 per cent silver, 25 to 1 per cent gold, and 1 to 2 per cent copper, selenium and tellurium, is placed in cast steel or cast-iron kettles and digested with boiling concentrates H_2SO_4 (66° B) for 15 to 20 hours, until only fine particles are left. (If the alloy contains over 40 per cent gold, it dissolves very slowly.) The reactions for dissolving are:

$$2\,Ag + 2\,H_2SO_4 = Ag_2SO_4 + SO_2 + 2\,H_2O.$$
$$Cu + 2\,H_2SO_4 = CuSO_4 + SO_2 + 2\,H_2O.$$

The usual capacity of the kettles is from 10,000 to 20,000 Troy oz. of doré bullion per 24 hours; 2.5 lb. of concentrated H_2SO_4 are required per 12 oz. of bullion.

As $CuSO_4$ is only slightly soluble in concentrated sulphuric acid, the copper in the doré bullion must be less than 8 per cent, otherwise crystals of $CuSO_4$ will coat the metal and prevent it dissolving. Ag_2SO_4 is, however, soluble in one-quarter its weight of boiling concentrated H_2SO_4; whereas, in dilute (5 per cent) H_2SO_4 at 90° C. it is soluble in 25 parts, and in acid of the same dilution at 20° C. one part of Ag_2SO_4 is soluble in 100 parts.

During the dissolving of the silver, a large amount of SO_2 is formed; the kettles are, therefore, covered with an iron hood which is connected with a ventilating chimney. The heating of the kettles is done by means of a fire beneath.

When the silver is all dissolved, the fire is withdrawn from under the kettle, and a small amount of cold dilute sulphuric acid (specific gravity 1.6) is added to the hot solution. The addition of the cold dilute acid causes the precipitation of some Ag_2SO_4, which carries with it any suspended particles of gold. If much copper is present, the addition of cold dilute acid is not necessary, as some $CuSO_4$ will precipitate when the solution cools, collecting the suspended particles of gold in the same manner as the Ag_2SO_4. In some plants the hot solution is ladled into a second, heated, kettle, in which the suspended finer residue is allowed to settle. This procedure gives a more rapid recovery of the major portion of the gold, as the coarser residue which remained in the digesting kettle may be under final treatment during the time that the finer particles are settling from the solution which is in the settling kettle.

The hot solution, as soon as clear, is siphoned into about 20 *times its volume of hot water* in a covered lead-lined vat. Any fine gold, which may have been carried over, is allowed to settle and the clear dilute hot solution is run into another lead-lined vat in which are suspended strips of copper, which precipitate the silver as "cement silver," $Ag_2SO_4 + Cu = CuSO_4 + 2Ag$. The solution is kept boiling by steam. The end of precipitation of the silver is determined by testing with HCl; if silver is in solution, AgCl forms.

The cement silver is washed free of copper sulphate, dried and melted. If any selenium or tellurium are present, some KNO_3 is added to oxidize and slag them. The refined silver is cast into bars weighing 1000 to 1200 Troy oz., which assay 998 + fine. The slag is added to charge in the smelting furnace, or placed on a charge of metal in the cupel furnace for the recovery of its silver content.

The $CuSO_4$ solutions are evaporated to crystallization for $CuSO_4.5H_2O$, and the mother liquors are re-evaporated to crystallization several times; the crystals are refined by dissolving in water and recrystallizing. The final mother liquors are evaporated to 55° B in lead pans, then transferred to an iron kettle and evaporated to 66° B. The concentrated acid is allowed to cool and the residue to settle; the acid is then siphoned off and used for parting doré bullion. The residue is collected and sent to the smelting furnace, or else put in the cupel furnace.

The *residue of gold* from the parting of the doré still contains some silver. It is transferred to a small iron kettle and again digested with boiling concentrated H_2SO_4 for four to six hours. The solution is poured off and added to the large parting kettle. The residue of gold is washed with hot water, transferred to a porcelain dish and digested with boiling dilute HNO_3 in order to remove the last traces of silver and copper. The solution is poured off and the silver precipitated on metallic copper. The residue of gold is well washed with hot water, dried and melted with a flux of borax-glass in a graphite crucible, and cast into bars (998 + fine). The slag is smelted, or else placed in the cupel furnace.

Some metallurgists dispense with the last HNO_3 digestion, and others digest first with boiling dilute HNO_3, after which the residue is finally digested with boiling sulphuric acid.

Electrolytic Parting of Doré Bullion.

The parting of doré bullion by sulphuric acid is rapidly giving place to the cleaner and less expensive electrolytic separation.

The parting of doré bullion containing 85 to 95 per cent of silver, 1 to 8 per cent gold and 2 to 6 per cent copper, is accomplished by making the metal an anode in an electrolyte containing from 2 to 4 per cent $AgNO_3$, 0.1 per cent to 1.0 per cent free nitric acid, and 1 to 4 per cent of $Cu(NO_3)_2$. The silver is dissolved, leaving the gold (Pt, Pd, Ir, Pb) intermixed with a small amount of finely divided silver as the anode sludge; copper, if present in the anode, dissolves along with the silver; it is not precipitated on the cathode, but accumulates in the electrolyte.

The anodes must be, as far as possible, ridded of lead, bismuth and antimony by cupellation, before the doré is cast into anodes.

The electrolytic parting of doré bullion is carried on either in the Moebius cell or in the Balbach-Thum cell, which differ in their construction. However the electrolytic principles in each are the same. The Moebius cell is the older form of apparatus.

The **Moebius cell** (Fig. 2) is a square wooden cell heavily coated with asphaltum, or an acid-proof earthenware cell (size about 24 in. by 26 in. and 22 in. deep.) In each cell are suspended four sets of four anodes (14 in. by 5½ in. by ⅜ in.) and five rolled sheet cathodes (cathode surface 18 sq. ft.).

Each set of four anodes is surrounded by a muslin bag, the purpose of which is to prevent the anode sludge becoming mixed with the deposited silver. Between each set of anodes and cathodes are two wooden stirrers, which operate back and forth for the double purpose of agitating the electrolyte, and to prevent the anode and cathodes becoming short-circuited by the long crystals of cathode silver. The stirrers knock the crystals from the cathodes, and they drop to the bottom of the cell, collecting in a wooden tray, which has a hinged bottom. The conductor bars of the anodes and the cathodes, the stirrers and the tray are all attached to a wooden frame, which may be raised from and lowered into the cell by means of a small windlass. Once or twice a day the current is shunted, the frame lifted from the cell and the silver scraped from the cathode and caught in the tray beneath. The silver is dumped from the tray, the frame then lowered into the cell and the current turned on. The refined silver is carried to the washing vat. The dried silver crystals are melted in a graphite crucible or in an oil-fired reverberatory furnace and cast into bars, assaying 999 + fine silver.

The gold sludge in the muslin bags, which contains all the gold (Pt, Ir, Pd), most of the bismuth (as $Bi_2O_5xH_2O$), and lead (as PbO_2) of the original doré bullion, and a small amount of the silver and copper, is removed every second or third day. It is refined by acid treatment, or by the Wohlwill gold refining process.*

In the course of the electrolysis of doré bullion, its copper content dissolves along with the silver and accumulates in the electrolyte, replacing an equivalent of silver (here Cu stands above Ag in the e.m.f. series). The copper in the electrolyte may amount to as much as 5 per cent, and the silver reduced to as low as 0.5 per cent without the copper being precipitated along with the silver. When the copper reaches about 4 per cent, part of the electrolyte is withdrawn from the cell, and it is replaced by a fresh lot of $AgNO_3$ solution from the treatment of the anode sludge.

*See papers by E. Wohlwill, this journal, vol. II, pp. 221 and 261; vol. VI, p. 450, and by Tuttle, vol. I, p. 157; also vol. VI, pp. 355 and 408 and vol. VIII, p. 82.

In some cases, when the copper content of the anode is low, the electrolyte which adheres to the cathode silver is ample to keep the copper in the electrolyte sufficiently low to prevent it depositing with the silver. The silver in the withdrawn solution is precipitated on sheets, or plates, of copper, giving cement silver, which is collected and melted with the washed cathode silver. The copper in the precipitated solution is recovered by either precipitating on metallic iron, or else the solution is thrown on roasted ore or matte which is to be smelted.

The Moebius cells are usually connected in multiples of five or six. The current per cell is from 350 to 450 amp. (C. D. = 20 to 25 amp. per sq. ft.) and e.m.f. per cell is 1.4 to 1.7 volt. The current efficiency is from 92 per cent to 95 per cent. The amount of silver permanently tied up as cathodes and in electrolyte is from 40 to 45 per cent of daily capacity; but by operating with higher current density, the silver permanently tied up per daily capacity of the cell is reduced. A current density of 40 to 50 amp. per square foot has been successfully employed.

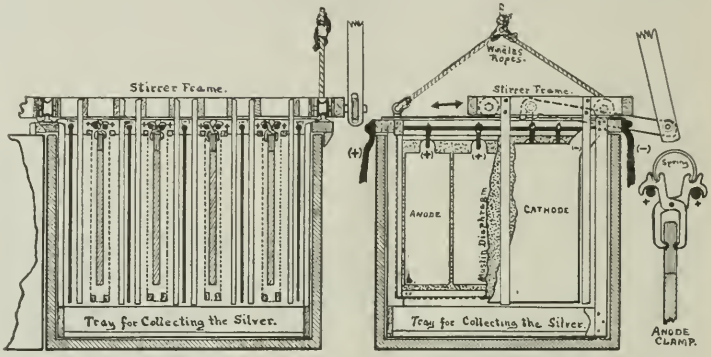


FIG. 2.—SECTIONS OF MOEBIUS CELL.

The power consumption in the Moebius cell is about 13.5 watt-hours per Troy ounce of silver deposited. Cost of refining in this cell is from ¼ to ½ cent per Troy ounce of silver deposited. The parting is rapid, as the anodes are dissolved on all sides; and when operating with high current density, the recovery of the refined silver is as rapid as by the sulphuric-acid parting process.

Dr. D. K. Tuttle (this journal, vol. IV, p. 306) and assistants, at the Philadelphia Mint, developed a method of electrolytically refining silver-gold alloys which gives solid, adherent, crystalline deposits of silver on the cathode, and thus obviates the need of mechanical stirrers, as are used in the Moebius cell. The process is in use at the U. S. Mints in Philadelphia, Denver and San Francisco.

The general arrangement of the cells and the operation of refining are similar to the refining of copper and of lead, but on a smaller scale. The cells are made of acid-proof earthenware, 20 in. wide and 11 in. deep, and hold 42 anodes (6 rows of 7 anodes side by side) and 40 cathodes (5 rows of 8 side by side), suspended vertically with 1.5-in. space between centers of electrodes. The anodes are 7.25 in. long, 2.5 in. wide and ⅜ in. thick. The cathodes are of same width and length as the anodes and are made of rolled sheet silver 0.016 in. thick. The electrodes in each cell are in multiple.

The cells are connected in series, and are operated at a current density of about 7 amp. per square foot of anode surface, and an e.m.f. of from 1 to 1.4 volt, with a current efficiency above 90 per cent.

The gold which is present in the anode in amounts between

15 and 30 per cent, remains attached to the anode as a chocolate colored mud. If the gold is present in smaller amounts, the anodes must be enclosed in muslin bags for the purpose of preventing the anode sludge floating in the electrolyte and becoming attached to the cathode.

The electrolyte contains about 4 per cent $AgNO_3$ and 1.0 per cent HNO_3. It is circulated by means of small propellers at one end of the cell, or by gravity. (The gravity cascade system of circulation gives better circulation than the propellers, and does not keep particles of the sludge in suspension so long a time as the propellers, therefore affording a higher recovery of the gold residue and the production of purer silver.)

By maintaining rapid circulation and by having present in the electrolyte 1 *part by weight of gelatine in* 8000 *to* 10,000 *parts of solution*, the silver deposits on the cathodes as a mass of solid coherent crystals, which seldom short-circuit the cell. The successful operation of the process depends upon the crystals remaining attached to the cathode and to the overcoming of short-circuits by producing smooth deposits. The addition of the gelatine is made each day as needed, which is judged by the appearance of the deposits. It is added as a solution in hot water (10 to 20 grams per litre). Too much gelatine darkens the electrolyte (due to the chemical precipitation of cement silver), and also causes the deposit to form non-coherent.

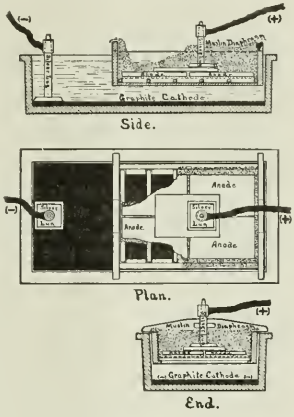


FIG. 3.—BALBACH-THUM CELL.

The cathodes, when the anodes are almost completely dissolved, are removed from the cell, washed, melted and cast into bars of 999 + fine. The gold residue is collected, washed, and treated by either the acid leaching method (when only silver is present), or else is melted, cast into anodes, and refined by the Wohlwill electrolytic process (in case Pt, Pd, and Ir are present).

The scraps of anode are remelted along with silver to be refined.

On account of the low current density which is necessary for the production of smooth, coherent deposits of silver, this process of refining cannot be economically used in the works. But as the alloys which are refined in the Mints contain from 15 to 30 per cent of gold, and as no interest is charged against the metal which is "tied up" in the cells, this method of parting silver and gold can be used in this special case.

The Balbach-Thum cell has no mechanically moving parts. It consists of a long, shallow, narrow cell about 10 in. deep by 26 in. wide by 48 in. long, made of chemical-proof earthenware. They are supported on acid-proof insulated pillars.

The cathode is made up of strips of Acheson graphite, about ¾ in. thick and completely covering the bottom of the cell. Surface, 8 sq. ft. A silver lug is used for electric connection.

The anodes of doré bullion, size 8 in. by 12 in., are suspended horizontally in the cell by being placed in a wooden or earthenware tray (24 in. by 24 to 36 in.), with muslin bottom, which is about 4 in. from the cathode. The muslin is supported on cross-bars which form the bottom of the trays. Electrical contact of the doré slabs is made by means of a slab of silver with a vertical lug.

The silver is deposited as loosely adhering crystals on the graphite slabs, and from time to time are removed with a hardwood shovel. The crystals are washed, melted and cast into bars assaying 999 + fine, and weighing 1000 to 1200 Troy oz.

The gold (and Pt), bismuth (as $Bi_2O_3.xH_2O$), lead (as PbO_2) collect as sludge on the muslin in the tray, and are removed every third day and treated in the same way as anode sludge from the Moebius cells.

The electrolyte is kept in proper condition by removing a portion, when copper reaches about 4 per cent, and replacing with same amount of fresh solution, in the same way as is done in Moebius cell.

The cells are arranged in sets of 4 to 6 in parallel, and operate with a current of about 150 amp. per cell.

The cathode surface is 8 sq. ft., giving a current density of 20 amp. per sq. ft.

The anode surface is about 4 sq. ft., giving a current density of 40 amp. per sq. ft.

The current efficiency averages 88 per cent; at times, 90 per cent.

The e.m.f. per cell ranges from 3.2 to 3.8 volts.

The power consumption in the Balbach-Thum cell is about 31.5 watt-hours per 1 Troy oz. of silver deposited.

The amount of silver permanently tied up in the cell is about 32 per cent of the daily output of the refinery.

Advantages claimed for the Balbach-Thum cell over the Moebius cell:

1. No moving parts, therefore less labor and apparatus to keep the plant operating.
2. Less silver tied up in the plant (32 per cent as against 45 per cent).
3. Less time lost in removing the silver and sludge.
4. Less wear and tear on apparatus, therefore less repair costs.
5. No anode scrap to remelt. (The anodes allowed to completely dissolve in the cell).

Advantages claimed for the Earthenware Tray cells over the Wooden Tray cells:

1. Cleaner and more permanent cell.
2. Deposited silver forms as larger crystals which are easily washed and more readily melted. (The action of the electrolyte on the wood forms organic compounds which, when present in the electrolyte, causes the silver to deposit as fine crystals).
3. Higher current efficiency (93 to 96 per cent) as against 88 to 90 per cent).

Objections claimed for the Balbach-Thum cell as compared with the Moebius cell:

1. Larger power consumption (3.5 volts as compared with 1.5 volt).
2. Large consumption of HNO_3, due to high e.m.f., forming NH_3.
3. Larger space required per given daily capacity, (about 6 times more space).
4. The anodes dissolved only on under side, facing the cathode, therefore less rapid per anodes under treatment.
5. More silver tied up in the electrolyte per given parting capacity, (larger volume of electrolyte per weight of anodes).

Advantages of electrolytic parting over acid parting of doré Bullion:

1. Production of purer silver.
2. Complete separation of selenium and tellurium (these render silver brittle).
3. Less cost of operation, due to smaller consumption of chemicals.
4. Less loss of metal, due to less handling of materials.
5. $CuSO_4.5HO_2O$ not produced as by-product.

6. The purified silver recovered in shorter time, if high current density is used.

Disadvantages of electrolytic parting, as compared with acid parting of doré bullion:

1. First cost of plant (electric generators, conductor cells, etc).

2. Time the gold residue held for parting (interest on same). It requires from 48 to 60 hours to have the gold in bars, whereas about 30 hours required when the parting is done with sulphuric acid.

Treatment of Silver Anode Sludge.

There are two methods of treating the silver anode sludge:

A.—Chemical Method. When there is not enough Pt. Pd and Ir present to pay for recovery: The sludge consists of all the gold (Pt, Pd, Ir), most of the bismuth (as $Bi_2O_3xH_2O$) and lead (as PbO_2) contained in the doré bullion, and a small amount of silver and copper. It is treated in porcelain, or acid-proof, earthen-ware bowls, which are heated by steam jackets, which conform to the bottom of the bowl, or else are heated by being placed in a sand-bath with flame underneath. The bowls are covered with an acid-proof hood which is in connection with a good draft, for the purpose of carrying off the acid fumes.

The sludge is first boiled with dilute HNO_3 (2 acid: 3 water) till no further reaction occurs; the solution is decanted, the residue of gold washed, and retreated with boiling concentrator H_2SO_4. When acid treatment is complete, the solution is decanted and the residue thoroughly washed with hot water, dried, and melted under borax glass in a graphite crucible. It is then cast into bars assaying 998 + fine.

The HNO_3 solution, containing silver nitrate and free nitric acid, is diluted and added to the silver electrolyte in the parting cells in place of the solution which is removed from time to time in order to keep the copper content down.

The H_2SO_4 solution is diluted, and the silver precipitated by boiling with sheets of metallic copper, giving cement-silver, which is melted with the cathode silver and cast into bars. The resulting $CuSO_4$ solution is either crystallized, or else sent to the copper refinery and added to the copper electrolytic tanks.

B.—Electrolytic Method. When platinum, palladium, and iridium are in sufficient amounts to pay for their recovery: The sludge is washed free of electrolyte, dried, melted under borax glass and soda ash flux, and cast into anodes, which are refined by the Wohlwill electrolytic process.

Protection of Wood Work of the Parting Plant.

All exposed woodwork of the parting plant should be well protected with several coats of an acid-proof paint (asphaltum paint, or patented P. & B. paint) in order to prevent the possibility of fire by spontaneous combustion, due to the action of nitric acid fumes on the wood. Ample ventilation for the removal of the fumes should be provided also, in order to protect the health of the workmen.

Electrolytic Refining of Gold.

The Wohlwill Electrolytic Gold Refining Process is used at the U. S. mints in Philadelphia, Denver and San Francisco for purifying gold, and for the recovery of the valuable impurities (such as Pt., Pd., Ir., and Os).

The anodes are gold bullion, of reasonable purity, running over 940 fine, and the cathodes are thin sheets of hard rolled gold.

The electrolyte contains from 3 to 4 per cent $AuCl_3$, 2 to 5 per cent free HCl (sp. gr. 1.2) and 1 to 3 per cent NaCl. If the anodes contain lead, then from 1 to 2 per cent H_2SO_4 is added to prevent the lead dissolving and being deposited with the gold; the H_2SO_4 causes the lead to enter the anode sludge as $PbSO_4$.

The conditions necessary to obtain high current efficiency and to deposit the gold in solid form are the presence of free HCl in the electrolyte, circulation of electrolyte, and temperature between 50 and 70 C. With neutral $AuCl_3$ electrolyte and at ordinary temperature, the anode is slowly dissolved, free chlorine being liberated at the anode, and hydrogen at the cathode.

The circulation of the electrolyte is accomplished by means of small propellers in one end of the cell, or else by the gravity cascade system. The former method of circulation is not so satisfactory as the latter, as it keeps the anode sludge longer in suspension, which permits particles becoming attached to the cathode. In case of the gravity-cascade system of circulation, the cells are arranged in rows of 12 or more, with ample height between adjoining cells to allow the solution flowing from the upper cell to the lower one. Porcelain pumps are used for raising the electrolyte from the sump cell to the upper cell. The circulation should be ample to maintain uniform composition of the electrolyte in all parts of the cell, which may be judged by the corrosion of the anode and the appearance of the cathode.

The cells are acid-proof earthenware, or porcelain, about 15 in. by 12 in. and 10 in. deep (or larger, depending on size and number of anodes).

A sufficient current density is used so as to allow the refining being done as rapidly as by the acid refining process, which is in 24 hr. The usual current density is between 80 and 120 amp. per square foot. The e.m.f. per cell is between 1 and 1.3 volt, with electrodes spaced about ¾ in. centers. The energy consumed is about 10 watt-hours per 1 troy ounce of cathode gold.

The heating of the cells is accomplished by having them placed on a bed of sand, which has steam coils imbedded. At a temperature of 50 to 70° C. much evaporation occurs, so water must be added from time to time to keep the volume of electrolyte constant.

The anodes of impure gold (940 + fine) are cast in iron molds, size 6 in. long by 3 in. wide and about ⅜-in. thick (24 hour anodes). The cathodes are of same length and width as the anodes, and are made of pure rolled gold sheets.

The purity of the deposited gold is 999.8 fine. It is melted in graphite crucibles and cast into bars.

The sludge contains AgCl, $PbSO_4$, Ir. and Os. If the anodes run less than 940 fine (6 per cent Ag + Pb) they become coated with the AgCl and $PbSO_4$ which are non-conductors. In this case, it is necessary to scrape the anodes when voltage of cell becomes high and current becomes low.

The silver is recovered from the sludge by fusing it with Na_2CO_3, which reduces the AgCl to metal:

$$2AgCl + xNa_2CO_3 = 2Ag + 2NaCl + CO_2 - O + (x-1)\ Na_2CO_3$$

If iridium and osmium are present in the anode, they collect with the AgCl, and are separated from the silver by the acid parting method.

If copper, platinum and paladium are constituents of the anode, they are dissolved along with the gold, replacing an equivalent of gold in the electrolyte (higher in e.m.f. series than gold), but are not deposited unless allowed to accumulate to high percentage. (Pd must not exceed 5 per cent.)

Each day, gold chloride solution must be added to the cells in order to supply the equivalent of gold which was replaced during electrolysis by Cu, Pt, and Pd. This solution is prepared either chemically or electrolytically. The chemical method of preparation is by dissolving gold in aqua-regia and evaporating off the chlorine, which is an expensive method and is very disagreeable on account of chlorine. The electrolytic method is by electrolyzing concentrated hydrochloric acid between gold anodes, and gold cathodes (in a porous cup), the anode current density 100 amp. per square foot, and the cathode current density 250+ amp. per square foot. Temperature of solution 60° to 70° C. The electrolyte around the anode becomes more and more concentrated in $AuCl_3$. The high current density of the current at the cathode prevents gold being deposited, hydrogen being liberated instead. When the solution in the anode compartment becomes suffi-

ciently concentrated in gold chloride, it is removed and replaced by the solution in the porous cup surrounding the cathode, which in turn is replaced by fresh concentrated hydrochloric acid.

The platinum (and paladium) is recovered from the electrolyte by precipitating as insoluble double ammonium chloride salts $(NH_4Cl)_2PtCl_4$. The gold must first be precipitated by a current of SO_2, the solution is filtered, then NH_4Cl added and the solution evaporated to small volume. The precipitated $(NH_4Cl)_2PtCl_4$ on ignition is reduced to metal, NH_4Cl and Cl being volatilized.

Reactions:

$$2AuCl_3 + 3SO_2 + 6H_2O = 2Au + 6HCl + 3H_2SO_4.$$
$$PtCl_4 + 2NH_4Cl = (NH_4Cl)_2\,PtCl_4.$$

If palladium is present it is precipitated along with the platinum as the double salt. It is separated from the platinum by digesting the ignited precipitate with hot conc. HNO_3, which dissolves the Pd, leaving the Pt as fine particles.

Use of a Pulsating Current for Refining Gold.

It has been found (this journal, Vol. VIII, p. 82) that when a *pulsating direct current* is used for electrolysis of gold anodes, which contain over 6 per cent of silver, the AgCl does not accumulate on the anode and stop the current. Even as much as 20 per cent silver in the anode does not prevent the electrolysis, when using pulsating direct current.

The *pulsating direct current* is accomplished by connecting a single-phase alternator in series with the direct-current generator, the voltage of the alternator being lower than the voltage of the generator, the effect being a pulsating current. The alternating current does not effect the deposition of the gold, but causes the AgCl to break from the anode, thereby avoiding mechanical scrapers for this purpose. The electrolytic effect of the alternating and direct current combination is equal to that of the direct current alone.

The effect of the alternating current on the electrolysis is: (a) it reduces the actual e. m. f. of the cell; (b) it prevents the liberation of Cl at the anode; (c) it breaks the AgCl from the anode; (d) it allows the use of a higher current density; (e) less gold is left in the anode sludge; (f) the electrolysis may be conducted at a lower temperature—25 to 35° C.

The direct current density from the electrodes is 110+ amperes per square foot.

The power consumption is the sum of both the alternating and the direct current.

In the design of the generators, figure on 1.3 volts as the maximum for the direct current generator per each cell in series, and 1.0 volt for that of the alternator per each cell in series. Since the generators are connected in series, their armatures must be so dimensioned as to withstand the heating effect of the maximum total combined alternating plus direct current. The frequency of the alternations has no practical effect on the electrolysis, the usual frequency of 60 per second is very satisfactory.

Electrolytic refining of gold is advantageous in that:

1. It recovers the Pt, Pd, Ir, and Os.
2. It gives the purest kind of gold.
3. It is cheaper than the chemical method of refining.
4. No disagreeable chlorine gas is produced.

Floor of Parting Plants.

The floor of the silver parting plant is laid with an acid-proof and solution-tight material, usually rough concrete bottom with an asphaltum topping. The floor is made with a general slope to a common sump.

At the United States Mints the floors in the gold refinery are made of concrete bottom covered with acid-proof slate slabs two inches thick, with their joints cemented with a mixture of litharge and glycerine. The sweepings from the floor and the residues from the floor washings are melted with the impure gold and cast into anodes and refined.

Cost of Power; Conversion Factors.

By Carl Hering.

Owing to the fact that the cost of power is sometimes stated in cents per hour and at other times in dollars per year, and that the year is sometimes taken at the rate of 24 hours per day and at other times 10, and that this is further complicated by the power being sometimes stated in horse-powers (and in some foreign countries in metric horse-powers) and at other times in kilowatts, the conversions from any one to another sometimes become tedious and are often liable to errors owing to the difference between a unit and a quantity measured in terms of that unit.

The following table of constants has therefore been determined accurately, giving the values of each in terms of each of the others, so that a single multiplication is all that is required for reducing a cost given in terms of one of them into any of the others, as the reciprocals of all values are always given in their proper places. The constants are arranged in the same style and order as those in the writer's "Conversion Tables" for a new edition of which they have been determined. They are in the order of size, beginning with the smallest. The approximate values, which are sufficiently accurate for many calculations, have been selected so as to minimize calculations.

Two kinds of years have been included, one the complete or full year, of 365¼ days of 24 hours each, and the other a working year of 300 days of 10 hours each, and they have been indicated by the names "full" and "working" respectively.

A horse-power-hour or year is really a measure of a quantity of energy and not of power; but as such quantities are popularly referred to as "power" this term has been used in the title, although not strictly correct.

All the values have been carefully calculated by two independent methods besides being cross checked, and it is believed there are no errors.

Cost of Energy; Charges for Power; Cost of Power During Specified Times.

(Money ÷ product of power and time: money ÷ energy.)

NOTE: A "full" year means 24 hours per day and 365¼ days per year = 8766 hours.
A "working" year means 10 hours per day and 300 days per year = 3000 hours.
Aprx. means within 2%.

1 dollar per kilowatt year (full) [$/kw-yr.]:

= 0.745 650	dol. per hp-year (full).	Aprx. ¾.
= 0.342 231	dol. per kw-year (working).	Aprx. 34 ÷ 100.
= 0.255 185	dol. per hp-year (working).	Aprx. 23/9 ÷ 10.
= 0.011 407 7	ct. per kw-hour.	Aprx. 8/7 ÷ 100.
= 0.008 506 16	ct. per hp-hour.	Aprx. 6/7 ÷ 100.

1 dollar per horse-power-year (full) [$/hp-yr.]:

= 1.341 11	dol. per kw-year (full).	Aprx. add 1/3.
= 0.458 971	dol. per kw-year (working).	Aprx. 5/11.
= 0.342 231	dol. per hp-year (working).	Aprx. 34 ÷ 100.
= 0.015 299 0	cts. per kw-hour.	Aprx. 3 2 ÷ 100.
= 0.011 407 7	cts. per hp-hour.	Aprx. 8 7 ÷ 100.

1 dollar per kilowatt year (working) [$ kw-yr.]:

= 2.922 00	dol. per kw-year (full).	Aprx. 29 ÷ 10.
= 2.178 79	dol. per hp-year (full).	Aprx. 11/5.
= 0.745 650	dol. per hp-year (working).	Aprx. 3/4.
= 0.033 333 3	ct. per kw-hour.	Aprx. 1 3 ÷ 10.
= 0.024 855 0	ct. per hp-hour.	Aprx. 1/4 ÷ 10.

1 dollar per horse-power-year (working) [$/hp-yr.]:

= 3.918 73	dol. per kw-year (full).	Aprx. 35/9.
= 2.922 00	dol. per hp-year (full).	Aprx. 29 ÷ 10.
= 1.341 11	dol. per kw-year (working).	Aprx. add 1/3.
= 0.044 703 8	ct. per kw-hour.	Aprx. 9 2 ÷ 100.
= 0.033 333 3	ct. per hp-hour.	Aprx. 1 3 ÷ 10.

1 cent per kilowatt hour [cts./kw-hr]:

= 87.660 0	dol. per kw-year (full).	Aprx. 7/8 × 100.
= 65.363 7	dol. per hp-year (full).	Aprx. 65.
= 30.	dol. per kw-year (working).	
= 22.369 5	dol. per hp-year (working).	Aprx. 9/4 × 10.
= 0.745 650	ct. per hp-hour.	Aprx. 3/4.

1 cent per horse-power-hour [cts./hp-hr]:

= 117.562	dol. per kw-year (full).	Aprx. 7 6 × 100.
= 87.660 0	dol. per hp-year (full).	Aprx. 7 8×100.
= 40.233 4	dol. per kw-year (working).	Aprx. 40.
= 30.	dol. per hp-year (working).	
= 1.341 11	cts. per kw-hour.	Aprx. add 1 3.

Foreign.

The values of foreign moneys here used are those specified in the writer's "Conversion Tables" and are those used by the U. S. Treasury Dept. They are the true relative values based on gold, as determined by the U. S. Mint in terms of the U. S. gold dollar. The three fundamental values are: 1 pound sterling = 240 pence = $4.8665; 1 mark = 100 pfennige = $0.238; 1 franc = 100 centimes = $0.193. Those constants which depend upon these money values are given to only three or four significant figures; those independent of them are given to six figures, as usual. It has been assumed that the regular horse-power is the only one used in England (as in the United States) and that the metric horse-power is the only one used in the rest of Europe; when this is not the case two calculations will be necessary. One metric horse-power = 0.986-318 horse-power; one horse-power = 1.013 87 metric horse-powers: the difference is therefore about 1½%.

1 franc per kilowatt-year (full):
= 0.811 mark per kilowatt-year (full). Aprx. 4/5.
= 0.735 448 franc per metric horse-power-year (full). Aprx. 22/3 ÷ 10.
= 0.342 231 franc per kilowatt-year (working). Aprx. 34 ÷ 100.
= 0.251 693 franc per metric horse-power-year (working). Aprx. 1/4.
= 0.193 dollar per kilowatt-year (full). Aprx. 19 ÷ 100.
= 0.039 7 pound per kilowatt-year (full). Aprx. 4 ÷ 100.
= 0.011 407 7 centime per kilowatt-hour. Aprx. 8/7 ÷ 100.
= 0.008 389 77 centime per metric horse-power-hour. Aprx. 5/6 ÷ 100.

1 mark per kilowatt-year (full):
= 1.23 francs per kilowatt-year (full). Aprx. 1 8 × 10.
= 0.735 448 mark per metric horse-power-year (full). Aprx. 22/3 ÷ 10.
= 0.342 231 mark per kilowatt-year (working). Aprx. 34 ÷ 100.
= 0.251 693 mark per metric horse-power-year (working). Aprx. 1/4.
= 0.238 dollar per kilowatt-year (full). Aprx. 12/5 ÷ 10.
= 0.048 9 pound per kilowatt-year (full). Aprx. 49 ÷ 1000.
= 0.011 407 7 pfennig per kilowatt-hour. Aprx. 8/7 ÷ 100.
= 0.008 389 78 pfennig per metric horse-power-hour. Aprx. 5/6 ÷ 100.

1 franc per metric horse-power-year (full):
= 1.359 72 francs per kilowatt-year (full). Aprx. add 1/3.
= 0.811 mark per metric horse-power-year (full). Aprx. 4/5.
= 0.465 337 franc per kilowatt-year (working). Aprx. 7/15.
= 0.342 231 franc per metric horse-power-year (working). Aprx. 34 ÷ 100.
= 0.196 dollar per horse-power-year (full). Aprx. 8/41.
= 0.040 2 pound per horse-power-year (full). Aprx. 4 ÷ 100.
= 0.011 511 2 centime per kilowatt-hour. Aprx. 31/2 ÷ 1000.
= 0.011 407 7 centime per metric horse-power-hour. Aprx. 8/7 ÷ 100.

1 mark per metric horse-power-year (full):
= 1.359 72 marks per kilowatt-year (full). Aprx. add 1/3.
= 1.23 francs per metric horse-power-year (full). Aprx. 1/8×10.
= 0.465 337 mark per kilowatt-year (working). Aprx. 7/15.
= 0.342 231 mark per metric horse-power-year (working). Aprx. 34 ÷ 100.
= 0.241 dollar per horse-power-year (working.) Aprx. 12/5 ÷ 10.
= 0.049 6 pound per horse-power-year (full). Aprx. 5 ÷ 100.
= 0.015 511 2 pfennig per kilowatt-hour. Aprx. 31/2 ÷ 1000.
= 0.011 407 7 pfennig per metric horse-power-hour. Aprx. 8/7 ÷ 100.

1 franc per kilowatt-year (working):
= 2.922 00 francs per kilowatt-year (full). Aprx. 29 ÷ 10.
= 2.148 98 francs per metric horse-power-year (full). Aprx. 43/2 ÷ 10.
= 0.811 mark per kilowatt-year (working). Aprx. 4/5.
= 0.735 448 franc per metric horse-power-year (working). Aprx. 3/4.
= 0.193 dollar per kilowatt-year (working). Aprx. 19 ÷ 100.
= 0.039 7 pound per kilowatt-year (working). Aprx. 4 ÷ 100.
= 0.033 333 3 centime per kilowatt-hour. Aprx. 1/3 ÷ 10.
= 0.024 514 9 centime per metric horse-power-hour. Aprx. 1/4 ÷ 10.

1 mark per kilowatt-year (working):
= 2.922 00 marks per kilowatt-year (full). Aprx. 29 ÷ 10.
= 2.148 98 marks per metric horse-power-year (full). Aprx. 43/2 ÷ 10.
= 1.23 francs per kilowatt-year (working). Aprx. 1/8×10.
= 0.735 448 mark per metric horse-power-year (working). Aprx. 3/4.
= 0.238 dollar per kilowatt-year (working). Aprx. 12/5 ÷ 10.
= 0.048 9 pound per kilowatt-year (working). Aprx. 49 ÷ 1000.
= 0.033 333 3 pfennig per kilowatt-hour. Aprx. 1/3 ÷ 10.
= 0.024 514 9 pfennig per metric horse-power-hour. Aprx. 1/4 ÷ 10.

1 franc per metric horse-power-year (working):
= 3.973 09 francs per kilowatt-year (full). Aprx. 4.
= 2.922 00 francs per metric horse-power-year (full). Aprx. 29 ÷ 10.
= 1.359 72 francs per kilowatt-year (working). Aprx. add 1/3.
= 0.811 mark per metric horse-power-year (working). Aprx. 4/5.
= 0.196 dollar per horse-power-year (working). Aprx. 8/41.
= 0.045 323 9 centime per kilowatt-hour. Aprx. 9/2 ÷ 100.
= 0.040 2 pound per horse-power-year (working). Aprx. 4 ÷ 100.
= 0.033 333 3 centime per metric horse-power-hour. Aprx. 1/3 ÷ 10.

1 mark per metric horse-power-year (working):
= 3.973 09 marks per kilowatt-year (full). Aprx. 4.
= 2.922 00 marks per metric horse-power-year (full). Aprx. 29 ÷ 10.
= 1.359 72 marks per kilowatt-year (working). Aprx. add 1/3.
= 1.23 francs per metric horse-power-year (working). Aprx. 1/8 × 10.
= 0.241 dollar per horse-power-year (working). Aprx. 12/5 ÷ 10.
= 0.049 6 pound per horse-power-year (working). Aprx. 5 ÷ 100.
= 0.045 323 9 pfennig per kilowatt-hour. Aprx. 9/2 ÷ 100.
= 0.033 333 3 pfennig per metric horse-power-hour. Aprx. 1/3 ÷ 10.

1 dollar per kilowatt-year (full):
= 5.18 francs per kilowatt-year (full). Aprx. 26/5.
= 4.20 marks per kilowatt-year (full). Aprx. 21/5.
= 0.205 pound per kilowatt-year (full). Aprx. 41/2 ÷ 100.

1 dollar per horse-power-year (full).
= 5.11 francs per metric horse-power-year (full). Aprx. 51 ÷ 10.
= 4.15 marks per metric horse-power-year (full). Aprx. 25/6.
= 0.205 pound per horse-power-year (full). Aprx. 41/2 ÷ 100.

1 dollar per kilowatt-year (working):
= 5.18 francs per kilowatt-year (working). Aprx. 26/5.
= 4.20 marks per kilowatt-year (working). Aprx. 21/5.
= 0.205 pound per kilowatt-year (working). Aprx. 41/2 ÷ 100.

1 dollar per horse-power-year (working):
= 5.11 francs per metric horse-power-year (working). Aprx. 51 ÷ 10.
= 4.15 marks per metric horse-power-year (working). Aprx. 25/6.
= 0.205 pound per horse-power-year (working). Aprx. 41/2 ÷ 100.

1 pound per kilowatt-year (full):
= 25.2 francs per kilowatt-year (full). Aprx. 1/4 × 100.
= 20.4 marks per kilowatt-year (full). Aprx. 41/2.
= 4.87 dollars per kilowatt-year (full). Aprx. 2/41 × 100.
= 0.745 650 pound per horse-power-year (full). Aprx. 3/4.
= 0.342 231 pound per kilowatt-year (working). Aprx. 34 ÷ 100.
= 0.255 185 pound per horse-power-year (working). Aprx. 23/9 ÷ 10.
= 0.027 378 5 penny per kilowatt-hour. Aprx. 3/11 ÷ 10.
= 0.020 414 8 penny per horse-power-hour. Aprx. 1/50.

1 pound per horse-power-year (full):
= 24.9 francs per metric horse-power-year (full). Aprx. 1/4 × 100.
= 20.2 marks per metric horse-power-year (full). Aprx. 1/5 × 100.
= 4.87 dollars per horse-power-year (full). Aprx. 2/41 × 100.
= 1.341 11 pounds per kilowatt-year (full). Aprx. add 1/3.
= 0.458 971 pound per kilowatt-year (working). Aprx. 5/11.
= 0.342 231 pound per horse-power-year (working). Aprx. 34 ÷ 100.
= 0.036 717 7 penny per kilowatt-hour. Aprx. 11/3 ÷ 100.
= 0.027 378 5 penny per horse-power-hour. Aprx. 3/11 ÷ 10.

1 pound per kilowatt-year (working):
= 25.2 francs per kilowatt-year (working). Aprx. 1/4 × 100.
= 20.4 marks per kilowatt-year (working). Aprx. 41/2.
= 4.87 dollars per kilowatt-year (working). Aprx. 2/41 × 100.
= 2.922 00 pounds per kilowatt-year (full). Aprx. 29 ÷ 10.
= 2.178 79 pounds per horse-power-year (full). Aprx. 11/5.
= 0.745 650 pound per horse-power-year (working). Aprx. 3/4.
= 0.080 00 penny per kilowatt-hour. Aprx. 8/100.
= 0.059 652 0 penny per horse-power-hour. Aprx. 6/100.

1 centime per kilowatt-hour:
= 87.660 0 francs per kilowatt-year (full). Aprx. 7/8×100.
= 64.469 0 francs per metric horse-power-year (full). Aprx. 65.
= 30. francs per kilowatt-year (working).
= 22.063 4 francs per metric horse-power-year (working). Aprx. 22.
= 0.811 pfennig per kilowatt-hour. Aprx. 4/5.
= 0.735 448 centime per metric horse-power-hour. Aprx. 3/4.
= 0.596 pfennig per metric horse-power-hour. Aprx. 6/10.
= 0.193 cent per kilowatt-hour. Aprx. 19 ÷ 100.
= 0.144 cent per horse-power-hour. Aprx. 1/7.
= 0.095 2 penny per kilowatt-hour. Aprx. 2/21.
= 0.070 9 penny per horse-power-hour. Aprx. 7/100.

1 pound per horse-power-year (working):
= 24.9 francs per metric horse-power-year (working). Aprx. 1/4 × 100.
= 20.2 marks per metric horse-power-year (working). Aprx. 1/5 × 100.
= 4.87 dollars per horse-power-year (working). Aprx. 2/41 × 100.
= 3.918 73 pounds per kilowatt-year (full). Aprx. 35/9.
= 2.922 00 pounds per horse-power-year (full). Aprx. 29 ÷ 10.
= 1.341 11 pounds per kilowatt-year (working). Aprx. add 1/3.
= 0.107 289 penny per kilowatt-hour. Aprx. 3/28.
= 0.080 0 penny per horse-power-hour. Aprx. 8/100.

1 pfennig per kilowatt-hour:
= 87.660 0 marks per kilowatt-year (full). Aprx. 7/8 × 100.
= 64.469 0 marks per metric horse-power-year (full). Aprx. 65.
= 30. marks per kilowatt-year (working).
= 22.063 4 marks per metric hp-year (working). Aprx. 22.
= 1.23 centimes per kilowatt-hour. Aprx. 1 8 × 10.
= 0.907 centime per metric horse-power-hour. Aprx. 9/10.
= 0.735 448 pfennig per metric horse-power-hour. Aprx. 3/4.
= 0.238 cent per kilowatt-hour. Aprx. 12/5 ÷ 10.
= 0.178 cent per horse-power-hour. Aprx. 7/4 ÷ 10.
= 0.117 penny per kilowatt-hour. Aprx. 7/6 ÷ 10.
= 0.087 5 penny per horse-power-hour. Aprx. 7/8 ÷ 10.

1 centime per metric horse-power-hour:
= 119.193 francs per kilowatt-year (full). Aprx. 120.
= 87.660 0 francs per metric hp.-yr. (full). Aprx. 7/8 × 100.
= 40.791 5 francs per kilowatt-year (working). Aprx. 40.
= 30. francs per metric horse-power-year (working).
= 1.359 72 centimes per kilowatt-hour. Aprx. add 1/3.
= 1.10 pfennig per kilowatt-hour. Aprx. 11 ÷ 10.
= 9.811 pfennig per metric horse-power-hour. Aprx. 8/10.
= 0.263 cent per kilowatt-hour. Aprx. 5/19.
= 0.196 cent per horse-power-hour. Aprx. 8/41.
= 0.129 penny per kilowatt-hour. Aprx. 13 ÷ 100.
= 0.096 4 penny per horse-power-hour. Aprx. 2/21.

1 pfennig per metric horse-power-hour:
= 119.193 marks per kilowatt-year (full). Aprx. 120.
= 87.660 0 marks per metric hp-yr. (full). Aprx. 7/8 × 100.
= 40.791 5 marks per kilowatt-year (working). Aprx. 40.
= 30. marks per metric horse-power-year (working).
= 1.68 centimes per kilowatt-hour. Aprx. 1/6 × 10.
= 1.359 72 pfennige per kilowatt-hour. Aprx. add 1/3.
= 1.23 centimes per metric hp.-hr. Aprx. 1/8 × 10.
= 0.324 cent per kilowatt-hour. Aprx. 32/100.
= 0.241 cent per horse-power-hour. Aprx. 24/100.
= 0.160 penny per kilowatt-hour. Aprx. 16/100.
= 0.119 penny per horse-power-hour. Aprx. 12/100.

1 cent per kilowatt-hour:
= 5.18 centimes per kilowatt-hour. Aprx. 26/5.
= 4.20 pfennige per kilowatt-hour. Aprx. 21/5.
= 3.81 centimes per metric horse-power-hour. Aprx. 19/5.
= 3.09 pfennige per metric horse-power-hour. Aprx. 31 ÷ 10.
= 0.493 penny per horse-power-hour. Aprx. 1/2.
= 0.368 penny per horse-power-hour. Aprx. 4/11.

1 cent per horse-power-hour:
= 6.95 centimes per kilowatt-hour. Aprx. 7.
= 5.64 pfennige per kilowatt-hour. Aprx. 4/7 × 10.
= 5.12 centimes per metric horse-power-hour. Aprx. 51 ÷ 10
= 4.15 pfennige per metric hp.-hr. Aprx. 25/6.
= 0.662 penny per kilowatt-hour. Aprx. 2/3.
= 0.493 penny per horse-power-hour. Aprx. 1/2.

1 penny per kilowatt-hour:
= 36.525 0 pounds per kilowatt-year (full). Aprx. 36.
= 27.234 9 pounds per horse-power-year (full). Aprx. 27.
= 12.500 0 pounds per kilowatt-year (working). Aprx. 1/8×100.
= 10.5 centimes per kilowatt-hour. Aprx. 21/2.
= 9.320 62 pounds per horse-power-year (working). Aprx. 65/7.
= 8.52 pfennige per kilowatt-hour. Aprx. 17/2.
= 7.73 centimes per metric horse-power-hour. Aprx. 31/4.
= 6.26 pfennige per metric horse-power-hour. Aprx. 25/4.
= 2.03 cents per kilowatt-hour. Aprx. 2.
= 1.51 cents per horsepower-hour. Aprx. 1-1/2.
= 0.745 650 penny per horse-power-hour. Aprx. 3/4.

1 penny per horse-power-hour:
= 48.984 2 pounds per kilowatt-year (full). Aprx. 49.
= 36.525 0 pounds per horse-power-year (full). Aprx. 36.
= 16.763 9 pounds per kilowatt-year (working). Aprx. 1/6 × 100.
= 14.1 centimes per kilowatt-hour. Aprx. 1/7 × 100.
= 12.500 0 pounds per horse-power-year (working). Aprx. 1/8 × 100.
= 11.4 pfennige per kilowatt-hour. Aprx. 8/7 × 100.
= 10.4 centimes per metric horse-power-hour. Aprx. 21/2.
= 8.41 pfennige per metric horse-power-hour. Aprx. 1/12 × 100.
= 2.72 cents per kilowatt-hour. Aprx. 11 4.
= 2.03 cents per horse-power-hour. Aprx. 2.
= 1.341 11 pence per kilowatt-hour. Aprx. add 1/3.

Recent Progress in Calorimetry.—III.

By W. P. White.

Temperature Measurement and Corrections.

Two previous papers of this series have dealt with the progress that has recently been made in eliminating various calorimetric errors due to uncertain or lagging temperatures in parts of the calorimeter, and, more especially, in its surroundings. To avoid these errors is not difficult; when this has been done the principal errors are in the temperature measurement, and are mainly accidental.

Of *mercury thermometers* for calorimetry the most sensitive are read to 0.001°, but many of the best authorities regard 0.003° to 0.002° as the maximum accuracy with which a temperature interval can in practice be reproduced by one of these thermometers. If somewhat different intervals are employed at different times unavoidable calibration errors will make the probable error appreciably greater. These sensitive thermometers seldom have a range of over 6°. Hence the maximum accuracy which could be expected with them is about 0.4 per mille. This, however, supposes that the full range of 6° is employed. Considerably shorter intervals are often recommended on the ground that Newton's law of cooling cannot be assumed to hold for intervals much over 2°. This recommendation is a mistake. The deviation from Newton's law for a 4-degree interval will not equal the thermometric error (unless the period is considerably over 10 minutes), while, on the other hand, a decrease in the interval will often increase the thermometric error almost proportionally. In fact, the effect of the deviation from Newton's law is usually quite negligible for an 8° interval, or can easily be made so by a rough correction.

The advantage of an increase above 6° is more doubtful with mercury thermometers, since several of the purely thermometric errors would tend to increase with the interval. This remark applies to the substitution of longer range thermometers for the most sensitive ones now in use. *With a given thermometer* it is probably always best, as far as the temperature measurement is concerned, to use the full range. And this has been done, up to 20°, without introducing any error chargeable to the calorimeter or the cooling.

For the best calorimetric work yet reported with mercury thermometers the mean accidental error (that is, the average variation of each determination from the mean result of the series to which it belongs) is about 0.5 per mille.

For electric calorimetric thermometers the sensitiveness and the accuracy can readily be made each 10 times the corresponding value for the mercury thermometer. That is, these electric thermometers can be read to 0.0001°, will reproduce ordinary calorimetric intervals with fair certainty to 0.0002°, and will in general yield measurements agreeing within from one ten-thousandth to one fifty-thousandth (0.1 to 0.02 per mille), their performance in this regard depending mainly on the accuracy of the auxiliary resistances. Two observers, working independently with different types of electric thermometer, have each succeeded in getting calorimetric results whose mean accidental error is one thirty-thousandth (0.03 per mille) or less. The *absolute* accuracy of these determinations may not be greater than 0.2 per mille, but this is sufficient for most present requirements and is as great as the accuracy yet attained in determining the variation in the specific heat of water itself. The electric installation is not only more accurate than a mercury thermometer, but is in some ways more convenient. The observer sitting, notebook in hand, before his galvanometer scale, within easy reach of switches which give complete control of the electrical system, can make readings with great rapidity and in great variety. His facilities in these directions will generally be of more use in the complex and varied problems of the research laboratory than in the simpler and more nearly standardized processes of commercial work. The disadvantages of the electric thermometer are three: The cost, the special electrical knowledge required, and the increased chance of occasional loss of time through slight accidents or blunders in the operation of the more complicated system. None of these disadvantages is really serious, unless the cost is. The strongest reason for not using electrical methods will probably be, in most cases, that greater accuracy than that given by the mercury thermometer is either not needed or is prevented by some cause quite outside the calorimeter, such as impurity or variability of materials.

Two forms of electric thermometer, the resistance thermometer and the thermoelement, have been used in calorimetry and have given results of (so far) about equal and very high accuracy. For some types of recording and regulating apparatus the resistance thermometer is much more sensitive than the thermoelement, but in existing calorimetric thermometers there is little difference in this direction.

The resistance thermometer has at present the advantage that a form of it, developed and now used at the Bureau of Standards, can be purchased ready made, with suitable accessory apparatus. A form accurate at least to 0.0003° in calorimetric work costs, complete with accessories, about $225. For the thermoelement also highly satisfactory accessory apparatus (potentiometers, etc.) can now be purchased, and thermoelements suitable for calorimetry require for their construction neither skill, experience, nor great labor, but they are not regularly manufactured at present. Most standardizing laboratories are better equipped for the calibration of resistance thermometers than of thermoelements, though the calibration of thermoelements is very easy wherever suitable standards are available. The chief advantage of the thermoelement appears to be the greater ease with which the thermoelectric installation lends itself to the simultaneous measurement of several temperatures, and of currents and voltages as well, if desired, but this advantage, again, is greatest in the case of the research laboratory.

The Calculation of Corrections for Cooling seems to deserve attention here because (1) it is mainly through the abbreviation of these calculations that shortening and simplification of the labor of calorimetric determinations can be secured, and (2) the customary formulas for these calculations have been misunderstood to a surprising extent, considering their age and simplicity. It will probably be sufficient, however, to give merely results and conclusions. A derivation of the two formulas here given will be found in the articles on calorimetry already referred to, those in the November and December *Physical Review* of last year. One of the formulas, also, the Pfaundler, has been often presented, though with a bewildering variety of notations, which I have made some effort to simplify.

Three kinds of data may in different cases be needed to get cooling corrections: (1) Data as to the temperature changes produced in the calorimeter by stirring and evaporation; (2) the temperatures of the calorimeter during three periods, namely, the *transfer* period, during which there is communicated to the calorimeter the heat which is to be measured, and two *cooling* periods, before and after the transfer period; (3) the rate of [temperature change of] the calorimeter during the cooling periods.

It is generally known, but sometimes overlooked, that these data may properly be much less accurate than the measurements which determine the main temperature rise, since the cooling correction is only about one-fiftieth of the whole interval. The observer who corrects his transfer period temperatures to thousandths of a degree is simply wasting time. But the rates, though needing no great accuracy in themselves, are each obtained from the difference of two temperatures, so that these temperatures need to be known with considerable accuracy. There may be, therefore, an appreciable error connected with the cooling correction, though this error is not in the cooling itself, but is an additional *accidental thermometric* error, coming in the determination of the cooling rate. Methods of diminishing or eliminating it can more conveniently be presented a little later.

Cooling correction calculations are easier to make if the rate is proportional to the temperature difference of calorimeter and jacket—that is, if Newton's law of cooling holds. The same advantages can be obtained where that law does not hold, however, by introducing the variation from it as a correction. Hence, in what follows, Newton's law will at first be supposed to apply, and the correction for deviations from it will then be discussed.

The simplest case of a complete cooling correction formula occurs where the heat of stirring and evaporation and the jacket temperature are known, for the amounts of direct cooling are then simply proportional to known temperature differences and only one cooling period is needed. For instance: Let ϕ_{tr} be the integrated temperature difference (of calorimeter and jacket) for the *transfer* period (i.e., the period when the heat to be measured is transferred to the calorimeter water), and ϕ_c the same for the subsequent *cooling* period. (The notation here is the same as in the *Physical Review* articles already referred to, except that the subscripts indicating the periods are numbers in those articles.) Let T_{tr} and T_c be the number of minutes in each period, w the heat lost by evaporation and stirring per minute (w, of course, will often be negative), $\delta\theta_c$ the observed temperature fall in the cooling period. Then $\delta\theta_c - wT_c$ is the direct heat loss in period (c), and that in period (tr) equals:

$$(\delta\theta_c - wT_c)\frac{\phi_{tr}}{\phi_c}$$

the cooling correction is, therefore, simply

$$wT_{tr} + (\delta\theta_c - wT_c)\frac{\phi_{tr}}{\phi_c} \qquad (1)$$

This can easily be manipulated into the form:

$$\delta\theta_c + (\delta\theta_c - wT_c)\frac{\phi_{tr} - \phi_c}{\phi_c} + w(T_{tr} - T_c) \qquad (2)$$

which is very convenient practically whenever the calorimeter is initially near the temperature of the jacket, for $\phi_{tr} - \phi_c$ will then be relatively small, often very small, while $T_{tr} - T_c$ will often be zero. Hence w, which is always multiplied by one or the other of these, need not be known at all accurately, and the two multiplications will give small products and so can be done by a slide rule, and usually by inspection.

Equation (1) shows how to avoid the thermometric error, mentioned above, which occurs in the determination of the cooling rate. That error comes in $\delta\theta_c$, which is multiplied by $\frac{\phi_{tr}}{\phi_c}$ hence the smaller ϕ_{tr} is, or the larger ϕ_c, the less will be the final error produced. ϕ_c can be increased by lengthening the cooling period, but this method is time-consuming and may tend to introduce other errors. ϕ_{tr}, however, can be decreased by taking the initial temperature below that of the jacket, so that the calorimeter receives heat from the jacket during part of the transfer period and gives it back again during the other part; and the total net cooling (proportional to ϕ_{tr}) is nearly zero. This is not Rumford's old rule of taking the jacket temperature *midway* between the initial and final calorimeter temperatures, but usually calls for a jacket temperature somewhat nearer the final. If this method is not used, and ϕ_{tr} nearly equals ϕ_c [as in formula (2)] the determination of the rate about doubles the maximum possible accidental thermometric error, but, in accordance with the laws of probability, only increases the probable mean error of a series by 42 per cent; hence this method of reducing the cooling correction cuts in two the maximum error, and diminishes the mean error about a third. One price which must be paid for this advantage is an exact knowledge of the value of W, the heat of stirring and evaporation, for errors in this value are no longer diminished as by formula (2).

The method just described, namely, that of diminishing ϕ_{tr}, can be used to save time as well as to gain accuracy. For since it enables us to diminish almost indefinitely the effect of errors in the value of the cooling rate, and, therefore, to get good results with rates inaccurately known, we may determine the relation of rate to temperature (the cooling factor) once for all, and then, assuming that this relation is constant, we may use it in each subsequent determination. We thus get rid of the cooling period altogether, yet with no loss of accuracy. The formula for the correction with this procedure is:

$$wT_{tr} + K\phi_{tr},$$

where K is the constant,

$$\frac{\delta\theta_c - wT_c}{\phi_c} \qquad (3)$$

Some observers have found that the cooling factors of their calorimeters were constant to ½ per cent or better, which suffices for an accuracy of 0.1 per mille even where the calorimeter is started at jacket temperature. The causes which may prevent this constancy do not seem to be fully known as yet, but it seems probable that the measurement of a cooling rate for each separate calorimetric determination will soon be generally regarded as not only a waste of time, but a detriment to accuracy.

Of the labor of getting the cooling correction there now remains only the determination of ϕ_{tr}—that is, the taking of temperature observations during the transfer period and the addition of the results. Even this labor is sometimes avoided, and the value of ϕ_{tr}, the total of the transfer period temperatures, is obtained from the initial and final values of ϕ by means of a table of some sort. The saving in this case is relatively not great, but is often easily obtained. As soon as anyone has performed a few determinations he has the material for expressing ϕ_{tr} in terms of the other temperatures. Of course, this method succeeds better the more uniform the various conditions—such as time, room temperature, material.

In many cases the jacket temperature and the heats of stirring and evaporation are not known. But *if they are constant* they can be eliminated by the use of another cooling period. The most convenient way possible of accomplishing this seems to be by the *Pfaundler formula*. The essential principle of that method is this: The proportionality between heat flow and the temperature difference (ϕ) of calorimeter and jacket cannot be used because ϕ is not known. But one consequence of that proportionality can be made useful, and that is: That the *difference between any two* cooling rates is proportional to the difference of the corresponding temperatures. Hence, not knowing the jacket temperature, we may refer our formula to any other temperature for which we know the rate and proceed much as before.

The additional cooling period furnishes the reference temperature in question. To illustrate, let this period, which comes before the transfer period, and is therefore a preliminary period, be denoted by the subscript, p. Since ϕ is not known, the *mean* temperatures of the periods will be used instead;[1] call these θ_p, θ_{tr}, θ_c. It will also be necessary to use, instead of the temperature intervals, $\delta\theta$, the rates, V_p, V_c, which equal $\frac{\delta\theta_p}{T_p}$, etc. Then the mean rate for the transfer period will be the mean rate for the preliminary period, plus the difference in rate due to the difference in temperature. The cooling correction, which equals this rate, multiplied by T_{tr}, the time of the transfer period, is, therefore:

$$T_{tr}\left\{V_p + \frac{V_c - V_p}{\theta_c - \theta_p}(\theta_{tr} - \theta_p)\right\} \quad (4)$$

which is essentially the Pfaundler formula.

Here two things are noticeable. (1) After the first term the formula is exactly like formula (1) in type, but instead of the temperatures of the two last periods there appear the differences between these temperatures and those of the preliminary (reference) period. (2) The heats of stirring and evaporation have vanished, since, being the same for all the periods, they are eliminated in the subtractions. This has always been counted one great advantage of the Pfaundler formula, and is, indeed, evident enough, but still seems to deserve emphasis, for some recent publications indicate that it has sometimes been overlooked.

As ordinarily printed the Pfaundler formula shows a summation, which is the same as the term $\theta_{tr}T_{tr}$ in the form here given. If the periods are all taken equal the computation is easier, since $\delta\theta$ can then be used instead of V. The computation is then as in formula (1), except that the three indicated subtractions must first be performed.

This formula, being like (1), can, like it, be treated as (1) was in getting (2), and thus becomes:

$$T_{tr}\left\{V_c + \frac{V_c - V_p}{\theta_c - \theta_p}(\theta_{tr} - \theta_c)\right\} \quad (5)$$

which is very convenient when θ_{tr} is nearly equal to θ_c, since then the computed term is so small that it can be obtained by approximate methods. θ_{tr} nearly equals θ_c when the calorimeter is started at about the jacket temperature, which is usually the most convenient way to do. I believe this variation of the formula is new.

The method described above of diminishing the accidental thermometric error by diminishing the final magnitude of the correction is practically useless with the Pfaundler formula. The proof is simple, and seems unnecessary here. There is, therefore, contrary to a frequently expressed opinion, little or nothing to be gained in accuracy by any particular choice of initial temperature if this formula is used, and the most convenient procedure may be employed, whatever that is. It will often be with the calorimeter temperature near the jacket temperature at the beginning, for which formula (5) is convenient. It also follows that the Pfaundler formula cannot give the highest accuracy, but its compensating advantages are often great. Where the environing temperature cannot be measured, because the environment consists of several bodies at different temperatures, the Pfaundler formula, or something equivalent, is indespensable, and this is probably the case with most existing calorimeters.

On the whole, then, the advantageous treatment of the cooling correction depends upon the observer's knowledge of conditions; the more he knows, the shorter and more accurate his work will be. If jacket temperature and heat of stirring and evaporation are all unknown the Pfaundler formula is essential, with three periods to observe. If these conditions are known the periods may be reduced to two, and accuracy may also be increased by suitable choice of initial temperature. If the cooling factor of the calorimeter is also known the periods may be reduced to one, with no loss of accuracy. If, finally, the form of the heating curve (i.e., during the transfer period) is known, the observations in that one period may be omitted, but, probably, with some loss of accuracy in many cases.

For *adiabatic methods,* also, essentially the same relation holds between the observer's knowledge of conditions and the ease and accuracy attainable. In these methods the jacket temperature, of course, must always be measured, but if the heats of stirring and evaporation are not exactly known two periods are necessary, with the same thermometric error as when formula (2) is used. The periods can be reduced to one, with thermometric error as small as with formula (3), and this requires (as for formula (3)) an exact knowledge of the heats of evaporation and stirring. The cooling factor need not be known for the adiabatic method, but this is not a matter of much importance, since a very approximate knowledge is enough in any case. The computation of the correction is saved in the adiabatic method, but an equivalent labor is substituted for it. In the "diabatic" method the observer reads certain temperatures, which he afterwards adds; in the adiabatic he duplicates these temperatures in the jacket, and must then refill the jacket and adjust it for the next determination.

Graphic methods are sometimes used for computing the cooling correction. Instead of *adding* the individual temperatures, the observer *plots* them, and then measures the plot with a planimeter. There is no question, however, that, with observations equally spaced in time, the easiest and most accurate way, by far, of getting their sum is to add them. The graphic methods are only valuable and, as a rule, have only been used in cases where the observations are taken at irregular time intervals. With such observations all the periods are of irregular lengths, and the labor of computation is considerably increased. The graphic method is merely a means of diminishing the inconvenience inherent in observations of this sort. These irregularly timed observations result from a method of reading which is often adopted with electric thermometers, but which is not in the least necessary; and is even less convenient with properly arranged apparatus. In general, therefore, they can be avoided, and with them the graphic methods which they make necessary.

The treatment of the *corrections for large intervals* remains to be considered. All the above formulas treat the cooling rate as proportional to the temperature difference of calorimeter and jacket, and owe their convenience to that proportionality. It is desirable to use the same formulas even where that proportionality no longer exists. The method of doing this can, perhaps, be best illustrated by the actual description of a simple calculation.

For this method it is necessary to know the difference of calorimeter and jacket, and this involves some measurement of the jacket temperature. But variations in the jacket temperature do not need to be known as accurately as for small intervals. And in the computation only the temperature difference is used; the absolute temperature of the system is of no importance practically.

Suppose, now, in a systematic study of the cooling of the calorimeter, the following results have been reached:

When the temperature difference was:	The cooling per minute was:
1°	0.002°
2°	0.004°
3°	0.007°
4°	0.010°
5°	0.014°

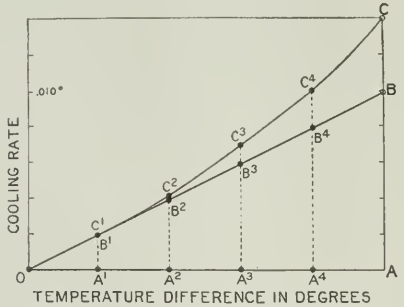
These results are plotted as in Fig. 1, where OA_1, OA_2, etc., represent the observed temperature differences, in degrees, and A_1C_1, A_2C_2, etc., represent the corresponding cooling rates of the calorimeter in thousandths of a degree.

[1] Of course ϕ may be used in the Pfaundler formula if this is desirable.

The straight line OB is now drawn through C_1. The points B_2, B_4, B_5 on it correspond to 3°, 4°, 5°. Since B is straight, and runs through the origin, the vertical distances A_2B_2, A_4B_4, etc., are proportional to th eorizontal distance OA_2, OA_4, etc.

Suppose, now, that a calorimetric determination is made, and that during it observations of temperature difference are made as follows:

1°, 3°, 4°, 5°, 5°, 5°, etc. These numbers have been obtained as a means of afterward finding the cooling that has taken place. That is their only function. It can be performed because to each number there corresponds a particular cooling rate. But a series of numbers *proportional* to the rates, if we had them, would enable us to obtain the rates more easily. Hence, if for the observed differences 1°, 3°, etc., we can substitute another set of numbers which are proportional to the desired rates, the purpose of the original observations will be accomplished and at the same time computation will be facilitated. Such numbers can be obtained from Fig. 1. Thus: the vertical distance A_1C_1 represents the rate for 1°. The ratio between rate and number is 0.002. The rate represented by A_3, C_3 does not bear the same ratio to 3°. But if the difference, 3°, is increased in the ratio $\frac{A_3C_3}{A_3B_3}$, becoming $\frac{7}{6} \times 3°$, or 3.5°, it will bear the desired ratio to the rate, for the rate is 0.007°, which is 0.002 of 3.5°. Similarly 4°, multiplied by the ratio $\frac{A_4C_4}{A_4B_4}$, or $\frac{5}{4}$, becomes 5°, which is proportional to the rate, 0.0100, which prevailed when the difference was 4°. Similarly, 5° becomes 7°, of which the 5-degree rate, 0.014°, is 0.002. And so on. Now, the condition for the



COOLING RATE AS FUNCTION OF TEMPERATURE DIFFERENCE.

cooling correction formulas given above was that rates should be proportional to temperature differences. The new temperature differences which we have just formed artificially fulfil this condition, and so they enable us to use *any* of those formulas with no further trouble.

In practice, the matter is much simpler than in the case just described, which was intended to illustrate the principle. For, (1), the curve of Fig. 1, is not used directly, but the values B_3C_3, B_4C_4, etc., are plotted in another curve on such a scale that they give at once the amounts which must be added to the temperature differences in order to obtain the proportional numbers. Thus, in the present illustration, against the abscissas 1°, 2°, 3°, 4°, 5° would be plotted the numbers 0, 0, 0.5°, 1°, 2°, and a smooth curve drawn through these points. Again, (2), the correction will almost never be needed for calorimetric intervals under 4°, and only occasionally even where the interval reaches 12°. For instance, in one case, where the cooling factor of the calorimeter increased 18 per cent as the temperature difference increased from 1° to 12°, it was nevertheless found that for determinations involving a 12-degree interval the correction for variation of the cooling factor was only $\frac{1}{20000}$ of the calorimetric interval, which, of course, is usually quite negligible. In many cases the total correction would not be negligible, as in this case, but the variations between different values of the correction would be, hence, after a few determinations had been carried through, a single constant value for the correction could be adopted, and no more computation would be needed.

The above method does not require that the cooling factor of the calorimeter remain constant from day to day. It merely supposes that the rates for different temperatures remain in nearly the same relation to each other, which will always be the case. The correction curve may be in error by 16 per cent, and usually much more, without causing perceptible error, as long as it is a smooth curve. But the different numbers obtained from the curve must not show *accidental irregularities* greater than the allowable error of the temperature difference observations, whatever this may be. This condition is easily secured by drawing the curve to a sufficiently large scale. (A discussion of the possible error in this method of correction will be found in the *Physical Review* articles already several times referred to.)

The present paper discusses (1) calorimetric thermometers. Electric thermometers, both resistance and thermoelectric, are at least ten times as accurate as mercurial, and are in some ways more convenient, though more expensive and complicated. (2) Cooling correction formulas are discussed, new forms being given which both cost less labor than the older forms and allow higher accuracy. Adiabatic methods are subject to the same principal limitations as those where corrections are used. In all, the accuracy and the ease increase with the observer's quantitative knowledge of such conditions as jacket temperature, heat of stirring, etc. (3) A method of applying ordinary formulas to large intervals is described in detail.

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Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in May, 1911, was 68. The total tonnage of ore milled during that period was 2,051,975 tons. There were 9,978 stamps in operation with an average duty of 7.66 tons per stamp in 24 hours. Tube mills in commission numbered 218. The yield for the month was 685,951 fine ounces gold. This record is one of the best ever made on the Rand, from the point of view of high tonnage and gross yield. The plant in operation is also greater than ever before.

Chrome iron is used extensively in the manufacture of special steels, either alone or with tungsten, manganese, nickel or other steel-hardening materials. It is also used for lining metallurgical furnaces to prevent excessive corrosion. For this purpose it is used in the crude form or as chrome brick. A third use is in the making of chromium compounds, such as the metallic chromates and alkaline bichromates for use as pigments, mordants and tannages. According to reports from the U. S. Geological Survey, the production of chromite in the United States fell off in 1910, amounting to only 205 long tons, valued at $2,729, as compared with the 1909 production of 598 long tons, valued at $8,300. Almost the entire production of chrome iron ore in the United States has come from California and Wyoming, mostly from the former State. The imports from New Caledonia, Greece and other countries amounted in 1910 to 38,579 long tons valued at $415,768, as compared with 39,624 long tons valued at $460,578 in 1909. Fifty per cent chromic oxide is the basis of valuing chrome iron ores, the price per ton rising or falling with higher or lower percentages.

The Marked Influence of Copper in Iron and Steel on the Acid Corrosion Test*

BY WILLIAM H. WALKER, PH. D.

The unwarranted use which has in the last few years been made of the name of the American Society for Testing Materials in urging the validity of the so-called acid corrosion test, concerning which definite action has been taken in the report of the Committee on Corrosion of this Society at the present meeting, has made the Society unwittingly more or less responsible for two rather grave errors. These are first, that the rapidity of a given sample of iron or steel to dissolve in acid measures its tendency toward natural corrosion, and second, that a low acid test is an index of great purity of the specimen—that is, of an extraordinarily high content of iron. Enough data are now available to show the unreliability of the first assumption, and it is the purpose of this short article to show the fallacy of the second.

In investigating samples of iron and steel which had withstood corrosion for years, and which notwithstanding dissolved in acid very readily, together with samples which withstood solution in acid in a remarkable manner, and yet were rusting at the ordinary rate, it was found that the acid resistant specimens contained in every case a substantial amount of copper; that is, the presence of copper seemed to be the controlling factor in the resistance to solution in acid.

Upon examining the literature of the subject sufficient data to justify this conclusion was found. A number of investigators have studied the matter, and the results of all are fairly well presented in the Carnegie Research Report of Pierre Breuil on Copper Steels in the Journal of the Iron and Steel Institute of Great Britain for 1907. It is here found that small amounts of copper in iron or steel reduce enormously the tendency to pass into solution in acid.

In order to determine whether there is any foundation for the alleged fact so widely disseminated, that a highly acid-resisting iron or steel is also therefore a very pure product, the writer succeeded in having made a number of heats in a basic open-hearth steel furnace, under ordinary conditions, and using ordinary materials, into which metallic copper was added a little before or during tapping. The treatment of the product in the bar and sheet mills, and the annealing operation, was in every case as nearly identical as could be maintained. Although as the work developed many heats were made, the results were so uniform that but few will be here presented.

	C	Mn	S	P	Cu	Loss in weight in 20% sulphuric acid in 3 hours.
A	.08	.50	.018	.017	—	.2235 grams
A_2	.08	.50	.018	.017	.21	.0075 "
B	.10	.41	.027	.026	.19	.0082 "
C	.09	.31	.031	.063	.19	.0095 "

The influence of small amounts of copper in these heats is truly remarkable and shows conclusively that this wonderfully slight tendency to dissolve in acid is not indicative of great purity.

Inasmuch as the presence of metallic copper in contact with iron *increases* enormously the rapidity with which such iron will dissolve in acid, it at first seems surprising that in this case copper should *retard* this action. But we have a direct analogy in the case of zinc and copper. Zinc free from iron dissolves in acid very slowly; mix with it some metallic copper, however, and solution is greatly accelerated. But make an alloy of the two, namely brass, and the solvent action of the acid is negligible. So long as copper is not added to iron or steel in excess of the amount which can remain homogeneously alloyed therewith, the writer can see no reason why there may not be an advantage in its use. But such advantage cannot be ascribed to purity, but rather to what we may call a ferro-brass of small copper content. Reasoning from analogy with zinc, pure iron should dissolve in acid very slowly, but the converse is here shown not to be true, namely, that a slow-dissolving iron or steel is on this account to be considered an exceptionally pure product.

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*A paper presented at the recent meeting of the American Society for Testing Materials at Atlantic City.

Electric Tin Smelting

A Commercial Scale Trial

BY JOHN HÄRDÉN.

Very little has as yet been said or published on the subject of electric-tin smelting; all that has been dealt with in the realm of the electrometallurgy of tin seems to refer to the detinning of tinplate. Stray notices may have appeared in transactions from time to time, but the writer has not been able to find any literature worth referring to, though this may be due to lack of opportunity.*

Even the smelting of tin in the orthodox way—by means of the reverberatory or the shaft furnace—is but scantily mentioned in textbooks, etc., and, as said in a recent editorial note in METALLURGICAL AND CHEMICAL ENGINEERING, very little is generally known, outside the circle of those directly interested in the subject, about the processes of the metallurgy of tin.

It may, therefore, be of interest to metallurgists to learn that electric tin smelting was carried out on a large scale as a commercial experiment in Cornwall, England, during last summer in a plant designed by the author and Messrs. Gröndal Kjellin Company, Ltd., London, as consulting engineers. In the following a brief report will be found of the work carried out, together with a few general remarks on tin smelting, which, of course, are not intended to embrace the whole scope of the subject.

Cornish Tin Smelting.

Tin smelting in the reverberatory furnace has been carried out in Cornwall since time immemorial, and the metallurgy of that process is, in a broad sense, very simple, though to make it an efficient one requires a good deal of minor technical details and long practice. Broadly speaking, the method is this:

The tin ore, chiefly consisting of stannic oxide, SnO_2, as mined in Cornwall, is being dressed in streams or on tables in the mines, so as to contain about 63 per cent to 64 per cent of tin. The ore is delivered in bags of about 3 cwt. each in the form of a dense, red-brown, moist powder, fairly finely ground, say about 15 to 20 mesh. After allowing to dry for some little time (depending upon the moisture, which may vary from 7 per cent to 15 per cent), the ore is mixed with the reducing agent—a low-grade anthracite called "culm"—which contains a considerable quantity of slatey clay or aluminium-magnesium silicate, etc.

The quantity of culm to be used depends, to a certain extent, upon the character of the ore. It is unusual to use one kind of ore only; indeed, the proper blending of different ores is one of the principal factors in producing an efficient melt. Bolivian or Straits tin ore is very often mixed with the Cornish ore. Generally about 20 per cent to 25 per cent of culm is used; for the first charges no other flux is added. One furnace is generally built so as to take a charge of three to four tons of ore and about three-quarters of a ton of culm; a certain portion of the culm acts as fuel. A quantity of the ores is previously mixed with culm in various proportions and smelted in a test crucible of graphite so as to give best yield.

It is very essential to conduct the smelt so that the temperature is raised gradually, especially if the ore is of a silicious nature, because if the temperature rise is too brisk tin silicate and silicides will be formed, which will go into the slag directly,

*See also: Reduction of Tin Dross in an Electric Furnace, Am. El. Ch. S., 1910, by R. S. Wile.

causing a great loss. Also volatilization must be guarded against, as the metal is comparatively strongly volatile. In more modern plants a condensing chamber is used to recover the losses. The charge is frequently rabbled with iron rakes so as to ensure an even distribution of the heat. In some plants a quantity of scrap iron is added to the charge before the melt is finished in order to cause impurities to be taken up by the slag rich in iron which is being formed.

The melt is finished after six to eight hours, when the tap-hole is opened and the metal and slag are run off together into a hemispherical cast-iron pan, which is often provided with special heating arrangements. Each pan will hold four to five tons of metal. The slag floats on the surface of the metal and is run off into iron boxes to solidify.

This first slag is generally rather rich in tin; it may sometimes contain 14 per cent to 16 per cent of tin in various combinations. It is, therefore, frequently used as an admixture to a subsequent smelt, or may be treated separately for tin. The final slag is said to contain about 6 per cent to 7 per cent of tin, and is, therefore, sometimes smelted together with lead refuse for solder tin. In more modern plants, however, the slag contains only 2.5 per cent to 3 per cent of tin.

The crude tin, as first obtained, is cast into large ingots, still containing a considerable quantity of impurities. These ingots are placed on the sole of the reverberatory furnace and remelted at slow heat and low temperature, so-called liquation process. The pure tin is thereby run off and the impurities remain behind, forming the so-called "hardhead," which chiefly consists of an alloy of tin with sulphides and arsenides of iron. The percentage of tin varies somewhat, but is always considerable, and this "hardhead" is causing the smelter considerable trouble. One of the objects of the electric smelting was to try to eliminate, if possible, the formation of this undesirable hardhead, which is so difficult to treat separately.

After the tin has been liquated it is collected in a large pan, or refining pot, heated from underneath. This tin is usually about 96 per cent to 98 per cent pure and still contains impurities of arsenic, tin-suboxides, etc. For further purification so-called "tossing" and "boiling" are employed. The former consists in scooping the hot metal in thin broad streams through the air, whereby the impurities are oxidized and float on the top. The "boiling," which is more common, consists in immersing a raw piece of wood of large dimensions in the metal, vigorously stirring the same. (Formerly it was believed that only wood from apple trees would serve the purpose properly.) A violent agitation sets in, oxidizing the impurities.

In modern plants dry steam and air are blown through the metal for the same purpose.

A small sample is ladled out from time to time and poured into the groove of the "test stone," a porous, white sandstone; the behavior of the solidifying metal, together with the fracture of the notched bar and the special sound of the "tin crying," tells the skilled smelter of the purity of his metal. Indeed, this test is so marked that an experienced smelter can tell within 0.2 per cent of the purity of the tin. Refined tin should have 99.6 per cent to 99.8 per cent of tin.

The refined tin is usually cast into 28-lb. ingots and branded with the trademark or else into strips of 18 in. and about ½ in. thick.

Electric Tin Smelting.

At a first glance it would seem that the electric furnace would not adapt itself very well to the smelting of tin ore, since the temperature of the electric furnace would appear to be unnecessarily high for the reduction, and even injurious. However, this assumption is quite erroneous. The reduction of stannous oxide by carbon requires a comparatively high temperature, especially if a good separation from impurities is to be effected. Besides, tin is readily volatile at this high temperature and in a reverberatory furnace with its wide hearth communicating directly with the smoke stack or else in the tin blast furnace where air is forced through the charge this volatilized metal is mostly lost altogether if complicated condensing devices are not provided for.

All this is mostly avoided with the electric furnace, if properly designed. Another feature is that the so objectionable formation of "hardhead" can be greatly reduced if the process is carefully conducted, as will be seen later. Finally, if the cost of power is not too high a decided economy, compared with the reverberatory furnace, is quite possible, as the experiments made have shown.

As to the losses of tin in slag, it has been conclusively shown that one has it in hand to so conduct the operation that slags containing only 0.25 per cent of tin may be produced in one melt; but it appears to be more economical to so conduct the process that a richer slag, containing, say, 14 per cent to 16 per cent of tin, is first produced, and to treat this richer slag while still hot in a second furnace as in this case the output is very much increased.

Theoretical Yield.

If tin oxide, SnO_2, is reduced by means of carbon, we may assume that the ideal equation is

$$SnO_2 + C = Sn + CO_2 \qquad (1)$$

but that part of the reaction takes place according to the equation,

$$SnO_2 + 2\,C = Sn + 2\,CO \qquad (2)$$

This is substantiated by the fact that a considerable quantity of CO can be seen burning at the mouth of the furnace; a certain amount of the CO will naturally serve as a reducer when passing through the charge piled up in the shaft, and, for reasons given before by the author in his article on "Electric Iron Reducation" (see Electrochemical and Metallurgical Industry, Vol. VII, 1909, page 16) it is of importance to so conduct the charge that the reaction (2) is predominant, though it will hardly be possible to realize this in practice to a greater extent than 50 per cent. Also the packing of the charge in the shaft, owing to the heaviness and fineness of the ore, will prevent this to a certain extent. We may therefore presume that under proper conditions, and after some hours' run, the reaction goes on according to 50 per cent of each of the equations 1 and 2.

The Heat Energy Required

According to Thomson and other investigators, 118 grams of tin (one gram-molecule) will yield 145,300 gram calories by combustion into tin oxide. This amount of energy will, therefore, be required for the liberating of 118 grams of metallic tin.

On the other hand, the combustion of 12 grams carbon into CO_2:

$$C + 2\,O = CO_2$$

will yield 96,960 gram calories,

and the combustion of 24 grams of carbon into 2CO:

$$2\,(C + O) = 2\,CO$$

will give (in round figures) 58,000 gram calories.

Assuming, therefore, that the reaction takes place according to equation (1), or $SnO_2 + C = Sn + CO_2$, we can calculate the energy requirements of the reaction per ton of metallic tin as follows: 145,300 — 96,960 = 48,340 kg calories per 118 grams of tin, or 409,661 kg calories per kilogram of tin, and as one kw-hour = 864 kg calories we have for one ton of tin:

$$\frac{409{,}661.1000}{864} = 474 \text{ kw-hours per ton of tin.}$$

In the same way we find that, if the reduction takes place according to equation (2) or $SnO_2 + 2\,C = Sn + 2\,CO$, 855 kw-hours per ton of tin are required.

Assuming that the reaction takes place to the extent of 50 per cent of either of the reactions (1) or (2), around 665 kilowatt-hours per ton of tin are required.

In addition to this, a certain amount of energy is required for the heating up of the charge to the temperature of reaction. No actual measurements of the temperature in the furnace were taken, but, judging from the appearance of the slag and the

metal tapped, it may be safe to assume that the main average of the temperature of the whole interior of the reactive zone was about 1400° C. to 1600° C.

Assuming that the ore contains 90 per cent tin oxide, we require 1280 kg of ore for each ton of tin. Experience has shown that about 16 per cent to 20 per cent of slag per ton of ore is an average, or about 220 kg of slag per ton of tin. The melting of 1 kg of slag requires, as an average, 500,000 gram calories = 0.6 kw-hour, or around 130 kw-hours for each ton of tin.

The heat capacity of the charge and the molten tin we may set at 65 kw-hours as being on the safe side.

The radiation losses through the furnace walls, electrodes and water jackets are naturally of such nature that no accurate figures can be obtained, but it seems safe to assume this loss to be equal to that consumed by the slag, or 130 kw-hours. The escaping gases, in the furnace used for our experiment, left the furnace mouth at a temperature of around 800° C.; the losses through this source may therefore be set at about 150 kw-hours for each ton of tin.

In summing up these items we therefore find that in practice it should be possible to produce one ton of tin with an expenditure of power equal to:

For the reaction	665	kw-hours
For the slag	130	"
For the specific heat	65	"
For the radiation losses	130	"
For the losses through gases	150	"
		"
	1,140	"

In the experiments carried out this theoretical figure naturally could not be obtained, as the furnace was too small, and also for other reasons given below, but a very fair result was obtained, as will be shown.

Practical Tests.

At the request of Mr. A. F. Maclaren, of London, the author was given the task of carrying out some experiments on these lines late in 1909. Some difficulty was at first experienced in finding a suitable place for the preliminary trials on a laboratory scale, but early in 1910 Dr. Walmsley, of the Northampton Institute, courteously permitted the tests to be made in the Electrochemical Laboratory of the Institute. A furnace had been designed for 800 amp and 50 volts alternating current, though only direct current of about 300 amp and 100 volts was obtainable at the Institute. The direct current is especially unsuitable for this class of work on account of the heating effect being more prominent on the positive electrode, which was fully manifest during the operation. Also electrolytic effects on the slag were to be reckoned with. In addition to this, the tests must be carried out so as not to disturb the ordinary work of the laboratory, which naturally also hampered the results.

In spite of these difficulties the following results were obtained, which proved the feasibility of the matter:

The furnace was built up in a manner somewhat resembling the one designed by Harmet for iron ore reduction. It was provided with a bottom chamber of magnesite bricks, rammed with calcined magnesite and tar, in the usual manner. A charging shaft rested on a vault or arch; the inner diameter of the hearth was about 5½ in. and the total height about 14 in.

Two lateral and inclined carbon rods about 2½ in. in diameter round were used as electrodes and reached into the center of the hearth about 2½ in. from the bottom. No cooling jacket was used, though it would have been of great assistance.

The wear and tear of these electrodes was so little, in spite of the primitive arrangement, that it could hardly be taken into account for the experiments; this was also manifested during later experiments.

A preliminary heat was made on Jan. 27, 1910. The charge consisted of:

Tin ore (about 63 per cent of tin)	20 lb.
Coke powder	5 "
Flux	5 "

Of this only 12 lb. of ore was actually reduced owing to lack of power. Totally about 40 kw-hours were spent, of which 15 were used for heating up and 25 for the reduction proper. The smelt yielded 5¼ lb. of pure bar tin and 1¼ lb. in shape of shots, thus corresponding to about 85.5 per cent of yield from ore. The slag was very irregular, as only one mutual taphole was provided for, so that some metallic tin remained in the slag. Thus the power expenditure amounted to around 4 kw-hours per pound of metal, or 8960 kw-hours per ton. Great losses of heat were experienced owing to the primitive arrangement of the furnace.

In the next trial the slag from the first melt was used as a flux. This proved to be a disadvantage, as it required a much higher temperature to be remelted, and the taphole froze very readily.

In a subsequent test 16 lb. of ore were smelted, giving 6½ lb. of pure bar tin and 2½ lb. of shots, or a total of 9 lb. of metal, equal to around 90 per cent of the metal in ore. The maximum power was 8.4 kw and the total duration of the melt two hours. The total power used, including heating up, was 19.2 kw-hours, or 2.15 kw-hours per pound, or about 4900 kw-hours per ton of metal.

The tin obtained direct from the furnace proved to be well over 99 per cent pure; it contained very little iron and no arsenic. The slag varied in appearance; if the run was smooth it flowed freely and showed a grayish glassy luster. Three average samples of this proper slag were analyzed and found to contain in the average 0.5 per cent of tin. The stiff black slag obtained in some melts contained, however, as much as 8 per cent of tin. Thus the best efficiency obtained from this little furnace, thermically, was about 25 per cent, while the metallurgical yield was as high as 90 per cent.

A test for electrolytic recovery of the tin in the slag was also made. Some slag and refuse containing an appreciable amount of metal was fused with soda (which may be carried out at once while the slag is hot). This was readily soluble in water and could be electrolyzed direct, with iron plates as cathode. A good deposit of tin was obtained, though the efficiency was not very high.

If, however, the oxide from the slag is converted into chloride of tin the theoretical power consumption would be about 1820 kw-hours per ton of metal. Assuming an efficiency of, say, only 50 per cent, this recovery may be possible, where the power price is low, with a rotating cathode and scrap tinplate as anode.

In view of the above promising metallurgical results it was decided to carry on experiments on a much larger scale on a commercial basis. A power supply of 100 kw was secured and a suitable site acquired where a galvanized iron building was erected, a tank for cooling water was built and every facility provided for a conclusive test.

The Electrical Equipment.

Current was obtained from a Diesel engine plant supplying power for a tin mining concern; the condition was that no disturbance or great fluctuation should be caused which might endanger the regular working of the pumps in the mine. The electric-power supply was three-phase, 50 cycles, 650-675 volts.

To meet this requirement a Berry three-phase transformer, oil-cooled, was installed, receiving constantly 660 volts on the primary side, in star connection, with only lightning arrester and main switch on the line; the secondary side, which was also in star connection with earthed neutral, had four tappings capable of giving either 30 volts, 40 volts, 50 volts or 60 volts The terminals ended in copper busbars on a slate panel provided with heavy copper screw plugs for obtaining the desired combinations. Each phase on the secondary side was provided with an ammeter; a voltmeter with a change-over switch served for

all three lines, while the recording and indicating wattmeters were on the primary side.

The busbars were connected with the furnace electrodes by means of each four flexible cables of suitable size arranged so as to give the least possible induction. As, however, the current was as high as about 2500 amp, a certain drop in power factor was inevitable; it was on the average about 0.90.

During the whole trial period no disturbance whatever was caused by the electrical gear, though one or two thunderstorms were experienced; the distance between power house and smelting plant was about 600 yd.

The Furnace.

The furnace used is a shaft furnace of special design. Owing to pending patent applications a detailed description of this, as well as of the details of the actual smelting process employed, will have to be deferred for a later occasion. One of the principal features in an ordinary reverberatory tin furnace is that the metal, like mercury, will percolate into the lining and brickwork in large quantities and remain lodged there until the furnace is broken up. In a large plant with a number of furnaces this means a locking up of a considerable capital for a long period, which, naturally, is objectionable. This is successfully avoided in the new design of the electric furnace, where only a minimum quantity of metal is withheld, all the metal being drawn off as soon as it is formed.

Three electrodes of carbon 8 in. x 8 in. square and 4 ft. 6 in. long, cooled by water jackets, were used. The electrodes were held by means of gripping holders extending over half the electrodes and moved upward as the ends are worn off. The holders were suspended by means of flexible steel rope and worm-gear winches, permitting of a fine adjustment.

No direct arc was permitted to be formed, as this caused a great variation of the yield and the working of the furnace. The charge formed a cone-shaped heap in the hearth, covering the reacting zone, permitting the electrodes to "burn free," forming a small space filled with incandescent gas round the ends which acted as a resistance, in which the reaction took place. This is, as is well known among furnace men, the only efficient way of properly working a furnace.

Arrangement was made to prevent the escaping of volatilized metal; only about 0.5 per cent or less of tin found its way out of the furnace.

Tapholes on different levels were provided for drawing off metal and slag separately. Sometimes the slaghole may be frozen up with a hard slag crust, especially if the ore is rich in iron which all went into the slag. In that case a small auxiliary electrode was used for burning the taphole open again, which could be made very easily, though it should be admitted that an oxy-acetylene jet would be even more handy.

The Smelting Process.

At the start of a campaign, which usually lasted 10-12 days, on account of the experimental nature of the trials, though there was no objection in keeping the furnace working for months in succession, the empty furnace was first pre-heated for about four to six hours by means of a small wood and coke fire in order to prevent cracks in the brickwork. In subsequent smelts this could be almost entirely dispensed with, since the lining had properly set.

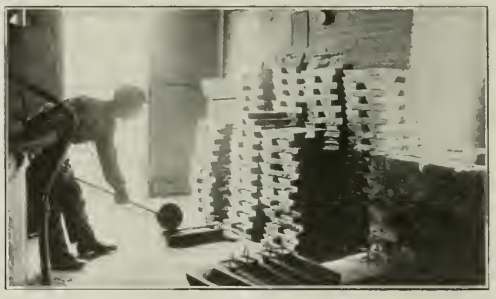
A charge of about 14 kg of culm for each 100 kg of ore, depending somewhat on the nature of the ore, was gradually filled in, while a current of about 1000 amp per phase, at 60 volts, was turned on. As the reaction commenced to set in the volts were lowered gradually to the working condition of 40 volts and 2500 amp, which was steadily maintained during the whole run. Only on special occasions, with certain ores, was 30 volts used.

The current fluctuated considerably at the beginning as the cold charge was being thrown in, but as soon as the electrodes had commenced to "burn free" properly the fluctuations ceased. Already half an hour after the first charge could the first tapping of metal take place, while the first slag was drawn after the furnace had been in proper working for a couple of hours. The appearance of the slag rendered it possible to the experienced eye to judge if the charge was of right composition or not; if of a glass-like, dark brownish green color, and freely leaving the taphole, the charge was right; if sluggish and light brown it had to be altered.

After a few hours' run, certain reagents were added at regular intervals; this successfully prevented the formation of "hard-head," causing the sulphides and arsenides of iron to enter into the slag instead of alloying with the tin. The result was that the metal, as it left the taphole of the furnace, was about 98 per cent pure, if the right blend of ore, not too much contaminated with impurities, was used. In one campaign ordinary Bolivian ore only, with only 49.5 per cent tin, and about 15 per cent of iron, was smelted solely for about ten days, and yielded over 92 per cent totally, on some days even 97 per cent of the metallic tin from the ore.

The liquid metal was accumulated in the usual way in large pots of wrought iron, heated from underneath and the sides. By means of an air-pump air was blown through the metal through a perforated iron pipe, whereafter the metal was ladled into a second tank to set properly. With this simple operation a metal of 99.75 per cent purity, and even more, was obtained, though this is well up to commercial standard.

The metal was then poured into ingots of about 28 lbs. weight, branded and stored in the usual way. The accompanying



STOCK OF INGOTS OF ELECTRIC TIN.

photograph shows a stock about 2 tons of ingots, being produced in about 1½ days.

Output, Power Consumption and Loss in Slag.

All three of these factors were found to be in close relation to each other. It was thus found that a slag containing as low as 0.25 per cent of metallic tin could easily be produced, but to do this in one operation proved to be less economical with the prevailing power price. The output was only about 6 to 8 cwt. per day, and the power consumption correspondingly high, or about 3000 kw-hours per ton of metal.

Going to the other extreme, the operation could be so conducted, by altering the composition of the charge, that a slag containing 17 per cent to 19 per cent of tin was produced. In that case the output increased to 1 ton 11 cwt. per day, with a power consumption of 1300 kw-hours per ton of metal.

As only one furnace was provided for, this manner of operation was not commercial, with such high losses of tin in the slag, although this slag could be treated again separately. It was found most economical under the prevailing conditions to operate the furnace so as to obtain a metallurgical yield of 90 per cent to 98 per cent, with a higher power consumption. The following weekly report for the week July 28 to August 4 is fairly typical:

Ore used: 21,056 lb. @ 57 per cent=	12,003	lb. metal
Scrap metal and charge refuse.... =	2,324	" "
	14,327	" "

Tin produced, 14,170 lb.
Net yield, 98.75 per cent.
Total number of kw-hours, 15,113.
To be deducted: for slag treatment (experiment), 678; for preparing furnace, etc., 500.
Total number of kw-hours used for reduction, 13,935.
Units per ton = 2220.
Electrode consumption, 28 lb. per ton of metal.

It should be noted that this figure for the power consumption was taken as an average over a single run of eight days, during which experiments with different charges were made. On single days the power consumption was only 1700 kw-hours per ton.

Conclusions.

The above figures bear out that a considerable saving in tin can be effected by means of electric smelting. If the plant is so arranged, as is so often done in other metallurgical processes, that two furnaces are working in series, the first one providing metal at a great rate continuously, and permitting the hot slag, rich in tin, to run direct into the second furnace for subsequent treatment, eventually with lead ore for solder in order to hasten the process, the power consumption should not exceed 1400 kw-hours per ton of tin (or about 55 per cent of theoretical) tin, in a sufficiently large furnace.

Even this figure may be decreased if proper blending of ores is strictly reverted to. In the above-mentioned cases single ores were mostly used, on account of the high price of the ore, which prevented the laying in of a large stock of expensive materials for experiments.

Also, it should be remembered that in a large commercial plant all the modern appliances for tin smelting should be applied, which again would help to increase the yield and lower the cost.

Another feature is of importance: The electric furnace will commence to yield pure tin already during the first hour of operation, and continue to do so without interruption, while the reverberatory furnace is intermittent in its operation.

Further, the formation of "hardhead" is materially minimized.

The amount of carbon necessary for the reduction is only 14 per cent, instead of 20 per cent to 25 per cent. Less labor is required, as three ordinary laborers per shift are quite capable to run the furnace.

Much less space is required for the electric furnace than for the reverberatory.

The consumption of electrodes is comparatively so small that it need not be regarded as an objectionable feature anywhere.

If the power can be obtained at a reasonable rate it therefore appears to be quite feasible to smelt tin electrically; especially should this be the case in the Straits Settlements, where large quantities of good ore, but not very rich in tin, are mined, and where waterfalls are said to exist in abundance, but only low grade of coal, and in small quantities. The use of charcoal, which probably would be much cheaper, should give no objection whatever.

In fact, with suitable flux, charcoal would probably work even better than culm.

Native labor could be employed under white supervision without hesitation, as practically no skilled labor at all is required. It is obvious that it would be much cheaper to ship metallic tin than an ore, which, as a rule, contains only about 55 per cent of tin. Naturally the working would at first cause a considerable amount of trouble, and would require a good deal of metallurgical and chemical engineering, but, on the whole, it seems justifiable to say that the principles of the problem are solved.

The above experiments will probably be resumed in a short time, in a somewhat modified way, by which an even greater saving is reasonably expected.

London, England.

The Right of Employer and Employee in Questions of Priority.

A recent decision in an interference case rendered by the United States Patent Office goes deeply into the legal right of employer and employee in questions of priority of invention. It is therefore of such a general and broad interest that we give here its salient features.

The case at issue was the following:

Davis E. Parker filed on May 18, 1906, serial No. 317,516, a patent which was granted him Jan. 5, 1909, bearing No. 909,017, covering an apparatus for preparing oxone.

An application covering practically the same subject was filed on Feb. 17, 1909, serial No. 478,508, by George Fred. Brindley, who was acting at the time in the interest of the Niagara Electro Chemical Company, Niagara Falls, N. Y.

To better illustrate the case we give herewith extracts of the decision rendered on June 9 a. c., which will give a clear understanding of the same, without going deeply into technical details of interest only to those immediately affected in the matter.

"This is an interference proceeding in which the subject-matter relates to an electrical furnace for fusing sodium peroxide. The fused peroxide has been given the trade name of 'oxone.' The issue, which is drawn to the details of the furnace structure, is defined in five counts . . . .

"The senior party, Parker, is a patentee, his patent No. 909,017 having been issued on Jan. 5, 1909, on an application filed May 18, 1906. Brindley's application was filed after the grant of the patent to Parker and is substantially a copy of said patent and the subject-matter of the interference is therefore the five claims of the Parker patent. The question at issue is one of originality rather than the ordinary one of priority of invention. Each of the parties has taken testimony in an effort to prove his claim as the originator of the invention in issue.

"At the time the invention in issue was originated both of the parties were in the employ of the Niagara Electro Chemical Company at Niagara Falls, N. Y. Brindley was works manager of the company and is shown to have had considerable training and experience in chemistry and metallurgy. He left the employ of the company in May, 1906. Parker was a machinist and was in charge of the machine shop of the company. He entered the employ of the company on March 28, 1904, and left in November, 1905. While both parties were in the employ of the same company, Parker's position was that of an assistant or subordinate to Brindley and the relation of the parties was therefore such as to bring them within the principles of employer and employee. In fact, there is no contention that the doctrine of employer and employee is not applicable to the case.

"The record shows that experiments for the production of fused sodium peroxide were begun by the Niagara Chemical Company at least as early as June, 1904. Brindley took up the problem at the Niagara Falls plant and during 1904 tried various kinds of apparatus. He claims to have built and tested a furnace like that shown in his Exhibit No. 1, and Parker admits having done the machine work on this furnace."

Then follow numerous quotations from the voluminous testimony taken in this case and the examiner then writes:

"These examples clearly indicate an indifference to accuracy on the part of Parker which cannot help detracting from the credibility of his testimony in a controversy of this character. It follows, therefore, that the question at issue will have to be determined largely from the circumstances of the case and the admissions of the parties together with the few facts that are not disputed or which are clearly established by the record."

The examiner goes thoroughly into the records submitted and comes to the following conclusion:

"Granting, therefore, that Parker did originate some of the features, it is not believed that he is entitled to claims covering a combination of such features with features which were admittedly originated by Brindley. In other words, if Brindley originated some of the features of the claim and Parker others, it seems clear that some claims must be regarded as the invention of Brindley and not of Parker. In fact the weight of decisions is believed to be clearly to the effect that such claims should belong to the employer rather than the employee. It is true that when an employee makes an invention which is independent of disclosures made to him by the employer and is able to prove his claim he and not the employer is entitled to the invention (Robinson v. McCormick, 128 O. G. 3289, 29 App. D. C. 98). But in no case known to this tribunal has it been held that claims to a combination made up of features admittedly furnished by an employer and others by the employee should be regarded as the property of the latter. In fact, the courts have frequently held that even where the features contributed by an employee amount to an independent invention a presumption exists in favor of the employer which can be overcome only by satisfactory proof that the independent invention was originated by the employee. On this point the Court of Appeals has said:

"What we hold is that when in the course of experiment by an employer with an invention a device is suggested for its improvement which in itself would reach the dignity of independent invention, and a dispute arises between employer and employee as to its conception, the presumption is justly in favor of the employer, and it is incumbent on the employee to overcome the presumption by satisfactory proof." (Miller v. Kelley, 96 O. G. 1038; 18 App. D. C. 163.)

"Evidently, therefore, in a case like that at bar where some of the features admittedly originated by the employer and the origin of the other features is in dispute a presumption exists that such other features were the work of the employer rather than the employee. And this presumption is strengthened when the particular facts and circumstances of the present case are considered. Brindley was admittedly the first to take up the problem of fusing sodium peroxide. He developed various forms of furnaces for this purpose before anything is claimed to have been done on the case in issue. He made the drawings (Brindley's Exhibits Nos. 3 to 10) which led up to the furnace in issue. He determined the size of the various parts which entered into the completed furnace, gave the order to build the furnace and, in fact, appears to have directed everything concerning the actual preparation of the first furnace involving the invention. Practically no evidence except the testimony of Parker himself has been introduced to show that any of the details were originated by Parker and this is not sufficient to overcome the presumption in favor of Brindley.

"Considering all the facts and circumstances of the case, it is believed that evidence presented on behalf of Brindley is sufficient to overcome the burden of proof under which he is placed and to show that the invention in controversy originated with him rather than with the senior party, Parker.

"On behalf of Parker it is contended that bar of public use exists the grant of a patent to Brindley and it is requested that, without deciding the question of priority, the examiner of interferences direct the attention of the commissioner to this fact. There appears to be no authority for the action requested. In fact, it appears to be well settled that where testimony has been taken and a case is in condition for final disposition action will not be suspended for the purpose of instituting public use proceeding (Brenizer v. Robinson, 166, O. G. 1281; Wert v. Borst and Groscop, 122 O. G. 2062; Doble v. Henry, 118 O. G. 2249). The question of priority has therefore been considered.

"The drawings made by Parker in the latter part of 1905 and the testimony relative thereto have not been discussed, as these matters appear to have little bearing on the question of priority. Likewise no reference has been made to the testimony bearing on the question of public use, although this testimony has been considered. It is not believed, however, that the circumstances of the case are such as to justify a recommendation as to public use under the provisions of Rule 126.

"Priority of invention of the subject-matter in issue is awarded to George F. Brindley, the junior party."

The case has been appealed.

Notes on Technical Analysis.

By R. S. Davis.

Analysis of Wulfenite Ores.—The Wulfenite ores found in the Southwestern States and Mexico usually contain more or less tungsten and vanadium, and as these elements exert a decided influence on the metallurgical process of extraction a practical method for the analysis of such ores was devised by the writer, as follows:

Pb. Mo. V. Treat one gram of the agate-ground sample with 10 cc. HCl and 5 cc. HNO_3, warm on hot plate and after the action has somewhat subsided add 10 cc. H_2SO_4 and evaporate to strong sulphuric fumes. Cool somewhat, dilute and heat until all is in solution excepting the lead sulphate and silica. Filter into a pressure bottle (a citrate of magnesia bottle with patent clasp stopper is well adapted for this purpose), washing with water containing a little sulphuric acid. Dissolve the lead sulphate in hot ammonium acetate and acetic acid, filter from the silica, washing well with hot ammonium acetate solution and estimate the lead by titrating with ammonium molybdate. If desired, one may add a large excess of sulphuric acid to the lead acetate solution, boil, add alcohol and determine the lead gravimetrically as sulphate.

Pass H_2S into the filtrate contained in the pressure bottle for about fifteen minutes, then stop the bottle, place in a vessel of warm water and heat to boiling for twenty minutes. If the amount of MoS_2 is small, and a preliminary test has shown copper to be absent collect the precipitate in a Gooch crucible, dry and ignite gently until there is no sulphurous odor, then heat to faint redness and weigh as MoO_3, containing 66.57 per cent. Mo. If this molybdic acid is not entirely soluble on warming with ammonia the residue should be deducted. It is safer, however, especially if there is much molybdenum, to proceed as follows:

Dissolve the MoS_2 in HNO_3, evaporate in a casserole to dryness then heat over a wire gauze to remove all sulphuric acid. Add 5 cc. ammonia and warm until the MoO_3 is all in solution (if copper is present use KOH instead of ammonia, dilute, boil, filter off the cupric oxide and estimate the copper by any convenient method). Filter if necessary, acidify with acetic acid, boil and add an excess of lead acetate. Filter and wash the precipitate with hot ammonium acetate solution to remove any trace of lead sulphate then with hot water, ignite gently and weigh as $PbMoO_4$, containing 26.15 per cent. Mo. The writer has not had much success in estimating Mo by the zinc reduction processes excepting when very little Mo is present.

Boil the filtrate from MoS_2, which will be green or blue if vanadium is present, to remove H_2S. Then add permanganate to a pink color. The amount of permanganate used will show approximately the Fe plus V present. Add an excess of permanganate and boil at least five minutes. Then add ferrous ammonium sulphate until the MnO_2 is dissolved and the liquid clear, and boil. Cool to about 70° C., add permanganate to a pink color, then run in ferrous ammonium sulphate until a ferricyanide spot shows an immediate blue color. Titrate back with permanganate until the spot shows no blue color in thirty seconds, read the burette and run in permanganate until a pink color is obtained, which persists one minute. The iron value of the permanganate multiplied by .916 gives the vanadium value. If the quantity of vanadium

is so small as to make its presence doubtful it is well to add H_2O_2 to the solution after titrating. This destroys the permanganate color and shows as little as .05 per cent vanadium by the characteristic red-brown tint.

The amount of iron present is approximately shown by deducting the permanganate used in the vanadium titration from the amount required to oxidize the iron and vanadium above.

W and Si are best determined in a separate sample. Treat one gram of the powdered ore with 20 cc. HCl and evaporate to dryness. Add 10 cc. HCl and about 100 cc. water and heat to boiling. Filter and wash by decantation with hot dilute hydrochloric acid until all lead chloride is removed from the precipitate. Add 10 cc. dilute ammonia containing a little ammonium chloride to the breaker containing the WO_3 and silica, allow to stand about five minutes and pour on to the filter, receiving the filtrate into a platinum dish. Wash four or five times using about 2 cc. of the ammonia solution for each wash. Evaporate to dryness, ignite gently until the ammonium salts are volatilized then at a moderate red heat. Do not heat excessively as WO_3 can be slowly volatilized with a strong Bunsen flame. Add a few drops of HF and a drop of sulphuric acid to the dish, evaporate, ignite, and weigh the WO_3.

Determine silica in the residue by loss with HF, and add to this result any silica found with the WO_3.

The Analysis of Ferro-Boron.—After considerable experimenting with various methods suggested for the determination of boron in metallurgical products, the writer has found the following plan to be both accurate and rapid:

Fuse one-half gram of the powdered sample with potassium nitrate and sodium carbonate in a platinum crucible. Pour the melt upon an iron plate, transfer with the crucible to a porcelain beaker and boil with just enough water to effect disintegration. Add a little sodium peroxide during the boiling to precipitate manganese. Filter into an Erlenmeyer flask and wash with hot water. Reserve the residue for manganese determination. Acidify the filtrate with hydrochloric acid and then add a moderate excess of calcium carbonate. Connect the flask with an upright reflux condenser and boil about ten minutes to remove all carbondioxide. Filter, cool to room temperature, add phenophthalein and run in deci-normal sodium hydrate to pink color; then add about one gram mannit. This destroys the color, and more sodium hydrate is added until the color is permanent, even on the addition of more mannit.

One cc. deci-normal sodium hydrate equals .0011 gm. boron. No deduction is required for blank.

One-half gram steel containing 4 per cent manganese, to which .03 gm. boron was added in the form of fused boric acid gave by this method .0299 gm. boron.

Dissolve the residue from the fusion in hydrochloric acid, add 15 cc. sulphuric acid and evaporate to strong sulphuric fumes. Cool, dilute, heat till clear and make up to 250 cc. Take 50 cc. for manganese determination by the bismuthate method, and 50 to 100 cc. for iron determination. Manganese may be made on a separate sample if desired by the bismuthate method without separating the boron.

Silica is determined in the usual way, and carbon by direct ignition in oxygen, spreading the sample over ignited asbestos in the combustion boat.

Primos Chemical Co., Newmire, Col.

The Electric Iron Reduction Plant at Trollhattan, Sweden—II.

In our July issue, page 368, was given a detailed description of the electric iron ore reduction plant at Trollhattan, Sweden, from a paper of Mr. J. A. LEFFLER, presented to the Jernkontoret (The Swedish Association of Iron Masters) at their meeting at Stockholm on May 31. The following is an abstract of the part of Mr. Leffler's paper in regard to the principles employed in the design of the furnace, analysis of materials, operating results, etc.

The Principles of the Reduction Furnace.

The fundamental principles of the construction of this furnace which was invented by Messrs. Grönwall, Lindblad, and Stalhane, is described in a communication from the Electrometals Ltd., of London, England, as follows:

1. As all known materials become good conductors at a high temperature, the construction must be such that the current does not pass away through the lining during the working.

2. The construction must be such that the charge itself protects the lining from destruction through radiation from the electric arc or through conduction; otherwise it would be necessary to employ water-cooling with consequent waste of energy.

3. The charge must not press harder against the electrodes than to permit the formation of an arc. If this condition is not fulfilled the working becomes irregular and the tension becomes unduly low, so that the electrodes and conductors would have to be of inordinately large dimensions.

FIG. 1.—FURNACE STACK.

4. In an electric furnace the carbon charged with the ore cannot burn, and in order to obtain a product of uniform quality it is, therefore, necessary to have a hearth of sufficiently large dimensions, to serve as a mixer. Otherwise every irregularity which inevitably occurs in charging will influence the carbon contents of the product.

In order to fulfil these conditions the hearth is constructed in the shape of a comparatively large crucible covered by an arched roof. When the charge sinks down into the crucible it spreads there in such manner that a free space is always formed between the charge and the brickwork of the roof. The electrodes are introduced into the crucible through the brick work at such points where it is not in contact with the

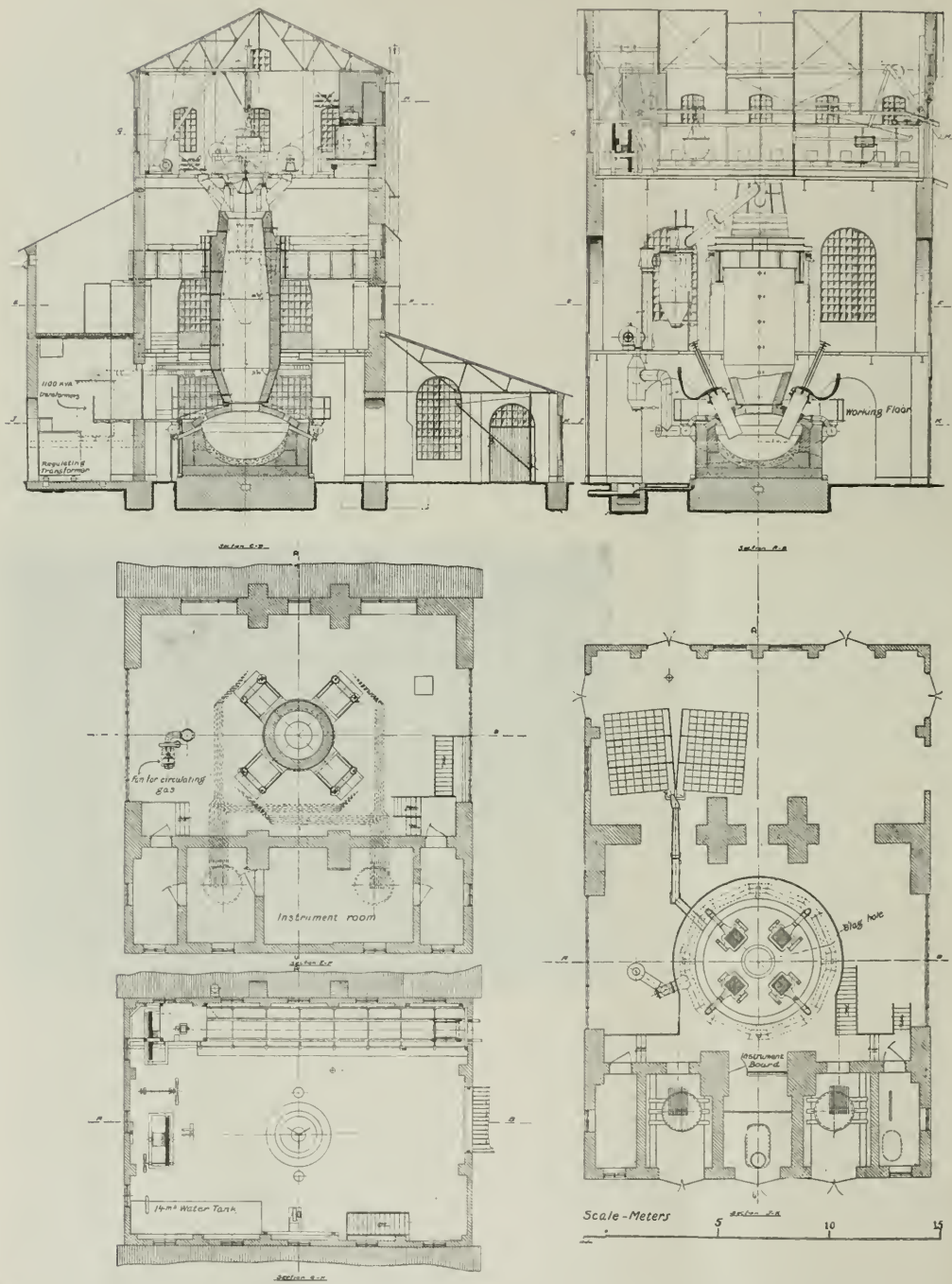


FIG. 2.—DIAGRAM OF ELECTRIC FURNACE CONSTRUCTION AND ARRANGEMENT OF ELECTRIC FURNACE PLANT AT TROLLHÄTTAN.

charge and enter the charge on the free surface exposed. By means of this construction the above enumerated conditions 1, 2 and 3 are fulfilled, and as the hearth can be made comparatively large condition 4 is also fulfilled.

Another important feature in a furnace of this kind is the regulating of the energy supply. In steel refining furnaces,

FIG. 3.—ROOF OF CRUCIBLE AND BOSH.

etc., this is done by raising or lowering the electrodes, but as a reduction furnace must necessarily work with a considerable gas pressure, this method cannot be conveniently employed, as it would be too difficult to keep the packings tight around the electrodes. Instead of this a system has been employed by which the electrodes are generally only adjusted corresponding to their wear, but all other regulating is accomplished by varying the tension of the current supply. This is done by cutting out windings on the high-tension side of the transformers and the arrangements are such that the different phases can work simultaneously with different tensions. This method for regulating the current which has been employed for the first time at Trollhättan has proved to greatly facilitate the working.

Principles Employed for Calculating the Form and Dimensions of the Shaft.

The starting point for the calculations was a manufacture of 7,500 tons of pig iron per year of 11 working months; that is about 23 tons per 24 hours. Further the calculations were based on the following assumptions:

1. 1 cu.-meter of charcoal weighs 150 kg.
2. 3 kg. of pig iron is obtained per 1 kg. charcoal.
3. For 1 ton iron 1,725 kg. ore is required, that is, 5,175 kg. per ton of charcoal.
4. One-half of the ore is contained in the interstices between the pieces of charcoal, the other half requires extra space.
5. The specific gravity of the ore is 2.5.
6. The ratio of volume of charge per day to shaft volume 1.55.

It is then found that 23 tons of pig iron correspond to a volume of 51.1 cub. meters of charcoal and an extra space of 7.9 cub. meters for the ore, or altogether 59 cub. meters. As the ratio of the volume of charge per day to the shaft volume is assumed at 1.55 the volume of the furnace should, therefore, be 59÷1.55=38 cub. meters.

This consists of the useful volume of the

crucible	12.45	cub. meters
neck	0.45	" "
bosh	3.50	" "
cylindrical portion of the shaft	11.80	" "
stack	9.80	" "
	38.00	cub. meters.

The diameter of the neck was the first dimension to be decided on, and this was fixed at 1200 mm. It appeared desirable to make it larger, but this would have made the crucible still larger and more unwieldy, as the inside diameter would have had to be increased in proportion on account of the electrodes. The depth of the crucible was determined on the strength of the experience with the Domnarfvet furnace and its bottom and sides designed in suitable form.

It is evident that with this diameter of the neck the diameter of the bosh could not be made very large. It also appeared desirable that the height of the shaft should be such that the best possible effect should be obtained from the ascending gas for the preheating and partial reduction of the ore. On the other hand, it might be expected that a too great height of the charge would cause a too great back pressure against the ascending gases. Also the cost of construction would be increased considerably, with an increased height of the shaft.

FIG. 4.—FURNACE TOP.

The problem then was how to best distribute the 25.55 cub.-meters of space that the effective portion of the shaft should contain.

A high and narrow shaft would give a good efficiency from the gas and reduce the risk of scaffolds forming in the bosh.

At the same time no undue increase of the cost of construction should be caused, and an excessive back pressure against ascending gases in the shaft must be avoided. The result of these speculations was the form shown in Fig. 1. (For convenience this is reproduced here again from our July issue.) The reasoning was that if the shaft should prove too high the remedy would be to work with a lower charge. The total height of the furnace was thus fixed at 12.7 m. from the bottom of the hearth to the furnace top. The bottom of the hearth is one m. above the floor level, and the height from the ground level to the furnace top is therefore 13.7 m.

FIG. 5.—FRONT OF FURNACE WITH TAP HOLE.

The Charcoal.

Apart from the small quantity of coke used for starting the furnace, charcoal has been used exclusively for the reduction. The charges are made by volume (0.65 cub. meters per charge), but all of the charcoal is weighed and analysis made every day.

The average composition of the charcoal is 14.58 per cent H_2O, 10.26 gases, 3.06 ash, 72.10 fixed C, 87.95 C, 2.60 H, 0.15 N, 0.61 O.

The average weight has been 168.6 kg. per cub. meter, corresponding to 144.9 kg. per cub. meter of dry charcoal. Both moisture and ash have occasionally been unduly high, in the latter case owing to stones and gravel mixed with the charcoal.

The average results of the daily analysis of the ash give 39.1 per cent SiO_2, 7.3 $K_2O + Na_2O$, 19.55 CaO, 3.85 MgO, 23.1 $Fe_2O_3 + Al_2O_3 + Mn_3O_4$, 0.95 P_2O_5, 3.3 SO_3, 0.6 Cl, 1.0 CO_2.

Besides the regular daily routine analyses a number of special analyses were also made.

The average of 99 special analyses of the ash gave 38.02 SiO_2, 6.1 $K_2O + Na_2O$, 19.52 CaO, 3.59 MgO, 22.69 Fe_2O_3, 3.88 Al_2O_3, 1.89 Mn_3O_4, 1.01 P_2O_5, 0.93 SO_3, 1.35 CO_2.

The analyses have also shown that the volume of gas obtained per gram of charcoal (dry and free from ashes) in in heating to 900 deg. Cent. was 170.35 cu. m. in average. The loss in weight of charcoal (dry and free from ashes) in heating to 900 deg. Cent. was 12.65 per cent. The weight of the gas was 9.22 per cent of the charcoal, and its composition in per cent of weight was as follows:

CO_2	CO	CH_4	H
22.47	38.03	30.47	9.03

Ore and Limestone.

The ore and limestone are broken up in a Blake crusher. This machine is large and powerful, as it was expected that

principally unroasted ore in large lumps would be crushed. Such has also generally been the case.

As the crusher delivered a coarser product than desired, a plate was fitted under one of the jaws so that the ore which at first had been of egg size was reduced to nut size.

The following are a few of the analyses of the limestone and ore used:

Limestone from Gasgrufvan: 54.32 CaO, 0.24 Fe_2O_3, 0.35 MnO, 0.31 MgO, 1.68 SiO_2. Loss on ignition 42.94.

Iron ore from Tuolluvaara: 91.05 Fe_3O_4, 1.41 Fe_2O_3, 0.16 MnO, 1.12 MgO, 0.28 CaO, 0.16 Al_2O_3, 0.10 TiO_2, 3.42 SiO_2, 0.050 P_2O_5, 0.011 S.

Iron ore from Langban: 89.85 Fe_2O_3, 2.49 FeO, 0.18 MnO, 0.85 MgO, 0.99 CaO, 0.42 Al_2O_3, 5.08 SiO_2, 0.014 P_2O_5, 0.007 S.

The chapters of Mr. Leffer's paper, dealing with the behavior of the electrodes and the general working results, will be given in our next issue.

A Plea for an Inventory of the Coal Supplies of the World*

By Prof. Joseph W. Richards.

I will take a few minutes of your valuable time to speak on this subject from the standpoint of the metallurgist—the man who uses the ore which the geologist describes and which the mining engineer takes out of the ground.

I cannot agree with some of the detailed conclusions of the very valuable report presented to the Congress upon The Iron Ore Reserves of the World. For instance, the 250,000,000 tons of Lake Superior iron ore carrying between 35 and 50 per cent of iron are placed neither in the actual reserves nor even in the potential reserves and yet this enormous quantity of ore, averaging much better than the "Minette" of Luxembourg, should in my opinion be placed under the actual reserves of ore in the United States. The reason for this statement is that this ore is not now worked simply because we have, at present, all we need of the richer ore, on which we make a larger profit, but if we did not have this richer ore we would certainly be working 40 to 50 per cent ore, and working it at a profit. I believe that within twenty years this immense quantity of ore, which the report does not consider as ore at all, will be the principal supply of iron ore in the United States, and will furnish us ore for many centuries to come.

Aside from the above criticism, I wish to support the points raised by M. de Launay and Professor Kemp, to the effect that metallurgists are continually improving their methods, and that the improvements which are certain to be made as time passes will enable metallurgists to work poorer and poorer ore at a profit. In twenty years we will be making pig iron from 40 per cent ore as cheaply as we now make it from 50 per cent ore; this is very certain and so I believe that improvement in metallurgical processes will more than compensate for the decreasing richness of the ores.

This very valuable report on iron ores of the world has therefore made it very clear to the metallurgist that there will never, in all the future history of the world, be any scarcity of iron ores for the metallurgist to work upon. Certainly, some localities will become exhausted, and the centers of production may change from place to place, but there will always be enough iron ore in the world to supply its needs.

A much more important question is the quantity of coal in the world with which to reduce this ore. This necessary material will be the real criterion of our future iron supply. Even at the present time the countries which have the most coal are dominating the iron supply of the world.

As already indicated by Prof. Kemp, the International Geological Congress would perform a very great service if it would call upon eminent geologists in all countries to make a careful inventory of the actual and potential coal reserves of the world. Such a report would not be more difficult to make than the present report on the iron ores, but probably less difficult, and it would be a splendid and highly valuable supplement to the report before us, giving us more real information about the future supply of iron and its localities of production. I wish the Congress would take up this task, and give the world this much-needed information.

However, in the distant future, when coal is exhausted, the world must produce iron by the use of the energy of waterfalls and electrical reduction, with charcoal from wood as the reducing agent. Here in Scandinavia, with no coking coal, you have already begun this era, you have been the first to show the world the economic possibility of producing pig iron electrically, using only water-power and charcoal. With your very pure ores, cheap water-power and abundant supply of wood, you will develop in Scandinavia a large electro-metallurgical iron industry. It is not a matter of the distant future for you, but within twenty years your iron industry will have been revolutionized and have become electrical. I congratulate Scandinavia on being in the lead in this new era of metallurgy, just as it has always led the world in the production of the purest commercial iron.

*An address delivered before the International Geological Congress in Stockholm, Sweden.

Notes on Chemistry and Metallurgy in Great Britain

(*From our London Correspondent.*)

The Chemical Society's Faraday Lecture.

The Fundamental Properties of the Elements.

On June 14 Prof. Theodore William Richards, of Harvard, delivered this year's Faraday lecture at the Royal Institution, and was afterwards presented with the Faraday medal. The following is an abstract of the lecture:

The lecture concerns the methods and results of some extended series of investigations designed to add to the foundations of human knowledge by careful experiment. Among quantities worthy of exact measurement the properties of the chemical elements are surely some of the most fundamental because the elements are the vehicles of all the manifold phenomena within the range of our perception. Weight is one of the most significant of these properties, and in the determination of atomic weights each substance employed must be free from all suspicion of impurity. Volume is also important. Twelve years ago the quantity b in Van der Waals' equation seemed to the author to be not really constant but subject to change under influence of temperature and pressure. This quantity is supposed to be dependent on the space occupied by the molecules, and if it be changeable are not the molecules themselves compressible?

If changes in the bulk of molecules are to be inferred even from gases, may not expansion and contraction of solids and liquids afford a much better clue to the relative expansion and contraction of these molecules? The ordinary conception of a solid has always seemed to me little short of an absurdity. A gas may be imagined with moving particles far apart; but could a rigid body like steel have such an unstable structure? The evidence seems to show that the interstices between atoms in solids and liquids must be small relatively to the size of the atoms themselves, if, indeed, there are any interstices at all. The compressibilities of 35 elements were studied with sufficient care to leave no doubt as to their relative values. It became manifest that the formation of a compound of a compressible element was attended with greater decrease of volume than was the formation of a similar compound of a less compressible element, other things being equal. This is just what the theory leads us to expect, and is inexplicable by any other hypothesis known to me. Then as to cohesion, if the pressure of chemical affinity causes compression of the atom, may not

the pressure of cohesive affinity do the same? The affinity which prevents solids and liquids from vaporising is generally admitted to set up great internal pressure, and must it not tend to compress the molecules? Molecules of high cohesive affinity—those of substances hard to volatilize—should be much compressed and have a small volume, whilst molecules with little cohesive affinity should be larger. Moreover, those molecules already much compressed by their self-affinity would be only slightly affected by additional pressure. Given two otherwise similar substances, the less volatile would be less compressible, denser, and have greater surface tension. These outcomes of the theory correspond with observed facts in most of the cases so far studied. Differences of structure and of chemical nature sometimes hide these relations, but the bulk of the evidence strongly indicates that cohesiveness, as well as chemical affinity exerts pressure, and that each plays a part in determining the volumes occupied by molecules.

Carried through to its logical conclusion, the idea that atoms are compressible gives one quite a new conception of the molecular mechanics of the universe. The influence of atomic compressibility may be perceived everywhere, and in most cases each fact seems to fit easily and without constraint into its place in the hypothesis. Even apparent exceptions, such as the abnormal bulk of ice, may be ascribed in a reasonable fashion to superimposed effects.

The lecture was received with marked appreciation by a full audience.

The Influence of Temperature on Corrosions.

A paper on "The Action of Salt Solution and of Sea-water on Iron at Various Temperatures," by J. Newton Friend and J. Hallam Brown was read at a recent meeting of the Chemical Society. The authors point out that the results obtained by Heyn and Bauer indicated that solutions of chlorides and sulphates of the alkali metals always exerted less corrosive action on iron than did fresh water at the same temperature. As these results did not conform to the observations of other chemists it was thought desirable to investigate the whole question. Kahlbaum's pure iron foil was used, access of dust was prevented while free passage of air was allowed, unequal illumination was guarded against and equality of conditions was maintained. The loss in weight after 14 days was taken as a measure of the corrosion, and the results showed that at temperatures above 13 deg. C. salt solutions of all concentrations exert a less corrosive action than distilled water, but below this temperature the more dilute solutions are much more corrosive than fresh water. At 13 deg. dilute solutions of salts were practically equally corrosive as fresh water. The results are of importance in this country, where natural waters are usually at a lower temperature than 13 deg. C.

The Measurement of High Temperatures.

At a recent meeting of the London section of the Society of Chemical Industry Mr. C. T. Heycock, F.R.S., read a very interesting paper on the modern methods of measuring high temperatures. The discussion was opened by Mr. Darling, who said he had the impression that Siemens had given up his platinum resistance pyrometer, and that nothing further was effected with it until Callendar and Griffiths worked on it; but the difficulty arising from the heating of the stem was overcome by covering the platinum wire with magnesia instead of winding it on mica. The relative merits of resistance pyrometers and thermo-couples depended on the use to which they were put. For exact determinations of boiling points or melting points the platinum resistance pyrometer gave the closer results; but if the two instruments were compared in a furnace at a high temperature for a considerable time experience showed that for workshop purposes the thermo-electric was the better. Prolonged heating of platinum wires at slightly over 1000 deg. C. materially altered their resistance, and the calibration was then wrong. Taking a reading even with such indicators as the Whipple was a troublesome matter, involving balancing with the Wheatstone bridge, and, even in skilled hands, by the time a balance was obtained the temperature might have varied considerably.

With the thermo-couple a direct reading was got at the instant, so that it had a decided practical superiority. The deteriorations in thermo-junctions occurred equally with platinum, and consequently the thermo-electric instrument was not inferior in this respect. As to accuracy, it was not possible with a large furnace—gas or coal-fired—under workshop conditions to maintain a constant temperature within ten degrees, and it was useless to employ a method giving readings to a fraction of a degree. There were many instances—notably the Royal Arsenal at Woolwich—where after thorough trial resistance pyrometers had been given up and replaced with thermo-couples, mainly because an installation of the latter could be worked from a central control if all the wires were identical in composition.

It was only fair to say that resistance pyrometers had recently been made quite automatic, but it was not possible to adapt even such an instrument as the new Paul indicator to an installation of 50 resistance pyrometers. Another important advantage of the thermo-couple was the facility of repair, while if the resistance instrument got out of order it was best to return it to the maker at once. For laboratory use in getting precise results no doubt the resistance pyrometer was the better.

He wished to refer to a curious effect observed with the Féry pyrometer in which a spiral was employed. When heat was concentrated on that spiral the pointer moved to a certain point on the scale, remained stationary for a moment or two, and then moved further. Which reading should be taken? They recommended taking the first, but that was not always correct, and there was something about it which required investigation. The form with the spiral was not so nearly accurate as that with the thermo-junction and galvanometer.

As a practical example of the accuracy now attainable at high temperatures he would state that in taking temperatures at about 1500 deg. C. some time ago with a Féry pyrometer and an Holborn-Kurlbaum optical instrument, by Siemens, as a check, the two agreed perfectly and responded equally to fluctuations of about 10 deg. either way. They need not fear that pyrometers could only be used in the laboratory, for they could be used with confidence in the workshop.

Dr. R. Lessing asked how readings on the Féry pyrometer were affected by glass or mica windows. He found it best to put the glass on for adjusting the distance and remove it for the actual reading, which was, of course, higher than the reading through the window. He wished that the author had touched on the subject of testing by Seger cones. It was not a very exact method, but was used largely in ceramic industries and had been brought to some degree of perfection; moreover, it could be employed where the use of pyrometers was inconvenient or even impossible.

Prof. W. R. Hodgkinson thought Mr. Heycock had rather neglected the thermo-couple. He considered that it was the pyrometer of the day on account of its simplicity and permitting the attachment of a recorder. In metallurgical works where pyrometers were in constant use, night and day, a recorder was wanted to show what had been going on at any time. True, there was the fact that Rhodium was attacked by carbon compounds, carbon monoxide, etc., and therefore must be protected. No doubt some vapor—probably a carbon compound—came from the iron, but when the wires were enclosed in a silica tube difficulty from that source was overcome.

Mr. W. Macnab said he had no experience of resistance pyrometers, bue he had found the thermo-couple a most convenient instrument for the practical determination of high temperatures.

Mr. Gardner asked if it were quite certain that iridium volatilized.

Professor Hodgkinson: There is not the slightest doubt about that.

Mr. M. J. Butterfield remarked that Mr. Heycock had stated that the range of nitrogen-filled glass thermometers was below 400 deg. C., but they were now made of "boro-silicate" glass with scales extended to 550 deg. and 600 deg. C. They were very convenient, and he found them in fair agreement with thermo-couples.

Mr. Heycock, replying, said his object had been to point out the value of the resistance pyrometer for exact work, and he quite agreed that for practical workshop use it was not so convenient as the thermo-couple. He should hesitate to use a glass or mica window, but would employ a plate of rock salt. He was interested by what had been said about Seger cones, and confirmed Dr. Lessing's observations. As to the volatilization of iridium, the resistance of an alloy of iridium and platinum at high temperatures underwent a much greater change when kept hot for some hours than that of ordinary platinum.

Heat Treatment of Steel.

A. Jung has published some interesting details of his researches on the influence of heat treatment on the tenacity and microstructure of hyper-eutectoid steels contained from 0.99 to 1.56 per cent of carbon, 0.24 to 0.46 per cent of manganese, 0.019 to 0.032 per cent of silicon, 0.020 to 0.031 per cent of phosphorus and no sulphur. These were heated in an appropriate carbon resistance furnace, and the changes in the values obtained by mechanical tests were noted as well as the microstructure after the following variations of treatment: (a) Heating for equal times at different temperatures and hardening in water; (b) tempering at the temperature found to give the best results and noting the results obtained with variation of time; (c) quenching in baths of oil and lead-tin alloy at the same temperature as the tempering was effected at; (d) tempering of the steels hardened in water both at varying temperatures for equal times and for varying times at the same temperature; (e) annealing and subsequently noting the results obtained on mechanical tests, contraction of area, nature of fracture, quality of grain and hardness.

The investigator has arrived at the following conclusions: Such steels are best hardened at temperatures confined to a narrow range, which is as the carbon content; for any given carbon content the ultimate strength is inversely as the temperature at which quenching is effected; at constant quenching temperature the ultimate strength is as the carbon content; the quenching temperature should be the same for all baths; quenching in water and tempering at 350° C. produces the greatest degree of strength; by quenching in water at a low temperature and tempering a somewhat brittle but highly tenacious metal is obtained, but when quenching is done at the same temperature as the tempering the tenacity becomes less while the toughness increases; identical results cannot be obtained from varying kinds of heat treatment, and can only ensue with any one property of the steel; the mechanical properties are dependent on the conditions in which the carbon exists and on the metallic structure; structures containing martensite give the highest degree of hardness, tenacity is increased by troostite and sorbite as elongation is by pearlite; finally no consistent influence of the carbon content on the mechanical properties of the steels was detected.

The Cementation of Nickel Steel.

An investigation by F. Carnevali and F. Giolitti on steels containing from 2.03 per cent to 29.8 per cent of nickel, when cementation was effected with pure ethylene and carbon monoxide, shows that the maximum carbon content in the zone of cementation diminishes as the percentage of nickel increases, and becomes very marked when there is more than 5 per cent of nickel. The variation of concentration of carbon in the layers of the zone of cementation is more regular in nickel steel than in carbon steel. When nickel steels are so cemented as to show a hyper-eutectic zone, the phenomena occurring during slow cooling are similar to those exhibited by similarly treated carbon steel, but differ quantitatively from the latter in that the entity of the liquation phenomena is not so marked in the nickel steels, and the transition from the hyper-eutectic to the hypo-eutectic zone corresponds with that of a carbon steel having a lower carbon value, viz., 0.6 to 0.65 per cent as against 0.9 per cent.

Electrolysis of Sodium Chloride Solution.

R. Sacerdoti has investigated the effect of rise of temperature on the potentials of the electrodes in the electrolysis of sodium chloride solution, with platinum and with iron cathode and platinum and magnetite anodes. In the cathode curve the effect of temperature is but slight, but in the anode curves it is apparent that as the result of a rise of temperature from 20° C. to 100° C. the anode potential corresponding to a given current density falls 0.4 to 0.8 volt.

A diagram shows the relation between electric potential and temperature for different current densities at both electrodes, and demonstrates that the reduction in voltage due to rise of temperature is never more than 0.75 volt, and that when high densities are used the figure is always remote from the potential of reversible polarization. With a decidedly alkaline solution the reversible anode potential is 1.3 volts and the reversible cathode potential 0.8 volt.

Copper, Nickel, and Cobalt Ores.

An investigation by H. Pedersen of a mixture of magnetic pyrites, nickel ore and slag consisted in digesting the mixture with concentrated nitric acid and treating with sulphuric acid until SO_3 was evolved; when the solution was cold it was treated with water and again evaporated until fuming occurred. The residue was treated with water, and after filtration from the slag the filtrate was precipitated with ammonia, and the ferric hydrate dissolved in dilute sulphuric acid, and after reprecipitation was estimated by the usual titration. The solution containing the copper and cobalt was acidified with sulphuric acid, evaporated to a convenient bulk, and after addition of a small quantity of strong sulphuric acid the copper was estimated electrolytically. The solution containing the nickel and cobalt was neutralized with ammonia in distinct excess and the solution submitted to electrolysis. The nickel and cobalt on the cathode were dissolved in excess of nitric acid, the solution evaporated to dryness on the water bath. The nitrates were eliminated by a small addition of hydrochloric acid and repeated evaporation, then redissolved in dilute hydrochloric acid and suitably diluted; the nickel was then precipitated by an alcholic solution of dimethylgloxime and ammonia; the precipitate was collected in a dried and weighed Gooch crucible, and after drying for about two hours at about 110° C. the precipitate contains 20.31 per cent of nickel.

Engineering Imports and Exports.

According to the Board of Trade returns for the six months ended with June, the imports of iron and steel, including manufactured articles, had a value of £5,685,566, an increase of £1,343,165 on the value for the corresponding period of 1910; the exports amounted to £22,160,641, and showed an increase of £1,212,451. For other metals the imports were £13,446,938, an increase of £1,854,921, and exports were £5,485,763, an augmentation of £443,020. In electrical goods there is a decrease of £32,307 in imports and also a decrease of £338,361 in exports. The value of imported machinery is £3,148,648, an improvement of £867,086; while exports amounted to £15,729,408, showing an increase of £1,648,922. The value of imported ships is £47,040 and of exported ships £2,532,246, and there are respectively an increase of £45,950 and a decrease of £1,768,181.

Market Prices.

July, 1911.

Copper has been fairly steady throughout the month, with

downward tendency, dropping more sharply after the 24th. Opened at £57-5-0 and closes at £50-7-6.

Tin rose over the first fortnight, though not steadily, opening at £194-10-0 and reaching £196-15-0 by the 14th. It then dropped sharply away to £188-10-0, recovered to £193-10-0 by the 19th, and closes £189.

Lead was steady at £13-15-0 to the 18th, when it commenced to rise, reaching £14-3-9 by the 20th. Closes at £14-5-0.

Hæmatite dropped after opening at 62/- to 60/10, recovered by the 10th, and has since been from 61/3 to 61/4½.

Scotch Pig rose from 52/3 smartly to 53/-, but has since eased to 52/10.

Cleveland rose from 46/3 to 47/- and remained fairly steady, closing at 46/10½.

Rubber rose but has not exhibited very extensive fluctuations, being from 4/1 to 4/7 (19th) and closing at 4/8.

	£ s. d.
Alum, lump, loose, per ton	£ 5 12 6
Antimony, black sulphide powder, per ton	22 0 0
Borax, British refined crystal, per ton	16 0 0
Copper ore, 10 to 25 per cent, per unit	9s. to 10 0
Copper sulphate, per ton	19 13 0
Carbolic acid, liquid 97/99 per cent. per gal	1 4
Creosote, ordinary good liquid, per gal	2¼
Camphor, 1-oz. tablets	1 9½
Caustic soda, ash 48 per cent, ordinary, per ton	5 10 0
Litharge, flake, per ton	15 15 0
Mica, small slab, original cases, per lb	6d. to 2 0
Naphtha solvent, 90 per cent to 160° C., per gal	9
Petroleum, Russian spot	5½
Sal ammoniac, lump, firsts, delivered U. K., per ton	42 0 0
Sulphur, recovered, per ton	5 0 0
Sulphate of ammonia, f.o.b. Liverpool, per ton	14 0 0
Shellac, standard T. N., orange spots, per cwt	4 4 0
Tinore, 70 per cent, per ton	£123 to 125 0 0
Hydrochloric acid, per cwt	5 0
Zinc, Vielle Montague, f.o.b. Antwerp	30 12 6
Quicksilver, per bottle	9 0 0
Platinum, per oz	8 12 6

The following have risen since end of June, 1911:

Lead	£ 10 0
Hæmatite	4½
Scotch pig	7
Cleveland	7½
Rubber	7
Litharge	5 0
Petroleum	1⅛
Sulphate of ammonia	12 6
Quicksilver	12 6
Zinc V. M.	17 6

The following show a falling off in price:

Copper	£ 7 6
Tin	5 10 0
Copper sulphate	3 7 0
Carbolic acid	1
Tin ore	3 0 0

The stamp-tube mill combination for fine grinding, so common in South Africa, is being assailed as inefficient from a mechanical standpoint. Many engineers are inclining to the belief that other machines, particularly rolls and Chilean mills, will do the work with greater mechanical efficiency.

The principal use of fuller's earth in this country is in bleaching, clarifying or filtering fats, greases and oils. The common practice with mineral oils is to dry the earth carefully after it has been ground, and run it into long cylinders through which the crude black mineral oils are allowed to percolate very slowly. As a result the oil which comes out first is perfectly water white and much thinner than that which follows. The oil is allowed to continue percolating until maximum shade is reached. The earth is then purified.

SYNOPSIS.

Gold and Silver.

Cyaniding Antimonial Gold Ore.—An interesting experience in the treatment of a gold ore carrying some stibnite is detailed in the July, 1911, issue of the *Mining Magazine* (London), by Mr. H. T. Brett.

The Globe and Phoenix mine in Rhodesia produces a hard white quartz ore almost free from base minerals except for the presence of antimony as sulphide. This mineral occurs in average quantities of about 1 per cent, but in some places as much as 3 per cent is found. An analysis shows: SiO_2, 76.32 per cent; Sb, 0.73 per cent; Fe, 2.58; S, 0.91; CaO, 1.21; MgO, 6.44; As, Trace; Cu, Trace; Au, 1.6 oz. per ton; Ag. 6 oz. per ton.

More or less difficulty has been experienced in the treatment of this ore on account of the effect of the antimony. For example, in cynaiding sand and slime, with everything favorable for good extraction, the presence of only 1 per cent of stibnite makes an extraction of more than 20 per cent of the gold practically hopeless. Concentration is difficult on account of the friability of the antimony mineral.

The original plant erected for the treatment of the Globe and Phoenix ore consisted of 40 stamps, a sand-leaching plant and a slime decantation plant. When the antimony appeared in the ore, the extraction dropped to 30 per cent on slime and 40 per cent on sand. It was then discovered that by allowing the slime to weather for six months it was more amenable to cyanidation, and this course was then adopted. But the sand did not yield to such treatment and, moreover, rusty gold began to appear in the ore, thus complicating matters. Various attempts were made to overcome difficulties by regrinding the sand in pans, and by sorting out as much antimony as possible on sorting belts and jigs. The latter proved disappointing. All roasting of the ore gave satisfactory results, but the installation of a plant for this process would disturb existing conditions too seriously. Finally after much experimenting on a large scale a method of treatment was adopted which includes various features which were found valuable.

The ore is now crushed and washed in a trommel 10 ft. 6 in. long by 5 ft. 6 in. diameter at the large and 2 ft. 9 in. at the small end, perforated with 1½-in. holes for the first 4 ft. of its length and ¼-in. holes for the remainder. The trommel oversize is delivered onto a sorting belt from which massive stibnite is sorted in quantities ranging from ½ ton to 1 ton per day. After sorting, the oversize of the trommel is combined with the undersize and delivered to the mill bins.

Forty stamps of 1250 lb. weight crush the ore through 25-mesh screen. It then flows over amalgamated plates, the lower 3 ft. of which are covered with blankets to catch coarse rusty gold. This product is afterward amalgamated in a barrel. The tailing leaving the plates is classified into sand and slime. The sand is concentrated on vanners to remove as much pyrite and stibnite as possible. The concentrate is sent to the roasting plant while the sand tailing is reground in grinding and amalgamating pans and combined with the classifier slime. This fine pulp is then run over blankets where a fine concentrate is recovered, which is later treated in the roasting plant together with the coarse concentrate from the vanners. The fine tailing then flows to the sump of the tailing dump, whence it is pumped to a large spitzkasten for the separation of fine sand and slime.

The fine sand is collected for leaching in tanks which are filled to a depth of 6 ft. Each charge is subjected to from six to eight days actual treatment, but even then the extraction is unsatisfactory, and on ore carrying 1.5 oz. gold the tailings are too rich to throw away. It has been found that as soon as antimony on the sand charge exceeds 0.2 per cent the extraction drops rapidly.

The fine slime from the last spitzkasten is dewatered and

pumped to the storage dams. The clear water recovered from the slime is returned to the stamp mill. As soon as a dam accumulation has reached a depth of 6 ft. and contains about 10,000 tons dry slime, the stream is turned to another dam, and so on. After two months' time the dry accumulation is plowed and harrowed, and finally when the surface appears to be dry and thoroughly oxidized it is taken to the plant, mixed with cyanide solution in proportions of about 1½ parts solution to 1 of slime. This treatment of the surface of the dam is repeated until all the accumulated slime has been removed. Cyanidation of the slime is carried on in the decantation plant originally erected for this purpose, but a vacuum filter plant is under consideration. The extraction of the weathered slime is 75 to 80 per cent. The same material unweathered gives an extraction of only 30 per cent.

The pyrite and stibnite concentrates mentioned above are roasted in a Merton roaster, ground and amalgamated in pans, in which cyanide solution is used. It then flows over blankets where rusty gold is caught. Finally the pulp is agitated for 24 hours and filtered in a Dehne press. The residue is taken to the dump for oxidation and future treatment if required. The raw concentrate contains about 5 per cent Sb and 9 per cent S, while the roasted material carries about 3 per cent Sb and 0.72 per cent S, thus showing the difficulty of eliminating the Sb. The extraction varies from 85 to 90 per cent on a head value of from 2 to 15 oz. per ton. The following table shows the percentage recovery from the various materials treated:

Source.	Value per ton milled.	Per cent per ton milled.
Stamps and pans	$28.04	70.86
Concentrate	2.41	6.65
Sand	0.39	1.08
Slime	2.15	5.91
	$33.89	93.50

Slime Settlement.—Metallurgically speaking, the term slime applies to the finest portion of crushed ore, while sand is the coarsest. Various definitions have been proposed, but the foregoing statement is believed to cover the essentials as far as mill practice is concerned. The settling of slime is an important matter in the present practice of cyanidation, and various means have been adopted for accomplishing this end. In the *Monthly Journal* of the Chamber of Mines of Western Australia for May, 1911, Mr. B. L. GARDINER writes regarding the method adopted at the Sons of Gwalia mine.

The plant was originally equipped with nests of spitzkasten, but the settlement was inefficient and some means was sought of improving the efficiency by materially altering the apparatus then in use. The expedient hit upon resulted in practically increasing the settling area, and in delivering a thicker pulp than before. It consists simply of suspending in the pulp in an inclined position a number of sheets of corrugated iron. The idea is not altogether new, as it has been adopted in the United States and is one of the features embodied in the thickener of John E. Rothwell, made by the Colorado Iron Works Company.

The idea may be illustrated by filling a glass cylinder with slime and then noting the different rates of settlement when the cylinder is placed in a vertical position and at an angle of 45 degrees from the vertical. In experiments made by the author the rate in the vertical cylinder was 40 mm. per minute, while the rate in the cylinder set at an angle was 180 mm. per minute, or four and one-half times as great. The reason for this is obvious on close examination, for, on the upper side of the inclined cylinder a stream of clear water can be seen rising, while the particles of slime settle in a vertical direction from the inclined side of the cylinder.

The effect of the inclined sheets of corrugated iron in the spitzkasten is to reproduce many times the effect noted in the inclined glass cylinder. Wherever there is a surface facing downward the pulp must settle away from it, leaving the water to be forced up. The increased settling area obtained by this arrangement is stated by the author to be the total area of the sheets by the cosine of the angle they make with the vertical, or the projection of their area to the horizontal plane.

To test the efficiency of the inclined sheets, one section of spitzkasten was fitted with them, while three sections were allowed to remain in their original condition. The section fitted with the accelerating sheets showed a fall of slime at the rate of 0.168 in. per minute, while the three unaltered sections gave a fall of 0.107 in. per minute. The altered section gave a thickened pulp containing 41.1 per cent dry slime, while the three unaltered sections showed only 26.1 per cent.

The principle has its limitations, but a simple test with the glass cylinder will show what may be expected from any given pulp. The advantage of the device lies in the fact that it can be applied to existing spitzkasten.

Assay of Silver-Bearing Ores.—To test the effect of varying quantities of different reagents on the assay of silver ores CHARLES R. KEYES and D. F. RIDDELL made many experiments which are recorded in a paper published in the July, 1911, *Bulletin* of the A. I. M. E.

The ore analyzed as follows: Fe, 8.7 per cent; Pb, trace; Cu, 25.6; Zn, 6.2; Mn, 1.0; CaO, 15.0; S, 10. Assays were made to determine (1) how acidic a charge could be run; (2) how basic a charge could be run; (3) how large an amount of borax could be used in a charge; (4) how the normal charge for the kind of ore would behave, and (5) how the use of an excess of litharge would affect the results.

The conclusions deduced from the data given are: 1. A dead roast is not so satisfactory nor so accurate as a run with a large excess of niter or a scorification on a charge of 0.1 A. T. or more. 2. An excess of borax glass gives low silver results, especially in the presence of large amounts of sulphur, copper, iron or other matte-forming substances. The use of amounts approaching 1 A. T. must give low results. 3. An excessively acid flux fuses with great difficulty and yields poor buttons. In the presence of matte-forming substances the silver determination is low. 4. A very basic flux is open to all the objections that are raised to a very acid flux. Both give variant results most of the time. 5. Ores high in copper cannot be run by any crucible method without scorifying the lead buttons. 6. Direct scorification on high-grade copper ores does not check as well as the combination of crucible and scorification. 7. The normal or usual charge for coppery ores gives as high results as any method on this type of ore, but the checking is not as good as when an excess of litharge is used. 8. For ores high in copper, or carrying some zinc, a large excess of litharge in the flux improves the buttons and renders them easily handled. Furthermore, the time and temperature of fusion are decreased, and the buttons check well, with very close to actual valuations for the silver in the ore.

Cyanidation in Korea.—That improvement is the order of the day in all parts of the metallurgical world will be seen from the following description of present and proposed practice at the Unsan mines, Korea. The notes are abstracted from an article by G. C. EVANS in the June, 1911, issue of the *Mining & Engineering Review* (Australia).

The company has five mills, in all of which the same practice is followed. The ore is crushed by stamps, roughly classified in V-boxes for the purpose of getting some coarse sand to mix with the concentrate prior to cyaniding. The entire pulp with the exception of the sand mentioned is then concentrated on vanners. The vanner tailings run over canvas tables and then to waste. The concentrate is given an 18-day treatment in leaching tanks. Under this system the recovery is about 85 per cent. The vanner tailings average about 70 cents per ton and the recovery is 86 per cent; the concentrates are valued at about $22.30 per ton and the recovery is 84 per cent. The milling costs, exclusive of cyaniding, are 54 cents per ton, and the cyaniding costs $1.35.

A modern cyaniding plant is to be installed to treat the con-

centrate, together with an accumulation of old tailings amounting to 195,507 tons. The old tailings are valued at about $4.75 per ton. The new system will include concentration on Wilfley tables, grinding of the table concentrate in tube mills, agitation in Brown tanks, filtration in Dehne presses, and precipitation of the gold bearing solution in Merrill presses with zinc dust. This plan, it will be seen, embodies most of the latest ideas in cyanidation, and is expected to prove profitable to the company.

Mercury.

Wet Process for Cinnabar.—Little work has been done in improving methods of treatment of mercury ores, except in the line of furnaces for volatiling the metal. Low-grade ores have hitherto been without a suitable method of treatment. Mr. C. A. Mulholland has devised a wet process which he believes is practicable in view of the improvements made in the handling of large quantities of wet material. The details of the process are given in the *Aust. Min. Standard* for June 8, 1911.

The process consists in dissolving from the ore the very finely divided cinnabar, and reprecipitating it in a concentrated form ready for distillation in a comparatively small apparatus, in which efficient condensation can be carried out, and the escape of fumes prevented. It has long been known that cinnabar is soluble in alkaline sulphides containing free alkaline hydrates, and may be reprecipitated by the addition of suitable chemicals or electrolized for the recovery of the metal.

According to the author's experiments, the ore could be crushed to ¼ in., and then slimed with the solvent in a tube mill, or other convenient machine. Agitators could be used following the tube, if found necessary to complete solution, and these agitators should be heated by a steam jet. The solvent solution should contain at least one molecule each of alkaline sulphide and hydrate to each molecule of mercuric sulphide, but in practice an excess would be used. When solution is complete the pulp is settled and the clear solution decanted for precipitation. The residue is washed with weak solution and the washings used for extracting fresh ore.

The pregnant solution is now precipitated with zinc hydrate, when zinc and mercury sulphides will fall out of solution and are separated by filter pressing. The mixed precipitate is treated with dilute sulphuric acid and steam which will dissolve the zinc as sulphate, leaving a mixture of mercury sulphide and mercury. This is mixed with lime and iron and retorted in the usual way.

In treating the mixed zinc and mercury precipitate with acid, sulphuretted hydrogen is evolved, which is passed into the filter press effluent to regenerate alkaline sulphide for further use.

From small scale experiments the author has calculated the probable cost of operation on the basis of treating 100-ton lots of ore of varying grades, from 0.05 per cent to 1.0 per cent mercury. For example, on ore of 0.3 per cent grade, worth about $4 per ton, he believes that an efficiency of 80 per cent could be obtained, or a value of $3.20 per ton, at a cost of about $1.85.

Lead and Copper.

Concentrating Table.—In the July, 1911, *Bulletin* of the A. I. M. E. Mr. S. A. Krom presents some considerations on the proper form of table best adapted to the separation of minerals from gangue by methods depending on difference of specific gravity. He rather deprecates the use of riffles, and even states that in the near future they may be entirely eliminated from tables. He believes that riffles rather upset and retard the process of stratification which they are supposed to promote. He argues further that no concentrate can be delivered from a riffled table until the deck has become bedded, and that this bed must be maintained by very carefully adjusted feed of pulp and rate of discharge, or else the efficiency of the table is lowered. Perhaps Mr. Krom is too severe on the riffled table, and as he is engaged mainly in making out a case for an unriffled table, his criticisms will be taken properly modified by that fact.

The author describes a table invented by U. S. James, in which certain radical departures have been taken from the ordinary table. Stratification is obtained in a settling basin into which the table feed runs, and which is formed by two planes of different inclinations to each other, meeting in a line diagonal to the motion of the table. From this basin the concentrate is discharged over the deck of the table by reason of the table motion, and the tailings flow to the discharge. It is claimed that the settling basin does not require bedding and that it is difficult to overfeed." Adjustments in the eccentric motion of the table are made by an adjustable pin, which gives a wide range of different movements.

Copper Smelting in Michigan.—The simplicity of operations in smelting concentrates of native copper in Michigan has given rise to the idea that there is no metallurgy in Lake Superior smelting. In the School of Mines *Quarterly* of Columbia University for July, 1911, Mr. Henry D. Conant gives a descriptive statement of smelting operations in Michigan in which he shows that despite the simplicity of the operation there is opportunity for the engineer to show his ability in the effective use of fuel, the efficiency of labor, the production of clean slags, and the proper refining of the impure metal.

Lake Superior copper is obtained by concentrating the conglomerate or amygdaloid rock, which contains from 1 to 3 per cent copper, and obtaining what is locally called "mineral," containing 60 to 80 per cent copper. This concentrate is smelted in reverberatory furnaces without any fluxes, the slag skimmed off and further treated in cupola furnaces, and the reverberatory copper refined in either the same furnace or a second reverberatory into which it is tapped. In the first smelting operation the impurities are largely oxidized and slagged off. Some of the copper is at the same time oxidized, forming copper suboxide, which must be reduced again. This is accomplished by poling with green wood, after which the copper is cast into forms suitable for the various needs of commerce. Both the oxidizing and reducing operations are governed by the experience of skilled workmen, who take samples from time to time, and determine the end of the reaction by the appearance of the sample.

The reverberatory slags are either acid or basic, according to whether they are produced from amygdaloid or conglomerate mineral respectively. These are properly fluxed with lime, iron and silica, and reduced in cupolas, under a blast of from 3 to 4 oz.

The author describes in a general way the four plants now operating in Michigan, and gives some interesting historical data on the changes that have occurred in practice in later years.

Electrochemistry.

Fixation of Atmospheric Nitrogen with Intermittent Load.—The problem of utilizing the electrical energy of central stations at times of low load has always been attractive for electrochemists on account of low power costs, while electrochemical processes have been attractive for central station engineers to equalize their load curve. The question whether electrochemical processes can be operated intermittently with a financial profit has therefore often been discussed, but there is very little exact information available from commercial working. A note in a recent number of the London *Times* Engineering Supplement is therefore of considerable interest. As abstracted in London *Electrician* of August 4, it states that "for some time past experiments have been in progress at Legnano, Italy, with the object of utilizing the excess power generated during the night and certain hours of the day by a large hydro-electric station. The aim was to produce nitric acid and nitrates from the nitrogen of the air, and it is considered that the problem has now been satisfactorily solved. Some months ago the employment of 4000 kw. for the production of nitric acid was initiated, and now preparations are being made for the use of 9000 kw. more, energy being supplied by the Societa Lom-

barda per Distribuzione dell' Energia Elettrica at a pressure of 50,000 volts."

Electrolysis Destruction of Water and Gas Pipes Due to the Stray Currents from the Rails of Tramway Systems.—Considerable trouble was experienced at Pittsburg, Pa., with stray currents from the rails of the tramway system attacking underground structures of the water system. A very full discussion of these troubles is given in a paper by E. E. Lanpher and L. B. Smith before the Engineers' Society of Western Pennsylvania and published in the *Proceedings* of the Society for July. The authors conclude that the only certain method of preventing electrolysis is the installation of the double-trolley system and the most successful method of mitigating electrolysis is the installation of the "return pipe feeder system." The latter system was introduced in Pittsburg, but this solution was criticised in the discussion by various speakers especially by Prof. A. F. Ganz who favors the radial insulated return feeder system; the return feeders being designed to drain the rails of current at many selected points, leaving these points at the same potential, thus greatly reducing the rail drops and minimizing stray currents on underground structures. In the reply of the authors the unusual conditions prevailing in Pittsburg are pointed out but they do not claim that under different conditions the results would have been equally satisfactory, in fact the conditions as they existed in Pittsburg were almost ideal for the installation of the return pipe feeder system.

With respect to **local electrolysis** brought about by impurities in steel pipes some interesting remarks were made in the discussion by Mr. F. N. Speller of the National Tube Company. To correct a misapprehension that is very widespread as to corrosion being often due to irregularity in the composition, he said that they made a great many experiments along that line which indicated that steel or iron has to be much below the average before the irregularity incident to commercial material became an important influence in corrosion. As a matter of fact, the difference in potential between the greatest irregularity which is ordinarily found due to segregation does not begin to compare with the differences due to the accidental presence of foreign material such as mill scale on the surface of the metal itself. If all the scale could be removed from the pipe, one would probably have much less trouble with the corrosion of underground piping.

Primary Battery.—In our August issue, page 401, a paper by Mr. W. R. COOPER on the Benkö cell was given. In connection with this it is interesting to note a recent exhibition by Mr. Cooper of the Benkö cell at a meeting of the Physical Society in London. Several types of this cell were exhibited. The battery depends essentially upon the use of a porous carbon electrode in the form of a pot (open at the top) which is surrounded by a closed lead shell. The elecrolyte (a bichromate solution) is fed into this shell under a suitable head to give a flow of ½ liter or more per hour per cell. It passes through the carbon to the zinc, where it is removed by an overflow pipe. Polarization is thus reduced to a minimum and remains constant for a given current, provided the cell is not overloaded. The size of cell referred to is 9 ¼ in. high overall, 6 ¼ in. long and 1 ¾ in. wide; it weighs about 10 lb., excluding the tank for electrolyte, and will give a steady current (depending upon the strength of the electrolyte and rate of flow) up to, say, 25 amps. at 1.5 volts. The internal resistance is below 0.01 ohm. There is no "mess" and no difficulty in keeping the apparatus in order. The cell is therefore very suitable for laboratory use or work where the cost is not of prime importance. Various types are being made according to the work required, and these include cells which depend for their action upon diffusion as distinct from a steady flow.

In the discussion, which followed, G. L. Addenbrooke asked whether the contact between the lead and the carbon remained satisfactory over a long time. The author replied in the affirmative so far as his experience (extending over 15 months) had gone. Prof. Callendar inquired as to the maximum number of watts that could be taken from the battery continuously, and whether, weight for weight, more power could be taken from it than from a lead storage battery. The author replying to the first part of the question said about 25 to 40 watts, depending on the solution, and stated that, excluding the reservoir, 2.4 to 4 watts per pound could be taken from the battery as compared with ½ watt (8-hour rate) up to 2½ watts (automobile cells at 5-hour rate) for lead cells.

Mr. W. Duddell asked how the battery would compare with lead cells for giving a steady current of the order of 5 amperes to 1 part in 5,000 or so. Lead cells could only be relied on to keep steady to about one part in 1,000. The author said he could not reply definitely, but he would expect a battery result from the Benkö battery bulk for bulk. Replying to other speakers the author stated that the emf. of a diffusion cell began to fall after 10 hours' use, and that the electrochemical efficiency of the zinc in a Benkö cell was about the same as in an ordinary primary cell.

General Physics and Chemistry.

Chemical Effects of Canal Rays.—It has been the source of a long series of discussions as to whether chemical elements in the form of canal rays will react with other substances with a greater amount of kinetic energy than is possible if brought to reaction otherwise at highest attainable temperatures. V. KOHLSCHUETTER publishes another interesting contribution in the May 15 number of the *Zeitschrift fuer Elektrochemie*. Canal rays were permitted to act upon various substances, the tubes being filled at the start with small amounts (0.01 to 0.04 mm) of either hydrogen, nitrogen, oxygen, argon, carbon dioxide, sulphur dioxide, or hydrogen sulphide. The author found that the products of reaction were identical with those obtained whenever the substance, which had been acted upon by the canal rays, was used as cathode instead. It was impossible to show absolutely that singular chemical reactions could be brought about by the action of canal rays. The chemical reaction observed seemed to be secondary to a primary effect of the rays, possibly a bombardment or a sudden increase in temperature. At all events this primary effect, whatever it may be, is altogether independent of the chemical nature of the gas in the tube. The effect was stronger in the presence of the heavier gases. Compounds such as oxide of copper and sulphide of silver were reduced more readily in nitrogen than in hydrogen. Chloride of mercury was reduced to the metal even though the gas in the tube was oxygen. Many other examples are cited.

Electrical Resistance of Pure Metals at Very Low Temperatures.—For several years Prof. H. KAMERLINGH ONNES, of Holland, has carried out exceedingly interesting experiments on the enormous decrease of electrical resistivity which pure metals undergo at very low temperatures. He has now perfected his experimental facilities for the use of liquid helium (which by the way was found to be an excellent insulator) and the results of his determinations of the resistivity of pure gold and pure mercury at liquid helium temperatures are given in a recent issue of the *Proceedings* of the Koninklijke Akademie van Wetenschappen, Amsterdam, of which a translation in abstract appears in the London *Electrician* of August 4. The author's recent experiments with gold have greatly strengthened his former conclusion that the resistance of pure gold vanishes at helium temperatures. Even more interesting are the experiments with mercury. The measurements were made with the differential galvanometer by the method of overlapping shunts (Kohlrausch) and also by the method of the measurement

of current strength and of potential difference. The value of the mercury resistance used was 172.7 ohms in the liquid condition at 0°c; extrapolation from the melting point to 0 C by means of the temperature coefficient of solid mercury gives a resistance corresponding to this of 39.7 ohms in the solid state. At 43°K (4.3 degrees on the krypton themometer) this had sunk to 0.084 ohms, that is, 0.0021 times the resistance which the solid mercury would have at 0°C. At 3 K. the resistance was found to have fallen below 3 v 10^{-7} ohm, that is, to one-ten-millionth of the value which it would have at 0°C. As the temperature sank further to 1.5°K, this value remained the upper limit of the resistance. The next step was to look for the point at which the resistance first becomes measurable as the temperature is raised. The temperature of this point was found to be slightly more than 4.2°K, at which the resistance was found to be 230 microhms or one-hundred-thousandth of the resistance (solid) at 0°C.

Recent Metallurgical Patents.

Lead and Copper.

Hydrocarbon Blast Furnace.—The use of oil as fuel for producing the necessary heat in a blast furnace was the subject of considerable experimenting by the late John W. Nesmith, of the Colorado Iron Works Company, of Denver. Recently two patents have been issued, assigned to the company mentioned, based on the experimental work inaugurated by the original inventor.

In Fig. 1 is shown a vertical section of a furnace adapted to carry out the invention. The vertical stack *A* is connected

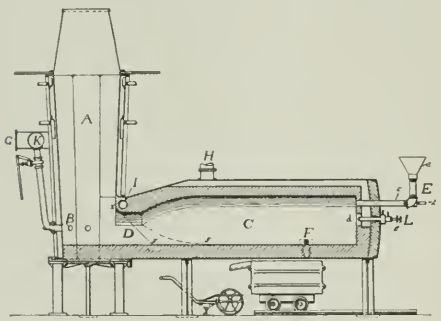


FIG. 1.—HYDROCARBON BLAST FURNACE.

by a passage *D* with a combustion chamber *C*. Tuyeres are located at *B*, and a bustle pipe *K* is connected with an air supply *G*. Air for the combustion chamber is admitted at *H* and passes along the top of the combustion chamber to the inlet shown at *D*. *L* represents an oil burner, *e* being the controlling valve. Provision is made to tap slag and matte at *F*. In order to prevent the burning out of the arch at the connection between stack and combustion chamber a water pipe *I* is located at this point. A hopper with device for forcing fine ore or concentrate into the combustion chamber is located at *E*.

In operation, the regular charge of coarse ore is fed to the stack in the usual manner, and the base of the column fills the opening *D*, the ore taking the natural slope of such material, say 45 degrees. Combustion is maintained in the combustion chamber by means of oil and air admitted at *d*, and the charge gradually smelts to a fluid condition, running onto the hearth of the chamber. The tuyeres are located below the top line of the combustion chamber, as it was found in practice that when located higher on the shaft there was tendency to form a secondary smelting zone in the shaft, causing some of the metals to liquate and flow to the hearth, leaving a skeleton hanging in the shaft. The hot gases from the combustion chamber pass upward through the charge in the shaft and escape.

The combustion chamber is relatively large, as it was found in actual operation that when using petroleum as fuel the volume of space in the chamber should be not less than 35 cu. ft. for each pound of combustible burned per minute.

The provision for introducing fine ore, concentrates or flue dust directly into the combustion chamber offers advantages in smelting such materials, which are always difficult to handle economically. Such fine material is introduced into this furnace from the hopper *E* and accessory devices, by means of a blast of air. Being introduced directly into the combustion chamber they are immediately melted or burned and combined with the molten mass on the hearth. If any part of them is not so smelted it is readily caught in the fusing and semi-fused mass in the stack.

The furnace is thus adapted to the continuous treatment of coarse ore in the usual manner, and the continuous or intermittent treatment of fine material. By this method it is possible to smelt current productions of flue dust, as well as moderate quantities of fine ore and concentrates. Furthermore, in the case of sulphide concentrates, the sulphur contained is utilized as fuel and reduces the quantity of fuel oil required. (905,365-6, June 13, 1911.)

Ore Roasting Furnace.

An annular mechanical roasting furnace adapted for the treatment of pyrite or blende is the basis of a patent of Curt Pfaul, of Blasewitz, Dresden, Germany. A representation of the furnace is given in vertical section in Fig. 2. It has a helical bed with rabbles *b* traveling parallel to it, they being

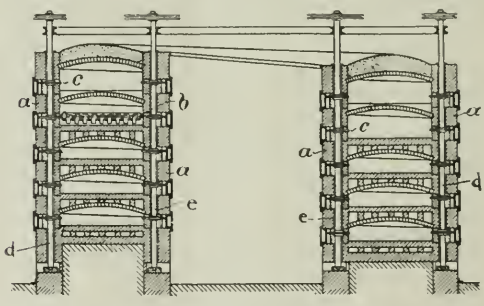


FIG. 2.—ROASTING FURNACE.

carried on rollers *c* bedded in the walls of the furnace. The vertical shafts *d* and *e* upon which the rollers are mounted are arranged at intervals in the circular walls of the furnace. The frame *b* is simultaneously supported by three rollers *c*, and by turning the shafts in either direction the frame carrying the rabbles may be made to move upward or downward in the furnace. The scheme avoids the production of the large quantities of dust formed in the multiple hearth furnaces where the rabbled material falls from one hearth to the other. In this furnace the rabbling is accomplished as the frame moves downward through the furnace. It is then returned to the top by reversing the movement of the shafts, and the operation is repeated (989,868, April 18, 1911).

Fume Condensing Apparatus.

The condensation of the solid and soluble parts of furnace gas by means of water and steam sprays continues to attract the attention of inventors despite the fact that apparatus of this kind has not been usefully applied on a large scale. Paul Kestner, of Lille, France, has patented apparatus for this purpose. He recognizes the fact that dry dust has characteristic repulsion for water, but that if it be preliminarily

dampened it may be readily absorbed in water. He therefore bases his process on first spraying the gas with steam and then cooling it, causing the steam to condense on the solid particles in the fume and dampen them. By subsequently spraying the gas with water he claims that the solids are easily removed. The apparatus is represented in Fig. 3, in which *A* is

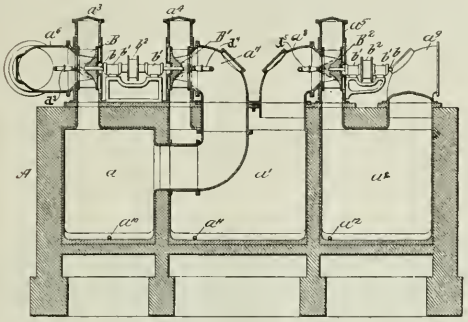


FIG. 3.—KESTNER FUME CONDENSING APPARATUS.

a chamber divided into three compartments a, a^1 and a^2. Above these are casings a^5 a^4, a^5 which are connected with their corresponding compartments and in which are located fans, B, B^1, B^2 operated by means of pulleys b^2. Spray pipes are arranged to discharge into the face of the fans. As the furnace gases flow into the receptacle at a^6 they are thoroughly mingled

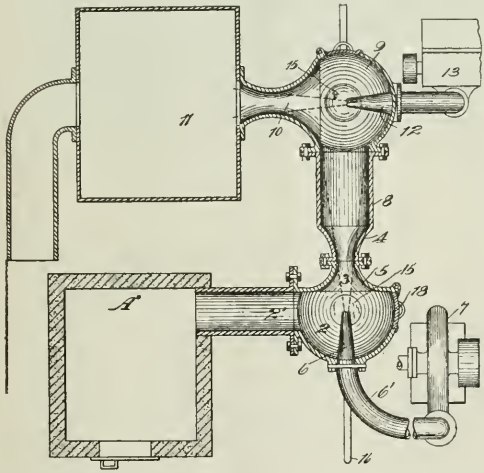


FIG. 4.—FUME CONDENSING APPARATUS.

with steam flowing from the jet against the fan B. They then flow into the compartment a, where they cool and become wetted with the condensed steam. They then flow successfully through the other compartments and fans, being sprayed with water in the latter. The cleaned gases pass out by way of a^5, and the water with the collected fume is accumulated in the chambers a, a^1, a^2, from which it flows to other receptacles by way of pipes a^{10}, a^{11}, a^{12} (991,157, May 2, 1911).

Another apparatus for condensing fume in furnace gas from smelters, garbage crematories, etc., is patented by GEORGE C. RICHARDS, of Berkeley, Cal. He first subjects the gases to a jet of water and later to a blast of air. The first jet wets the solid particles and the air blast cools the gases and aids condensation.

The apparatus is shown in plan in Fig. 4. The source of gas is represented by A, from which it passes through the passage $2'$ to a globular head 2, having a contracted discharge at 3. The water jet is delivered from the nozzle 6. The wetted gas then flows to another globular head 9 where it receives a blast of air from the nozzle 12, which discharges through a contracted orifice 10 into a chamber 11, where the fume is collected. The effect of the angular flow of gas is to break it up and avoid the formation of strata. The accumulations of tarry or other matter in the globular heads may be removed by suitable gates placed in the bottom of each head. The vacuum creating effect of the air blast assists in the force of the induced draft through the apparatus, but its main effect is its cooling action which insures precipitation and condensation (990,009, April 18, 1911).

Electrolytic Production of Oxygen and Hydrogen and Welding and Cutting by the Oxyhydrogen Flame.

The simplest and cheapest way of producing oxygen and hydrogen in equivalent quantities is by electrolysis of water. The process is used in numerous plants in Europe, and the oxygen and hydrogen gases thus produced are widely employed there for welding and cutting by the oxyhydrogen flame and for other purposes. The reason why this country has been behind Europe in this development is probably the fact that the oxy-acetylene flame was early introduced here and was believed, on account of its high temperature, to be superior to the oxyhydrogen flame. It has been since found, however, that both have their distinct fields of applications, and with the increasing demand for hydrogen alone (for instance, in connection with aeronautics) it is but natural that the electrolytic production of hydrogen and oxygen on a large scale has finally found its way to this country.

The International Oxygen Company, of 68 Nassau Street, New York City, has erected a plant at Waverly Park, Newark, N. J., which is one of the most completely equipped of its kind in the world, producing some 10,000 cu. ft. of oxygen daily. This company either furnishes to its customers hydrogen and oxygen gases in cylinders or instals their electrolytic apparatus at the consumers' premises to suit their requirements, so that they produce their own hydrogen and oxygen whenever needed.

The electrolytic cell used is very simple, an outside view being given in Fig. 1 and a diagram in Fig. 2. The iron tank A of Fig. 2 serves as the cathode, being connected to the negative pole of the electric supply circuit. From the cover B of this tank are suspended the hydraulic joint C and a perforated tank D which serves as the anode, being connected to the position pole of the supply circuit. It is made of a specially selected low-carbon steel, to prevent the formation of spongey rust.

By means of an asbestos sack E suspended from the hydraulic joint C two separate compartments are formed, one being the anode compartment and the other the cathode compartment. The cell is thus a true diaphragm cell.

Through the opening F in the cover a solution of caustic alkali in distilled water is poured into the hydraulic joint C and distributed in the two compartments.

The whole cell is placed on insulating supports H of porcelain.

When current is turned on, electrolysis results and the total reaction is the decomposition of water into oxygen gas and hydrogen gas. The oxygen gas is set free at the anode D and the hydrogen gas at the cathode A. The two gases, thus produced separately from each other, are prevented from mixing by the asbestos sack diaphragm.

The hydraulic joint C is a special feature of the cell, as it adjusts its inner working. Another feature is that the oxygen

and hydrogen gases evolved do not pass directly from their compartments to the off-take pipes, but first bubble through water contained in the two lanterns on top of the cell, as shown in Figs. 1 and 2. They enable the operator to see at a glance how the cell is working.

The purity of the gases produced is very high. A sample of hydrogen, produced by this electrolyzer analyzed by the Conservatoire National des Arts et Metier in Paris, showed 99.70 per cent hydrogen, the fraction of the impurities being so small that they were not looked for.

All that is required for the operation of the cell is the daily addition of some distilled water to make up for the quantity decomposed into hydrogen and oxygen. About 1 gal. of water is required to produce 100 cu. ft. of oxygen and 200 cu. ft. of hydrogen. Otherwise the operation of the cells needs no further attention, manipulation or supervision by an expert. As to the electrical energy requirements a joint test made in November, 1910, by the Laboratoire Centrale de l'Electricite and the Conservatoire National des Arts et Metiers with two unit cells of this type of electrolytic cell showed that the production of 1 cu. ft. of oxygen and 2 cu. ft. of hydrogen requires 0.2797

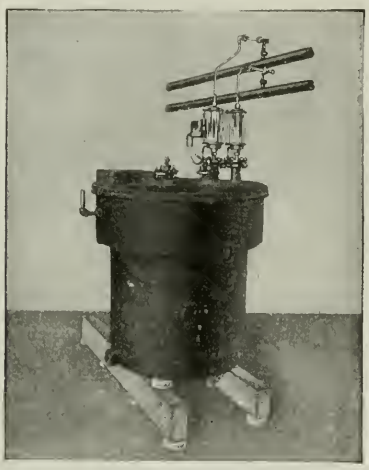


FIG. 1.—VIEW OF CELL FOR ELECTROLYTIC PRODUCTION OF OXYGEN AND HYDROGEN.

kw-hours. Reversely 1 kw-hour produces, 3.54 cu. ft. of oxygen and 7 cu. ft. of hydrogen.

Each unit cell requires a little above 2 volts and from 300 to 400 amp. A current of 350 amp produces about 65 cu. ft. of oxygen and 130 cu. ft. of hydrogen per day.

A plant made up of such unit cells is very flexible, since the units can be arranged in single file and double file, also in many rows and tiers, or in any way to suit the given conditions. Fig. 3 shows three double rows of cells, as installed at the Waverly Park plant of the International Oxygen Company.

The main item of the cost of operation is the cost for the electric current. In New York City the wholesale rates vary from 5 to 3 cents per kw-hour, according to the size of the plant; since 1 kw-hour produces about 3 cu. ft. of oxygen and 6 cu. ft. of hydrogen, the electric power cost for producing 1000 cu. ft. of oxygen and at the same time 2000 cu. ft. of hydrogen would be between $16.70 and $10. However, in large manufacturing plants which have their own power houses the cost of energy is much less; thus it is known that at the South Chicago works of the U. S. Steel Corporation the electrical energy supplied to the electric furnace is charged at the rate of half a cent per kw-hour. At this rate the electric power cost for producing 1000 cu. ft. of oxygen and 2000 cu. ft. of hydrogen would be only $1.67.

Besides the electric power cost there are the cost for attendance which is very small as already mentioned, and the cost for the distilled water which must be added to the cells to make up for the decomposed quantities of water. The latter cost may be calculated from the fact that 10 gal. of distilled water are required to produce 1000 cu. ft. of oxygen and 2000 cu. ft. of hydrogen.

As to the relative advantages of the oxyhydrogen and the oxy-acetylene flame for welding, the lower temperature of the oxyhydrogen flame and its easy regulation are of importance. They render the oxyhydrogen flame superior for thin sheets, especially in the form of kitchen materials and liquid containers that are to be enameled, also for welding aluminium, copper, bronze, and like metals. On the other hand, the oxy-acetylene flame is specially suitable for welding heavy pieces. (A most noteworthy example of heavy oxy-acetylene welding may be mentioned here, namely the recent welding of joints on the pipe line of the Central Colorado Power Company. On the part of the pipe line leading from the reservoir at the power house in Boulder Canyon down to the power house the joints were found leaking, and if the pipe line was to be saved it became

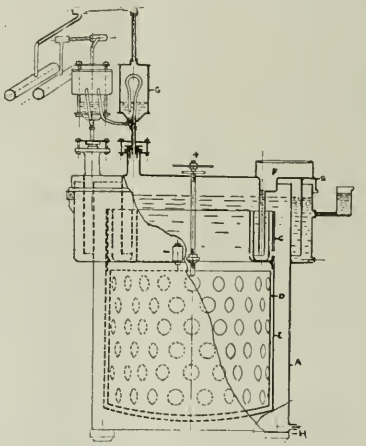


FIG. 2—DIAGRAM OF CELL FOR ELECTROLYTIC PRODUCTION OF OXYGEN AND HYDROGEN.

necessary to weld the joints. This has been carried out by the oxy-acetylene process. It is probably the biggest job of its kind every done and the difficulties were very great, as the pipe line is carried down in a very narrow tunnel and the heat in welding in such a narrow space was intense. Each complete joint took the time of two men for about a day and a half, the method used being quite analogous to that of lead burning. The Central Colorado Power Company must be congratulated on the successful outcome of this exceeding interesting piece of work. The welded pipe line is now giving absolute satisfaction.)

As to oxy-cutting, the following comparison is given by the International Oxygen Company. In oxy-cutting, oxygen and acetylene will cut on the average from 4 to 6 in., whereas with oxygen and hydrogen cuts up to 24 in. have been obtained. The cuts with oxygen and hydrogen are said to be usually smoother and more quickly made than with oxygen and acetylene.

In oxy-cutting the metal is first heated either by the oxyhydrogen or the oxy-acetylene flame, then the current of hydrogen or acetylene is turned off, and a jet of oxygen is blown upon the molten or melting metal. The cut which results is due to the formation of oxide iron by burning the metal in the

jet of oxygen. This oxide remains outside of the molten steel or iron, and as it is very light, it is blown away by the oxygen under pressure, and thus a cut is produced.

Now it is an important fact that if the oxygen used should contain any appreciable portion of inert gases they will prevent the oxygen from thoroughly oxidizing the steel or iron and will thus render good or deep cutting difficult. The International Oxygen Company attributes the difficulties of oxy-acetylene cutting to the fact that when oxygen combines with acetylene much carbon dioxide is formed which as an inert gas hinders the effectiveness of the oxygen.

The great effect which impurities in oxygen have on the results in cutting were well brought out in a recent paper of J. M. Morehead, of Chicago, before the New York Railroad Club. Numerous tests were made on cutting a mild steel plate ¾ in. thick, the total length of cut being in all cases about 67½ in. In the different experiments oxygen of different purity was

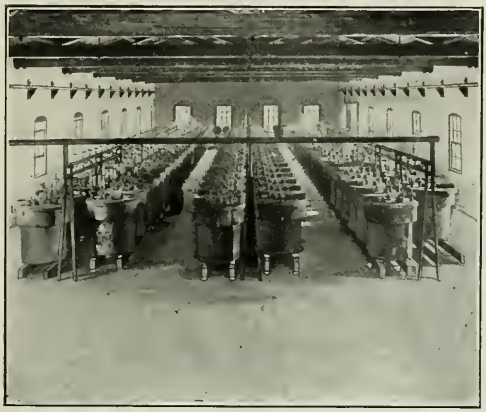


FIG. 3.—CELL ROOM OF INTERNATIONAL OXYGEN COMPANY.

used. We give here only the results of the tests with oxygen of 99.3 per cent, 96.4 per cent and 83.3 per cent purity.

Purity of oxygen, per cent.....	99.3	96.4	83.3
Cutting time in seconds	272	360	855
Oxygen used in cu. ft.	7.5	11.3	18.9

The cut with the 99.3 per cent oxygen was very good, like a hot saw cut. The cut with the 96.4 per cent oxygen was ragged and cindery. The cut with the 88.3 per cent of oxygen was very rough and slaggy, the metal not being properly cut through. The tests show clearly the very appreciable increase in cutting time and quantity of oxygen required, the less the purity of the oxygen. From the tests it would appear that with a percentage of about 80 per cent it is not possible to cut a ¾-in. plate.

An interesting simple application of oxy-cutting is the removal of rivets by the oxy-hydrogen process. An oxy-hydrogen welding torch is used, the rivet is heated until red, the hydrogen is then shut off, and the oxygen burns the rivet away. The meal about the rivet hole remains untouched and the hole is as clean and round as it was before the rivet was put in.

The Super-Calender is the title of a new home organ published monthly by the Pusey & Jones Company, of Wilmington, Delaware. Its circulation is limited to paper mill superintendents. The main part will be devoted to questions and answers relating to problems of paper manufacture of general intrest. The first issue contains an interesting picture of an early Chinese paper maker and a discussion of two interesting problems. The mechanical make-up is of very high class. Editorially it is of decided originality.

Lining Alloys.

The Lumen Bearing Company, brass founders, of Buffalo, N. Y., and of Toronto, Ont., has sent us an interesting little booklet, entitled "Progress in Lining Alloys," the author of which is the manager of their babbitt department, Mr. Alfred H. Greenburg.

It is emphasized that the term "genuine babbitt" has a rather indefinite meaning as to chemical composition. If it ever meant anything it was simply this: That the preponderant constituent was tin, and that its two other constituents were antimony and copper. Until recent years the term generally implied that the composition was free from lead. This, however, is no longer the case, since the low cost of antimonial lead as a by-product for the last fifteen years and the constant increase in the price of tin have weighed so heavily on the manufacturer of "genuine babbitt" that to-day the term no longer positively excludes lead from its composition. (The composition mentioned in Mr. Isaac Babbitt's first patent in 1839 was 50 parts tin, 5 antimony, and 1 copper. His favorite composition which he later sold to Mr. Phillips, was 10 parts tin, 1 antimony and 1 copper.)

There is no universal bearing composition that is best for all requirements. Hence bearing metal should be specified with great care according to requirements.

The heating of bearings is the principal cause of annoyance, and in cases where the metal punishment is so severe that heating cannot be avoided, a metal of high melting point should be selected. The efficiency of the alloy, therefore, depends upon the quality of the wearing surface that can be produced and maintained under service. A properly selected metal carefully applied, both as to design and workmanship, produces a bearing which, with proper lubrication, has no metallic contact while running. That is, the journal and its bearing are separated from each other by a film of oil which is maintained in operation. As soon as the movement of the journal is stopped, the film of oil is gradually squeezed out and the metallic surfaces are brought into contact. Therefore in selecting the metals for a bearing they should be sufficiently dissimilar so that when starting the machine there will be no danger of scoring the shaft until the oil film shall have been restored.

Aside from the chemical composition, even more important is the method of uniting the different constituents. In general, the metals should be melted in an iron vessel and kept covered as much as possible in order to prevent excessive oxidation. They should be heated considerably above their melting point before using, but must not be kept in a molten state at a high temperature longer than necessary. Overheating should be carefully avoided, and a good rule for general practice is to heat the molten babbitt to a point where it chars a pine stick, at which temperature it casts perfectly.

Further, with respect to analysis. Where the service conditions are severe, owing to great pressure, a metal having considerable compressive strength is necessary regardless of what the speed may be, and this condition would require a relatively high percentage of tin. Where there is high speed and the pressure light and moderate, a metal having a fairly high percentage of lead may be used. In the same manner with intermediate conditions between pressure and speed, correspondingly intermediate composition can be selected.

The surroundings of a bearing and the amount of care and attention which can be given to it must also be taken into consideration. A bearing that is lubricated at long intervals or with a poor grade of lubricant requires a higher grade of metal than that which would be required under more favorable conditions.

The gist of the matter is that while there is nothing very difficult in making babbitt suitable to any clearly specified kind of service, yet it requires an experienced metallurgist.

The little booklet of the Lumen Bearing Company is carefully prepared and well worth reading.

Titanium Rails.

The Titanium Alloy Manufacturing Company, of 1225 Oliver Building, Pittsburg, Pa., have just issued a handsomely illustrated pamphlet, entitled "Facts, Figures and Illustrations Proving That Titanium Has Solved the Rail Problem." It gives very interesting figures on the rapid increase of the use of titanium-treated rails in the last few years.

Three years ago titanium-treated rails were merely being tried on two or three roads, while now they are in service on more than fifty steam railways and on the more important trolley traction lines. Over 40,000 tons of titanium rails are now in actual use. The production of titanium rails in 1910 was 195,940 tons.

The beneficial effect of titanium is not that of an alloying element proper, but the titanium acts simply as a purifier. It is a powerful deoxidizer and removes part, if not all, of the nitrides as well. It prevents brittleness, hence the carbon content of the rail may be materially increased and should be increased if the full benefit of the titanium treatment be desired. The extra cost of the titanium is only about $2 per gross ton of finished rails.

The pamphlet contains interesting illustrations of the structure of titanium-treated and untreated rails, and illustrations with descriptions on the use of titanium rails on many roads in this country.

The following is a test of open-hearth 85-pound rail made recently by the Lackawanna Steel Company for the Delaware & Hudson Co. This steel was treated with 0.15 percent titanium alloy and tested by the National Tube Co. The elastic limit was 100,800 lb. per sq. in., the ultimate strength 147,200 lb. per sq. in., the elongation 12.50 percent, the reduction 16.29 percent.

The analysis was 0.86 C, 0.132 Si, 0.022 P, 0.032 S, 0.88 Mn, 0.065 Ti.

Manufacture of Ferro-Alloys in England.

The National Smelting Company, Ltd., has recently been formed with a capital of £200,000, which, when in full working order, will be practically the first of its kind in England. It has been formed chiefly for acquiring as a going concern the business and assets of Electric Furnace & Smelters, Limited, a company which was incorporated as a private company in 1910 for the purpose of making experiments in the smelting of ferro-alloys and various ores, and whose offices are at 110 Cannon Street, London. It will also acquire other electric furnace patents.

It is reported that the experiments made by Electric Furnaces, Limited, in the making of ferro-tungsten and other ferro-alloys were highly successfully, yielding a metal free from obnoxious impurities at a low cost For the purpose of demonstrating the success accomplished and ascertained by the experiments, a small factory, with plant capable of turning out about 12 tons of tungsten metal per month, was erected at Luton, and is in active daily operation. Adjoining the factory are the works of the Luton Iron Foundry, a business carried on for some years past by Mr. Archibald Finlay Maclaren, the managing director of Electric Furnaces. The benefit of a contract between Mr. Maclaren and Electric Furnaces, whereby the latter acquire an option to take over the land, business and plant of the Luton Iron Company, has been transferred to this company.

Large quantities of tungsten and other alloys are required by British steel manufacturers, over 900 tons of ferro-tungsten being used yearly in Sheffield alone, while the consumption of other high-priced alloys, such as ferro-vanadium, is at least as great. In the past these alloys have been imported to England from other countries. It is intended that the company shall, at its works in Luton, produce at least 1000 tons of ferro-tungsten and other ferro-alloys annually. Arrangements have been made whereby the company will be able to obtain a supply of electric power at a very low cost.

During the year 1910 an experimental electric furnace was erected at St. Ives, Cornwall, and about 20 tons of tin were smelted there. The result was highly satisfactory and proved the profitable nature of the process. (See the extensive article by Mr. Harden, published elsewhere in this issue.) It is proposed that this company shall proceed with the carrying on of this process, which is provisionally protected.

The directors have entered into an agreement with Mr. Maclaren whereby he becomes the managing director of this company for a period of ten years, and Mr. Johannes Harden will be the technical manager for the like period of ten years. The Board will consist of Mr. Archibald Finlay Maclaren, managing director of Electric Furnaces, Limited, who has been engaged in the foundry business all his life; Mr. Francis Barkley Henderson, director of The National Minerals Corporation, Limited; and Earl Russell, chairman of Plenty & Son, Limited.

Efficient Drying of Cement Rock.

The effect of the fineness to which Portland cement clinker is ground upon the physical property of the resulting cement is well understood, but it is not always appreciated that the fineness to which the material is to be ultimately ground depends to a large extent upon the drying of the product as it comes from the gyratories. Furthermore unless all the hygroscopic moisture is removed the clogging of screens and the cutting down of the capacity of fine grinding mills will result. It will, therefore, be seen that the dryers have played an important part in the development of the cement industry, which in 1910 in this country alone amounted to 70,000,000 barrels, as they have made it possible for cement plants to run at their maximum capacity.

In every industry the efficiency of machines performing certain operations has been studied with keen interest during the last few years in connection with the development of some scientific management scheme and in the cement industry the dryer has received particular notice as it was clearly seen that a considerable saving could be made by the use of the proper dryer and the utilization of the hot gases from the kilns.

Of course, the simplest form of machine for drying cement rock, clay, etc., is a single revolving shell equipped with lifting plates on the inside and having a furnace at one end and a fan at the other end, but the heavy expense of upkeep and its low efficiency due to radiation and the high temperature of the stack gases make it absolutely impracticable. The excessive fuel consumption of this type is decreased when the shell is protected by brick work and the hot gases pass under and back through it.

One of the most efficient and generally used types of dryers is the Ruggles-Coles, which consists of two concentric shells of steel rigidly connected and resting on eight bearing wheels and driven by a heavy cast gear located between the two tires.

It is of particular interest to note how the high economy is attained in the Ruggles type of dryer. In most dryers the hottest gases meet the cold material first with the result that the flames are suddenly quenched and a large amount of carbon passes off unconsumed which is the cause of one of the largest losses in efficiency. In the Ruggles type the hot air enters the central flue at the feed end, passes through this and then back through the space between the two shells, thus drying further the material which is advancing toward the other end.

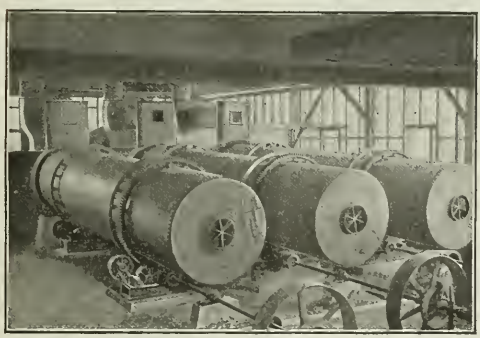
The material to be dryed as it is fed into the space between the concentric shells is picked up by lifting buckets and

dropped onto the inner shell. By revolution of the machine the material is then dropped on the outer shell and this operation continues until it reaches, due to the inclination of the dryer, the discharge end.

It will be seen that the long central flue acts as a large combustion chamber in which space and time are given for the perfect combination of oxygen and monoxide, thereby increasing the efficiency. A resulting exhaust temperature of 100 deg. Fahr. is generally found.

The capacity of any dryer depends directly upon the efficiency and when this is determined the capacity may be figured directly in proportion to the fuel consumed and almost inversely with the moisture contained in the material to be dried With a stack to produce draft the amount of fuel which can be burned is limited and, therefore, the capacity of the dryer cannot be increased without adding additional stack, which is a troublesome and expensive operation to say the least. By furnishing a fan to run at low speed the Ruggles-Coles dryer may be speeded up by changing a pulley and the capacity of the dryer increased in almost direct proportion. This is a very valuable feature when the material varies in moisture at different seasons of the year and in the cement industry sudden increases of output must be met.

As far as durability is concerned the single-shell type of dryer for cement rock, etc., is practically out of the question as the losses by radiation in such a machine are very large.



DRYER PLANT.

It is not unusual to see these machines red hot for one-third of their length. This means not only constant repairs, but that they are difficult to work near and are sometimes positively dangerous on account of fire. In the two-concentric-shell type, the cold material falling on the top of the inner flue at the hot end, it is impossible for this flue to become overheated, while the outer shell is always practically cold.

As to just what may be expected from a good cement dryer may be ascertained from the following data secured by actual test of a Ruggles-Coles dryer at the Pennsylvania Cement Company, Bath, Pa., in the drying of cement rock.

Calorific value of fuel used	14,100.
Fuel consumed per hour in lbs.	360.
Amount of moisture in material	2.3%
Water evaporated per hour in lbs.	1852.
Water evaporated per lb. of fuel in lbs.	5.1
Material dried per hour in lbs.	78,684.
Fuel per ton of dried material in lbs.	9.2
Heat lost in exhaust air	8.5%
Heat lost by radiation	5.0%
Heat used to evaporate water	40.9%
Heat used to raise temperature of material	45.6%

Hence the total efficiency, defined as the ratio of the heat used to raise the temperature of the material and evaporate the water to the total heat is 86.5 per cent.

Non-Corrosive Bronze.

"Tobin Bronze" is the title of a little booklet issued by the Ansonia Brass & Copper Company of 99 John Street, New York, from which the following information is taken. This bronze is the invention of Mr. John A. Tobin of the Engineer Corps of the U. S. Navy and is manufactured by the Ansonia Brass & Copper Company.

Tobin bronze, a "combination of copper with other metals" combines great mechanical strength with non-corrosibility under difficult conditions.

The specific gravity of tobin bronze is 8.379, hence less than that of copper (8.9).

As to the ability of tobin bronze to withstand chemical influences, tests made by Mr. Charles Taylor of the U. S. Assay Office in New York are interesting. Five samples of tobin bronze, each weighing about 5.5 ounces, were immersed in five different solutions for a period of five months each.

In a saturated salt solution the loss of weight in five months was 0.002 ounces (less than 0.04 per cent). In view of this its wide use in marine construction is not surprising. Thus the hulls of the American cup defenders Columbia and Reliance were entirely made of tobin bronze.

In a caustic potash solution the loss in weight of the 5.5 oz. sample in five months was 0.003 ounces. In a solution of sulphuric acid ("about one-third strength") the loss was 0.014 oz. In a U. S. Standard ammonia solution the loss was 0.161 oz.; in a solution consisting of one-half water and one-half U. S. Standard ammonia the loss was 0.081 oz.

Used in the form of bolts, rods, and plates, its resistance to wear and oxidation makes it a useful material for chemical works, dyewood extract works, tanneries, sugar houses, and coal mines.

When rolled hot, tobin bronze is remarkable for its high elastic limit, tensile strength, hardness, and unifom texture. It can be drawn to a very high degree of tensile strength. But in order to insure the best results in uniformity of tensile strength with the maximum degree of elongation, standard practice of the Ansonia Brass & Copper Company, as to ultimate tensile strength and elongation, of rods of different diameters is as follows:

Diameter of rods.	Ultimate tensile strength.
Up to 1 in.	60,000 lb. per sq. in.
1 to 1½ in.	58,000 " "
1½ to 2 in.	56,000 " "
2 to 2½ in.	54,000 " "
2½ to 3 in.	52,000 " "
Above 3 in.	50,000 " "

The elongation is not less than 30 per cent in 2 inches for all rods 1¼ in. diameter and larger. It is not less than 20 per cent. in 2 in., for all rods smaller than 1¼ in. diameter.

As stated before, the tensile strength can be made higher, but if this is done the elongation is decreased. The elastic limit is about one-half the ultimate tensile strength.

Tobin bronze can be welded electrically.

At a cherry red heat, tobin bronze can be forged as readily on steel. Bolts, nuts and rivets may be made from it either by hand or machinery.

Another feature in working it at cherry red heat is that it can be stamped and drop-forged in the same manner as steel.

Improved Method of Securing Battery Posts.

An arrangement for securing battery posts to the concrete foundation block is described by F. W. Sewell, in the *Informes y Memorias* of the Mexican Institute of Mining and Metallurgy, Vol. 2, No. 3. The arrangement is used at the mills of the Guanajuato Cons. M. & M. Company and the Zacatecas M. & M. Company.

A cast-iron shoe is bolted to the concrete block, and the

shoe, in turn, receives the battery post and holds it in place by suitable bolts and brackets. The shoe weighs 860 lb. and has a bearing surface on the concrete of 4 ft. 6 in. by 1 ft. It has a cavity cast in it 2 ft. long by 10 in. wide and 6 in. deep, into which the battery post fits.

The shoe is secured to the concrete block by two 2½-in. rods, each 5 ft. long, threaded at both ends. The threaded portions are of enlarged cross-section, so that the diameter of the base of the thread is not less than that of the remainder of the rod. The lower ends of the rods pass through broad, square, cast-iron washers embedded in the concrete, and are secured thereto by square nuts. Hexagonal nuts or lock nuts are used on the top end to hold down the shoe.

The battery post is held securely in the shoe by means of 1½-in. bolts 2 ft. 7 in. long, connecting brackets on the side of the post with the cast-iron shoe. At each end of the shoe are cast suitable projections which are bored vertically to receive the long bolts mentioned. The two side brackets, which weigh about 100 lb. each, are recessed into the 12-in. faces of the post and held in place by five ⅞-in. rods passing through the post from one bracket to the other. These rods are threaded at both ends, and lock nuts hold the brackets firmly in position. Passing vertically downward from each bracket to its respective end of the shoe are two rods, mentioned above, with square nuts to attach them to the shoe and hexagonal nuts to attach them to the bracket. The lowest part of the bracket is 1 ft. 2 in. above the top of the shoe.

In erecting the battery, the cavity in the shoe is half filled with oil, and the foot of the post is cut slightly larger than the cavity, the point being cut so that it will just enter the cavity. When the post is forced into place by means of tightening the connecting rods, aided by blows from heavy hammers on top, the oil is forced into the end of the post. This insures protection from rotting due to water. The entire post is then oiled occasionally until no more oil is absorbed, when it is painted.

Fibrous structure is one of the most useful properties of matter, and when found in mineral substance adds greatly to its value. The best grade of asbestos is beautiful silky fiber with a high degree of flexibility, elasticity and tensile strength. It can be spun into a thread so as to run 225 yards to the ounce.

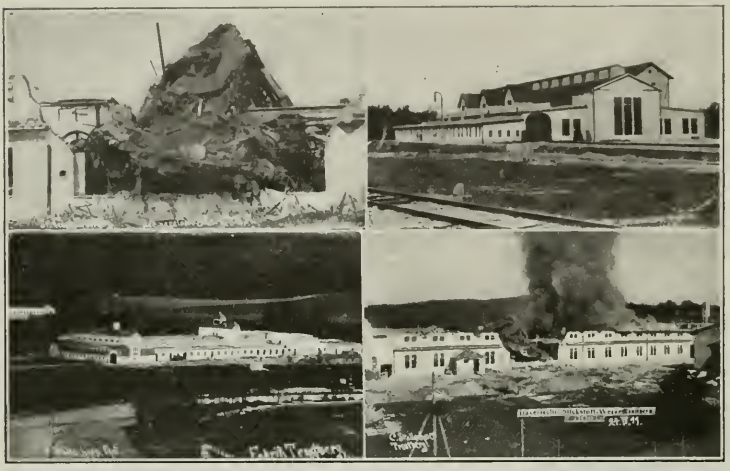
Acetylene Explosion in Cyanamide Plant.

The interesting pictures which are herewith reproduced are taken from some picture postcards sent us from Trostberg in Bavaria, Germany, where the Bayerische Stickstoffwerke have a monster plant, making calcium cyanamide. The factory is located in a valley with the snow-clad Alps surrounding it.

A couple of these pictures are of special interest, showing some details of an acetylene explosion which took place there on the 24th of June this year. Dealing with calcium carbide requires care and precautions, as acetylene is readily formed and even small amounts of acetylene mixed with air will give explosive mixtures. The strength of such an explosion is evident from the pictures.

From time to time reports have come from the European factories concerning similar disasters and it was expected that in this latest constructed plant all precautions had been taken, found in former experience, to make such explosions impossible.

It is only fair to state in this connection that in most older chemical industries where formerly many accidents occurred, these have been very largely reduced in recent years, due to a more careful arrangement and a better construction.



CALCIUM CYANAMIDE PLANT IN BAVARIA.

The lower right-hand picture shows the fire caused by the acetylene explosion; the upper left-hand picture shows a destroyed silo.

Alundum Refractory Exhibit.

The adjoining illustration shows the exhibit of alundum refractories displayed by the manufacturer, the Norton Company, of Worcester, Mass., at the last American Foundrymen's Convention in Pittsburgh.

REFRACTORY EXHIBIT.

As to the properties and manufacture of alundum refractories there is nothing to be added to the exhaustive paper of Mr. L. E. Saunders on page 257 of our May issue.

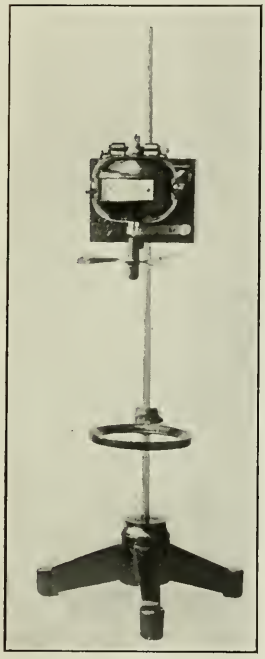
The exhibit brings out the numerous uses to which alundum refractories are now being put, including muffles of various

shapes and sizes, Cooch type filter crucibles, and crucibles for general laboratory work, combustion boats, extraction thimbles, filter tubes, filtering dishes, electric furnace sections and parts, corrugated furnace cores, arc lamp economizers, tubes, etc. It will be remembered that in the new Hering electric furnace, making use of the pinch phenomenon alundum is used for the electrode tubes with good success.

A New Rotating Anode.

The accompanying illustration shows a new design for a rotating anode for electrolytic analysis, made by Queen & Company, Inc., of Philadelphia, Pa.

The motor is of a new type totally enclosed except on the commutator end which is on top. A fan is mounted on the motor shaft immediately below the contact brush which supplies current to the anode, so that the possibility of gases entering the motor parts is eliminated. In this particular the tripod base, upright and the supporting ring for the dish cathode are heavily enameled. The clamp screws for adjusting the height of the motor and supporting ring have been designed so as not to injure the enamel. Current to the anode is supplied by a specially constructed brush, which bears against a suitable copper ring on the motor shaft. Provision is made to prevent excess of oil from the motor bearings finding its way down to the anode.



ROTATING ANODE.

The tripod base is of iron and heavy enough to prevent vibration and is fitted with soft rubber tips. Binding posts are provided for the current terminals and a rheostat not shown in the illustration controls the motor speed. The illustration does not show the electrodes. The anode is held in position by means of a knurled screw and can be quickly removed.

Institute of Metals.

The annual autumn meeting of the (British) Institute of Metals will be held at Newcastle-on-Tyne on September 20, 21, 22.

Sir Charles A. Parsons is acting as chairman of the local committee and Dr. J. T. Dunn as secretary.

Sessions for the reading and discussion of papers will be held on the mornings of September 20 and 21 at the Armstrong College; on the afternoons of these two days visits to plants will be made. In the evening of September 20 a conversazione will be held, and on September 22 a steamer trip down the Tyne River and back will be made.

Sir Gerard A. Muntz is the president and Mr. G. Shaw Scott, Custom House, Westminster S. W., London, is the secretary of the Institute.

The Non-Ferrous Metal Market.

The markets for non-ferrous metals generally showed a firm tone during August. The copper surplus was reduced in July, and the foreign demand coupled with the reduced production caused some expression of fear that a copper shortage was not outside the bounds of possibility. In zinc the situation is equally favorable, and that metal has a good outlook. Foreign consumption of zinc is heavy, and the price has advanced so that it is not impossible that there may develop a foreign demand for American spelter.

Copper.—The market has been firm, but without advances in price. Lake copper is quoted at 12⅝ to 12¾ cents; electrolytic at 12.40 to 12.45 cents.

Lead.—Business in this market has not been heavy, and the quotations are practically unchanged at 4.50 cents, New York, and 4.42½ to 4.45 cents, St. Louis. The St. Louis quotation is relatively high, as there is usually a greater disparity between New York and St. Louis lead markets.

Tin.—It is reported that the Syndicate in London has temporarily suspended operations, but the foreign market remains firm. The domestic market is good, but there is a scarcity of spot tin owing to dock strikes abroad. August tin was quoted at 42⅝ cents, with spot at 43⅛ cents.

Spelter.—The statistical position is strong and prices have advanced. The latest quotations are 5.90 to 6 cents, New York, and 5.75 to 5.85, St. Louis. The foreign market is reported as firm.

Other Metals.—Aluminium has not changed much during the past month. Quotations stand at 20 to 20½ cents for No. 1 ingots. Sales in antimony are small and prices range from 7.25 to 8.5 cents for various brands. Considerable business has been transacted in quicksilver and prices are steady at $48 to $49, New York, for 75 lb. flasks. San Francisco quotations are $48 for domestic and $45.50 for foreign orders.

Notes.

American Electrochemical Society.—The following gentlemen were elected members at the meeting of the Board of Directors on July 21: Albert A. Heimrod, Newark, N. J.; C. R. Corning, New York City; Frederic S. Mason, New York City; Irving R. Valentine, Schenectady, N. Y.; Thomas D. Robertson, Toronto, Canada; Otto Stalmann, Salt Lake City, Utah; John I. Blair, South Bethlehem, Pa.; Archibald W. Diack, Detroit, Mich.; Harrison W. Johnson, Niagara Falls, N. Y.; John M. Tippett, Colorado Springs, Colo.; Charles R. Barton Ellwood City, Pa.

Vanadium Steel for Aeroplanes.—The July issue of "American Vanadium Facts," published by the American Vanadium Company, of Pittsburgh, contains a great many interesting facts on the use of vanadium in steel making. Thus tubular shells of chrome-vanadium steel, not more than ⅛ inch in thickness are now successfully used as shafts for 50-hp motors to transmit the power on aeroplanes with an ample factor of safety. The shafts weigh 18 pounds each and are 2½ in. outside diameter, 40 in. long, with a 2¼ in. hole bored through their entire length. On account of its excellent forging qualities, its great strength and high elastic limit, and its freedom from crystallization, chrome-vanadium steel finds not only increased use for aeroplane, automobile, and marine motor purposes, but in the construction of chisels, rivet sets, punches, gears, etc.

Oildag for Aeroplanes and Automobiles.—Other interesting examples of the way in which the latest advances in aeroplane and automobile construction have been rendered possible by recent progress in chemistry and metallurgy, are given in a little pamphlet, issued by the International Acheson Graphite Company of Niagara Falls, and entitled "Three Little Stories of Three Big Events and Three Winning Lubricants." Lincoln Beachy, in his recent flight over the Falls of

Niagara, used oildag in the lubricating oil in the engine of his Curtiss biplane. Ray Harroun, the winner of the 500-mile automobile race in Indianapolis, used oildag and gredag. Harry F. Grant, the winner of the big sixth Vanderbilt race on Long Island, used oildag and gredag. It is hardly necessary to add here that oildag is deflocculated graphite diffused in oil (in colloidal condition); gredag is a combination of pure Acheson graphite and a high-grade grease, the two being blended in proper proportions.

The Driver-Harris Wire Company, Harrison, N. J., have developed a new high-fusion wire for combustion furnaces and laboratory heating furnaces. This wire which will be placed on the market under the trade name "Nichrome II" will be efficient for services up to 2000° F. and its chief points of advantage are its extremely high melting point and the proportionately greater resistance to oxidation.

The Electric Furnace Company, Alliance, Ohio, has completed its new furnace-testing laboratory. The building is located on a tract of land secured from the Stark Electric R. R. Co. at their Lake Park power house near Alliance and power for testing purposes will be received direct from the R. R. Company's bus bars. The plant will be supplied with a 200-kilovolt-ampere Westinghouse special regulating transformer, a complete set of electrical recording and indicating instruments, pyrometers, a small chemical laboratory, and equipment for microscopic examination of metals. The Electric Furnace Company has developed a line of electric heating furnaces for forge shops and similar plants, and the new testing laboratory is to enable the company to test new types of furnaces, that are being rapidly called for. The standard type of their electric furnace for heating bars and billets was described in detail and illustrated in a paper by Mr. Thaddeus F. Bailey on page 202 of our May issue.

The first by-product coke ovens in the United States were built in 1893 at Syracuse, N. Y. This country is far behind Germany and other European countries in adopting the economies resulting from coking coal in by-product ovens. Figures compiled by the U. S. Geological Survey show that, while beehive ovens in the United States in 1910 produced coke amounting to only 64.54 per cent of the coal burned, retort ovens in the same year gave a production of 74.9 per cent. The value of the by-products recovered from the retort ovens amounted to about one-third the value of the coke. The by-products recovered were 27,692,858 cu. ft. of surplus gas; 66,303,214 gal. of tar; 70,247,543 lb. of ammonium sulphate or its equivalent; 20,229,421 lb. of anhydrous ammonia; 4,654,282 gal. of ammonia liquor. The value of the recoverable but wasted by-products from beehive ovens in 1910 is estimated by the Survey at from $35,000,000 to $40,000,000. If all the coke produced in the United States were made in retort ovens, there would be a yield of gas over and above that required for the regeneration of the ovens, without by-product recovery, approximating 1,000,000 hp per hour for every day in the year. Complete data on the subject may be obtained from the U. S. Geological Survey, Washington, D. C.

The Bureau of Mines, Washington, D. C., has issued several pamphlets of interest. *Bulletin* 12 treats of the apparatus and methods for the sampling and analysis of furnace gases; *Bulletin* 14 on briquetting tests of lignite, with a chapter on sulphite-pitch binder; *Technical Paper* 1 on the sampling of coal in the mine, and *Technical Paper* 2 on the escape of gas from coal.

Personal.

Mr. A. D. Akin recently has accepted a position as managing engineer for the Monarch Mining & Smelting Company, Yavapai County, Ariz.

Mr. N. H. Brown, of Idaho Springs, Colo., has succeeded Mr. S. B. Tyler as superintendent of mines and mill of the Frontenac Consolidated Mines Company at Central City, Colo.

Prof. F. G. Cottrell, of the University of California, recently went to Montana to confer with Dr. J. L. Holmes of the Bureau of Mines in regard to the smelter smoke problem.

Mr. W. S. Ferris, superintendent of the cyanide plant of the San Rafael y Anexas Company, Pachuca, Mexico, was in Denver recently.

Mr. R. L. Hallett, formerly with the Arizona Smelting Company, Humboldt, Ariz., has been appointed superintendent of the Sandoval Zinc Company's smelter at Sandoval, Ill.

Mr. H. W. Hardinge has gone to Montana on professional business and will return to New York by way of Arizona.

Mr. George J. Hough, chemist and metallurgist, has recently established a laboratory and assay office at Salida, Colo.

Dr. L. D. Ricketts, who was appointed one of a committee of three to investigate the smelter fume problem at Anaconda, Mont., recently spent several days at Anaconda and vicinity.

Mr. Lloyd Robey, superintendent of the cyanide plant of the New York and Honduras Rosario Mining Company, San Juancito, Honduras, has spent a leave of absence in this country.

Mr. Sidney B. Tyler has resigned his position as superintendent of the Frontenac Consolidated Mines Company, Central City, Colo., and has accepted a similar position with the Continental Mining Company, Monclova, Coahuila, Mexico.

Mr. Pope Yeatman, consulting engineer for the Braden Copper Company, Chile, has returned to New York from an inspection of the company's mine and new concentrator.

Obituary.

William L. McLaughlin, of Deadwood, S. D., died at Deadwood on July 28, aged 50 years. Mr. McLaughlin had been identified with the mining and metallurgical industry in a managerial capacity for many years. He was at one time manager of the Horseshoe Mining Company and later manager of the Mogul Mining Company and the Wasp No. 2.

William C. Greene, known for his connection with the copper industry in general, and more particularly with the Green-Cananea copper mines and works at Cananea, Mexico, died at Cananea on August 5. The immediate cause of death was acute pneumonia induced by injuries received a few days ago previous in a runaway accident. Mr. Greene had a rather spectacular rise and fall in financial circles, and ultimately his copper interests were absorbed by other parties.

Mr. Alfred J. Bettles, of Salt Lake City, died in that city on August 3, 1911. Mr. Bettles was one of the foremost metallurgists in the West and had been identified with some of the largest projects in ore dressing. He was associated with Samuel Newhouse during that operator's development of the Cactus and Boston Consolidated companies. The immense mill of the latter was at the time of its construction one of the largest mills in existence. It is designed to treat the low grade copper ores of Bingham, Utah, carrying from 1.5 to 2 per cent. copper. Mr. Bettles adopted radically different steps in the treatment of this ore than were adopted in another mill built about the same time to treat the same ore. His judgment in installing stamps as grinding machines instead of Chilean mills and rolls, was seriously questioned at the time, and the departure attracted much attention in milling circles. The metallurgical results obtained in the Boston mill, however, apparently justified his judgment, although later the mill was remodeled to conform with more advanced practice and to obtain greater tonnage.

Mr. Bettles was recognized as a man of high integrity as well as excellent professional attainments. This combination rendered particularly valuable to his employers and associates at a time when so much depended on both qualities. In the last few years he had given attention to his own interests and to consulting practice. He was fifty-five years of age at the time of his death, which was due to uremic poisoning following an operation.

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Toronto Meeting of the American Electrochemical Society.

The report, printed elsewhere in this issue, of the Toronto meeting of the American Electrochemical Society held from September 21 to 23, will be found so complete that little could be added to the details of this highly enjoyable and successful convention. But we may use this opportunity to say a few words on some broader aspects of the remarkable development of this Society which is not yet ten years old.

Firstly, the Toronto meeting—the second one held on Canadian soil—confirmed what scrutiny of the membership list and of the papers published in the Transactions has long indicated: that in one respect at least the Society has already outgrown its name. It is no longer a national, it is an international society. In view of the unbounded hospitality which the visitors enjoyed at Toronto and the splendid arrangements which had been made by the local committee with Dr. W. Lash Miller as chairman and Mr. W. P. Cohoe as secretary, it goes without saying that everyone felt very much at home in Canada. And the coincidence of the first convention day being the day of the Canadian anti-reciprocity election only served to emphasize more strongly that in the interchange of ideas there can be no tariff. In the realm of exchange of ideas reciprocity expands freely by God-made laws.

Secondly, the glorious resurrection of Section Q with their clever and funny electric-furnace farce, presented at the smoker, deserves a word of notice. A keen observer who has attended many meetings of societies here and abroad has remarked that it is at the meetings of the American Electrochemical Society that "the most human crowd" is found. Nor is this to be wondered at. The Society is devoted to a borderland of sciences and industries and much of its work is pioneer work in virgin fields of science and engineering. We need only think of Bret Harte's stories to see how pioneer work of any kind brings out the human element, how it makes men respect each other for their accomplishments and love each other for their shortcomings, how it breeds a sense of humor. This was the spirit in which Section Q was born and this is the reason why the Electrochemical Society could not be without it. The fun of the farce presented at Toronto was so clever and good-natured that nobody could be hurt by the hits made in the fanciful play. Goethe's maxim of life of "Tages Arbeit, Abends Gäste, saure Wochen, frohe Feste," was followed in that long to be remembered smoker after the two heavy professional sessions of the first day.

Thirdly, for a Society with such a record as the American Electrochemical Society the danger would generally be great to rest on its laurels. But in this case the spirit of youthful unrest inherent in all electrochemists would not permit a policy of "let well enough alone." This is forcibly brought out in the report of the special committee appointed to consider and report "whether a higher standard of selection of papers read

at the meetings or published in the Transactions should be observed" and in the unanimous adoption of the committee's recommendations by the Board of Directors. These recommendations are now to be worked out to be carried into effect in due course of time. They involve no criticism of the splendid work done in the past, but recognize distinct possibilities of doing better in the future. As long as such a self-conscious and critical spirit is alive in those who are intrusted with the Society's management there can be no doubt that the career of the Society must continue to be one of ever-increasing usefulness.

The Eighth International Congress of Applied Chemistry.

The Eighth International Congress of Applied Chemistry—the first one to be held in this country—is now only one year off. The time has, therefore, come for everyone to start energetically to do his share to make the Congress a success worthy of this country and American chemistry.

The Congress has been organized on the principle of broadest chemical democracy. As the Acting President, Dr. Wm. H. Nichols, remarked in one of his early speeches: "Every chemist in the United States must feel that he is part of the Congress from the very creation of the organizing committee down through all the details of management to the very close of the Congress."

The different sections have been organized and are ready to begin their systematic work. But every single chemist in the United States himself, directly or indirectly through his professional, business or educational affiliations, has a personal share of responsibility in the success of the Congress.

The success of the Congress will depend essentially on the contents of its printed Transactions. Thus the securing of the papers becomes the most important task of the near future. Tentative rules and regulations for the papers have been formulated. They are printed in full on pages 510 and 511 of this issue. Since they will be adopted in their final form in December, it is necessary that any criticisms or suggestions should be made to the Secretary of the Congress before December.

But aside from this more formal matter, the all-important point must be kept continually in mind that everything be done to secure for the Congress any worthy paper and thus to make the Transactions a truly representative document

Transfer of the Ray Smelter.

The Ray Consolidated Copper Company, Arizona, was planned as a complete copper mining, concentrating and smelting concern, with full equipment for carrying on these several operations. Recently, however, the smelting end of the project has been abandoned before its completion, and the plant has been sold to the American Smelting & Refining Company. Apparently the Ray company intends to stick close to mining and concentration, in which fields its promoters have gained signal success; while the Smelting company takes advantage of an excellent opportunity to extend its smelting operations and gain a foothold in Arizona.

The arrangement should be advantageous to both companies, each of which will operate in its chosen and successful field. The Ray company has assurance that its concentrates will be treated by a friendly company practically as cheaply as in its own plant, and at the same time it is relieved of the arduous work of conducting a large smelting industry. The smelting company, on the other hand, enters into competition in one of the largest copper-producing States in the Union, and in a district where almost 150,000,000 tons of low grade copper ore are developed.

The promoters of the Ray began their immense operations in the Utah Copper Company, later developing the Ray in Arizona and the Chino in New Mexico. They also have been interested in Nevada Consolidated. Almost from the beginning they have had close working relations with the Guggenheims, and the recent transfer of the Ray smelter to the latter is a logical if unexpected move.

Some Recent Crushing and Grinding Tests.

Evidence is accumulating on the subject of crushing and grinding which forces a modification of some views formerly held, and the results obtained in some recent tests must be accepted regardless of stubborn prejudice or nice theory. In the Synopsis of Current Literature in another part of this issue reference is made to the work of ball mills in Australia and the excellent performance of these machines in dry crushing. Three-inch ore is crushed to pass approximately 30-mesh screen, with a power consumption of about 0.5 hp per ton. The quantity of minus 150-mesh material produced is not as great as with other pulverizers and the product is sharp and granular even in the finer sizes.

This reference to the ball mill recalls the belief which is finding acceptance among many engineers that the ball mill and tube mill in tandem will be found a most efficient system of wet grinding. This belief is substantiated in a large measure by the recent tests of the so-called Giesecke ball-tube mill in South Africa. This machine is a ball mill and tube mill combined, with compartments for coarse crushing and fine grinding, and a suitable connection for passing the product of the former to the latter. It has a total length of 24 ft., of which 5¾ ft. at the feed end is the ball mill compartment with a diameter of 7 ft. 6½ in., and the balance is the tube mill with a diameter of 5 ft. 10 in. Steel balls of various sizes ranging from 1¼ in. to 4 in. are used as the crushing medium. The larger sizes are used in the ball end and the small in the tube. Their total weight is 23 tons. The liners are 2-in. iron plates with one end bolted to the shell and the other projecting inward about 9 in. The entire mill weighs 68 tons without load, runs at 24 r.p.m. and requires about 235 hp when running. Its capacity is 360 tons per day of 7-in. material, which is reduced so that 80 per cent will pass a 200-mesh screen.

Here is a departure in grinding machines, with preliminary results that are very attractive. If the cost of operation and maintenance should prove to be no greater than that of stamps and tubes a marked saving is possible in cost of installation as compared with the latter combination. And even if the Giesecke mill in the form described should, for various reasons, prove an unwieldy combination, there still exists the probability that separate ball and tube mills may prove an efficient combination.

One feature in particular which contributes to the capacity

of the Giesecke mill is the small quantity of water used in grinding—just sufficient to make the pulp flow. This insures an efficient grinding action of the balls approaching that of the dry ball mill, where no fluid other than air can tend to wash the particles of ore out of the path of the rolling and tumbling balls.

The Giesecke mill or the separate ball and tube mill combination gives stage crushing in a compact apparatus. Judging from the success attained by the ball mill and the tube mill when used separately for specific purposes, their combination should be more thoroughly investigated, and the tests made in South Africa will serve to draw more attention to their possibilities.

Of a different nature, but relating to the subject of crushing, were the tests on rolls and Chilean mills as grinders of disseminated copper ore at the Miami mill in Arizona. In the construction of this plant one section was equipped with rolls and the others with Chilean mills. The tests have been completed to the marked advantage of the Chilean mills, which, although noted as producers of slime, did more efficient work in liberating the copper mineral from gangue than the rolls, which gave a more granular product. This test adds to the prestige of the Chilean mill in copper concentration in the West and gives it a position which it holds apparently through merit alone.

Wages in the Steel Industry.

In the past four years the steel industry has come to occupy a very unfortunate economical position. Its selling prices were admittedly not inordinately high in 1907, although they were entirely satisfactory. From the average prices of that year to the present time the decline has been fully $10 a gross ton as the average of all products outside of standard sections of steel rails, which are unchanged in price but are subject to much more rigid specifications and to a higher manufacturing cost. The general average level of commodity prices in the United States has not declined materially, if at all, since 1907, so that the steel industry stands at a very great disadvantage, in having suffered a reduction of $10 a ton in its selling prices, equal to between 20 and 25 per cent, computed upon material in the finished rolled form.

The average wage rate in the steel industry is higher than in 1907, through the general advance which was made in the spring of last year. The average increase is not large, but the fact that the change in wage rates has been in this direction places strong emphasis upon the peculiar economic position resulting from the great reduction in selling prices.

The last vestige of artificial control of prices in the steel industry appears to have vanished, if the maintenance of the $28 price on standard section rails be disregarded. It may be due to a survival of the co-operative idea or may be due to a refusal of the railroads to entertain lower prices. The railroads realize fully that the price is not high for the quality of rail they desire, and that a period of open competition in rail prices would make it much more difficult to secure the improvements in quality which they desire. Thus we may say that there is no longer any artificial control in steel prices through joint action or tacit understanding among the steel producers. The course of the markets in the past three months has clearly betokened conditions of open competition. Declines have not been clear cut, as have been those which occurred during the long period of a controlled market, but have been of the nature of a gradual sagging, with resistance on the part of sellers yielding to the necessity of making concessions in order to obtain individual orders. The downward trend in prices continues at this time and no indication is found in the markets that it will be arrested other than by prices reaching the cost level. Below that level they may possibly go, for most of the producers have large reserves and may become willing to make sacrifices in order to retain their trade. The "battle of the giants" which the daily newspaper paragraphers have delighted at times to predict is not among the possibilities, however, unless the total tonnage of demand should fall much below the present level, for the "giants" have not sufficient capacity to supply the existing demand. There are enough producers in a relatively poor position to be eliminated to leave a comfortable tonnage for those in better position.

In the circumstances outlined a general reduction in wages in the steel industry seems to be a certain prospect, and that not at a distant date but in the relatively near future. There is reason to believe that the subject has been thoughtfully discussed in the inner circles for several weeks past. In any circumstance a wage reduction is to be deplored, and would be a serious matter, but there seems to be no easy means of avoiding one.

The decline in steel prices has been so great that the average level of prices, outside of standard rails, is computed to be not more than about $4 a ton higher than the lowest level ever recorded, the average prices in the years 1897 and 1898, when individual commodities fluctuated but little, being about $4 a ton lower than the present level. Since then costs have increased in some respects and decreased in others. The wage rates in the iron and steel industry proper have greatly increased; the cost of ore mining has greatly increased, due to the necessity of seeking ore farther underground. Freight rates paid upon raw materials have materially increased, and practically all the commodities the industry buys are much higher. To offset these increases, which amount to very much more than $4 per ton of finished product, the industry has introduced many economies in practice. Its wage rates have not, we may assume, increased materially more than rates in other industries, and not as much, probably, as in the building industry, and it is particularly unfortunate that no means seems to be afforded whereby savings can be effected in other directions.

The situation is simply this: The steel industry finds itself with much lower selling prices than formerly, and naturally turns to the wage rates it pays. If the general level of wages should come down, then it would be fairer that the industries which the steel industry patronizes, like railroading and the manufacture of its various supplies, should themselves reduce wages, seeing that the steel industry must pay them higher rates than formerly. The steel industry has effected many economies in its processes, and it is unfortunate that if it cannot keep the results as return upon its capital investment it cannot at least use them to avoid a reduction in wages. A fair adjustment based upon theory would call for economy to be practised at these outside points.

American Institute of Chemical Engineers.

The fourth annual meeting of the American Institute of Chemical Engineers will be held in Washington, D. C., from Dec. 20 to 22 (Wednesday to Friday).

A number of papers will be presented on the general subject of patents and on the manufacture and testing of explosives as well as on other chemical engineering subjects.

One day will probably be devoted to visits to chemical engineering plants in Baltimore and vicinity. Visits to laboratories and other points of interest in Washington will also be arranged for.

Latest Results in Electric Furnace-Smelting of Iron Ore.

The Trollhättan furnace is now working with round electrodes 550 mm. diameter, screwed into each other and thus entirely consumed. These screw-joints hold well, not giving any trouble by heating at the joints. While the electrode consumption of this furnace using square unjointed electrodes, from Nov. 15, 1910 to April 9, 1911, was 10.28 kilos per ton of pig iron, counting the lost butts, the consumption from Aug. 4 to Sept. 4, 1911 was 6.2 kilos, and it has now (middle of September), reached 5.5 kilos.

The furnace has now been running steadily since Aug. 4. The gas pipes have been made stronger, so that pressures of 0.3 to 0.4 atmosphere are now used under the vault. The ore is charged more towards the middle of the furnace shaft, and the percentage of carbon dioxide in the escaping gas has been increased to over 30 per cent.

The output of pig iron has been increased, the consumption of charcoal has decreased, and the output per horsepower year has reached the figure of 5.05 tons per kw year, or 3.7 tons per hp year, this being the figure reached in the week Sept. 3-9. The detailed figures for that week are as follows:

Ore	192.5 metric tons
Limestone	7.7 metric tons
Charcoal	44.2 metric tons
Electric energy used	228,000 kw-hours
Average power	1357 kw
Pig iron produced	131.4 tons
Slag	22.1 tons
Charcoal per ton pig iron	336 kg
Pig iron per kw-year	5.05 tons
Pig iron per hp-year	3.70 tons
Kw-hours per ton of pig iron	1736
Hp-hours per ton of pig iron	2315

The builders and operators of this furnace feel every reason to be satisfied with these splendid results, which fulfill all expectations.

At about the end of 1911, four similar large electric pig iron furnaces will be in commercial operation in Scandinavia, viz.:

1—2500 hp at Domnarfet, Sweden.
1—3500 hp at Tyssaa, Norway.
2—3000 hp at Hagfors, Sweden.

The Western Metallurgical Field.

San Juan District, Colorado.

The production of vanadium and uranium from this district is assuming the proportions of a considerable industry, and the prospect is that within the next year a great deal of attention will be given to mining and milling these ores. A great deal of unfounded and irresponsible talk is current regarding this district, especially regarding the alleged fabulous value of the ores and the probability of supporting a thriving radium industry. The facts seem to be that in San Miguel County there are commercial deposits of uranium and vanadium ores some of which are treated locally and others shipped abroad.

The only local treatment of vanadium ore in operation now is carried on at Newmire by the Primos Chemical Company. This company is operating under lease and bond, the mines and mill owned by the Vanadium Alloys Company. The latter concern worked its properties for some time, but met with difficulties in the mill work. These were in a measure due to the fact that the work was new and there were no guides to successful operation. In addition to this some of the equipment installed was unsuited to the work. The Primos Chemical Company has now been operating the mill for about two years and is in a fair way to make a success.

The ore treated is an impregnated sandstone the nature of which will be found more fully described in Mr. Bleecker's article elsewhere in this issue. The average grade of ore milled is not known exactly, but is not in excess of 1.5 per cent vanadium and probably lower. The details of treatment are not given out by the company, but the method probably does not differ greatly from that used by the Vanadium Alloys Company, the principal improvements being in the mechanical operation. The former company roasted its ore with salt in a McDougal roaster. This roaster as installed proved very unsatisfactory and gave rise to constant trouble. Difficulty was experienced in getting the heat evenly distributed to the various hearths and on one occasion a rabble arm broke by sticking in the fused mass and caused a serious explosion when the cooling water was injected onto the hot material. Although the same roaster is now in use, it is understood that the Primos company has remodeled it to use oil and that it gives fair satisfaction, although plainly not adapted to this kind of work.

The roasted ore is leached with water, getting a solution of sodium vanadate, and this is precipitated with ferrous sulphate. Various means of leaching have been tried at this mill, both with and without agitators, and the latter have been discarded. The vanadate of iron obtained by the precipitation with ferrous sulphate is shipped to Primos, Pa., where it is reduced to ferrovanadium.

The ores shipped abroad are rather higher in vanadium content, containing probably 5 per cent to 6 per cent. It has been a matter of surprise to some that this ore could stand the charge of transportation abroad, but one shipper is authority for the statement that the rate from Placerville, Col., to Liverpool is 5 cents per ton less than from Placerville to Pittsburgh or common eastern points. The rate from Placerville to Pueblo, Col., is $6 per ton; from Pueblo to Galveston, $5, and from Galveston to Liverpool, $2.50. Total, $13.50. The rate to Pittsburgh is $13.55.

Silverton District.

The mining and milling industry in the vicinity of Silverton is in a quiescent state, due to a number of causes. Briefly these are the complexity and low grade of the ores, and the fact that the processes of milling installed some years ago are not applicable to present conditions. Recently there has been a commendable tendency to investigate new and accessory processes and the prospect is fair for renewed activity in the near future.

One of the greatest difficulties is the separation of the complex ores into concentrated products of shipping grade. Iron, lead, copper and zinc occur together in many of the ores, and ordinary wet processes of concentration are not profitable. Recently there has been renewed interest in the possibilities of electrostatic separation as a means of reviving many dormant properties. Inquiry brought out the fact that many mills already built but not in operation would be benefited by adding electrostatic separation to their present process, enabling them to separate the copper and iron from the zinc and make the latter a profitable metal instead of a stumbling block. Probably a score of these mills could be so benefited immediately. Two solutions of the problem are in sight, viz., the installation of electrostatic machines in the individual mills or the establishment of a separate electrostatic separation plant to be supplied with zinc-iron-copper middlings from the various mills.

Undoubtedly the former plan would be more satisfactory to all concerned if the individual companies could raise the necessary money. The latter offers the advantages and disadvantages which any custom plant gives to a community, but mainly its success depends on the assurance of a constant and adequate supply of material.

Still another possibility for the improvement of the industry in the Silverton district lies in cyanidation. There are silicious gold and silver ores which have proved amenable to this treatment, giving extractions of 90 odd per cent of the gold and 65 per cent to 75 per cent of the silver. An effort is now being made to secure a guaranteed tonnage of these ores to supply a custom cyanide mill and some progress has been made in this direction.

Idaho.

The Lost Packer smelter, which is situated about 100 miles from Mackay, Idaho, the nearest railroad point, was blown in for the summer campaign early in August. This is the first time in three years that this plant has operated, and it is not hard to see that the work is carried on under difficulties. Operations must cease with the advent of winter on account of the impossibility of getting fuel and other supplies. Coke has to be hauled to the smelter and matte has to be hauled out to the railroad, and considering the shortness of the campaign it is rather remarkable that the plant can be operated at all. Since the last operation of the plant a great deal of mining development has been carried on and the prospect is fair for a good campaign this summer.

In the Silver City district of Idaho the Rich Gulch Mining Company is preparing to erect a modern amalgamation and cyanidation plant. For the past six years this company has been opening a large body of low-grade gold and silver ore, averaging about $11 per ton, and sufficient ore has been blocked out to warrant the erection of the milling plant. The estimated cost for mining and milling is about $3 per ton, and with reasonably good extraction of the metals there should be a good profit.

Gold, Silver, Copper and Lead in California.

The mine output of gold, silver, copper and lead in California in 1910 had a value of $27,020,405, according to figures compiled by Charles G. Yale and just made public by the United States Geological Survey. The production of gold was $19,715,440; that of silver 1,840,085 fine ounces, valued at $993,646; that of copper 48,700,756 lb., valued at $6,184,996; and that of lead 2,870,977 lb., valued at $126,323.

These figures show a decrease in the output of gold, silver and copper as compared with the figures for 1909, but a large increase in lead. The 1909 figures are as follows: Gold, $20,237,870; silver, 2,098,253 oz., valued at $1,091,092; copper, 57,288,281 lb., valued at $7,447,476; lead, 1,502,597 lb., valued at $64,612.

There were 1079 mines producing gold, silver, copper or lead in California in 1910, of which 564 were gold placer mines. Of the deep mines 485 were gold mines, 9 were silver mines, 10 were silver-lead mines and 11 were copper mines. Of the placer producers 168 were hydraulic mines, 72 were dredges operated by 41 companies, 139 were drift mines in ancient river gravels and 185 were sluicing mines. Measured by the number of producers as well as by tonnage and metal output, deep mining decreased somewhat in California in 1910. Among the placers sluicing decreased also, but dredge and drift mining increased.

In 1910 there were 2,697,885 short tons of ore from deep mines sold or treated in California, with an average total recoverable value of $6.71 per ton, against 3,041,688 tons in 1909, with an average recoverable value of $6.41. Of the total recovered value in ores in 1910, milling ores yielded $10,005,190 in gold and silver; and smelting ores, $1,669,008 in gold and silver, $6,184,996 in copper and $126,323 in lead. Placers produced $8,888,795 in gold, of which $635,498 was from hydraulic mines, $516,929 from drift mines, $7,550,254 from dredges and $186,114 from sluicing mines, the yield from dredges alone being over 38 per cent of the total gold output from all sources in California in 1910.

Gold dredging has been of constantly increasing importance in California since 1899, when the yield from this source was $206,302. In 1910 it was $7,550,254. The total gold output by dredges in California to the end of 1910 was $40,318,775. The three great dredging fields are those of Yuba, Sacramento and Butte Counties, of which the Yuba County field showed an increased output of $730,557 in 1910, while the production from Butte County decreased $501,038 and that of Sacramento County decreased $164,542.

Of the total gold placers furnished $2,468,505 from Butte County, of which $2,389,235 was from dredges; $1,394,537 from Sacramento County, of which $1,369,594 was from dredges; and $3,199,952 from Yuba County, of which $3,172,476 was from dredges. Dredges also produced $191,900 in Calaveras County, $213,980 in Merced and Stanislaus Counties combined, and $109,182 in Siskiyou County.

Hydraulic placers produced $179,105 in gold from Siskiyou County and $328,154 from Trinity County.

Drift mines produced $147,599 in placer gold in Nevada County and $151,743 in Placer County.

Surface or sluice placers produced $29,700 in Plumas County, $22,339 in Siskiyou County and $27,476 in Yuba County, and smaller amounts of gold in most of the other producing counties of the State.

The largest increases in gold production by counties in California were $347,461 in Amador, $80,815 in Mono, $122,363 in Sierra, and $734,408 in Yuba. The largest decreases were $499,288 in Butte, $292,866 in Calaveras, $78,885 in Mariposa, $126,752 in Nevada, $272,940 in Sacramento, and $310,077 in Tuolumne. The Mother Lode in Amador, Calaveras, Eldorado, Mariposa and Tuolumne Counties produced 1,170,497 short tons of ore in 1910, with a total recovery in gold and silver valued at $4,435,802, or an average of $3.78 per ton, against 1,338,160 short tons yielding $4,854,355, an average of $3.60 per ton, in 1909.

Arizona and New Mexico.

The porphyry coppers continue to make an excellent record in ore development and milling. It is claimed that the work at the Ray Consolidated is resulting in 70 per cent recovery. Four of the eight sections of the concentrator are now in operation treating about 800 tons each per day, or slightly below their capacity of 1000 tons each. It is probable that no more sections will be placed in commission until after October.

During the past seven months the Inspiration Copper Company has added over 7,000,000 tons of developed ore to its supply. This gives a grand total of over 25,000,000 tons of developed and partly developed ore, averaging 2 per cent copper. Based on a milling capacity of 5000 tons per day, this amounts to about 14 years' supply. Plans for the mill are now being prepared and although a definite site has not been selected the Inspiration will shortly have to be counted among the producers.

The first section of the Chino mill was to have gone into commission before the end of September, treating 1000 tons of ore per day. A good record has been made in bringing this property to the producing state, only two years and three months having elapsed since its acquisition by the present owners. The entire original milling plant of three sections should be in operation before the end of this year and the two additional sections recently decided upon will be ready for operation next year. Over 50,000,000 tons of ore have been developed by means of churn drills. Two years' supply of ore is already stripped and three steam shovels are constantly at work on further stripping operations.

American Mining Congress.

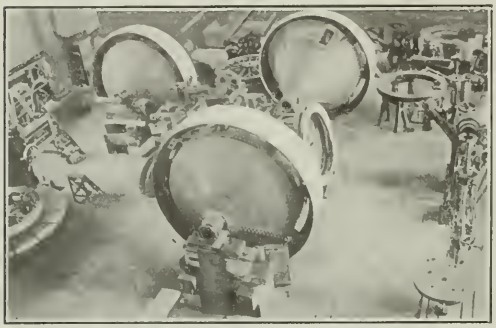
The annual convention of the American Mining Congress will be held in Chicago, October 24 to 28 inclusive. This is

the second change of date that has been made since the first announcement of the meeting, and corresponds with the date originally set. It is expected that President Taft, Secretary Fisher and Director Holmes will be present, and the change in date is for their accommodation. Matters of considerable interest to the metalliferous mining industry will come before this session of the congress, and it is expected that steps will be taken which will result in the national government taking a more direct interest in metallurgical problems, particularly those of the Western states.

Driving of Large Pebble Mills.

A certain innovation has recently been made in the driving of large-diameter pebble mills. The ordinary and common practice has always been gear drive owing to the unusual moment of high power necessary to start a mill from the state of rest.

An innovation was made by the Calumet & Hecla Mining



DRIVING OF LARGE PEBBLE MILLS.

Company, who, after testing out a gear-driven Hardinge 8-ft. conical pebble mill, equipped the 8-ft. mill with an improvised pulley around its largest diameter, extending the pulley diameter to 10 ft. The testing out of this mill under this improvised condition appeared to show that it would take about three to four horsepower in excess of the gear-driven mill. The results of this experiment, however, did not deter the Calumet & Hecla Company from instituting this method of driving this class of mills, which they were then building, as is shown by the enclosed photograph of the Calumet & Hecla shops.

This enormous pulley on a direct-driven mill was obtained with a 26-in. belt. It finally worked out that instead of taking an excess of power there was a decrease in power nearly to the amount as had been expected would be an increase. In other words, while a 35-hp to 36-hp mill under the gear-driven system was expected to require 40 hp, it was found that two of these mills were started from a state of rest with a full load when driven by a 70-hp motor without the motor showing any effect of an excessive load.

It is estimated that these mills are taking in fact from 30 hp to 32 hp.

The point of interest in a mechanical way is the elimination of gears at a sacrifice of a larger belt. It is stated that a belt of about 20 in. in width would give the necessary power. So successful was the operation of the four mills so constructed that the Calumet & Hecla Company are completing the balance of the first installation of about eight to ten more mills under the same conditions.

Arizona Copper Company.

It has been definitely decided to proceed with the erection of the new works which the Board of Directors have had under advisement for some time. Dr. L. D. Ricketts, of the Cananea Consolidated Copper Company, has been retained as consulting engineer for a period of three years. He will have charge of the preparation of plans and erection of the works, and will be assisted by the company's general manager, Mr. Norman Carmichael. It is expected that plans will be ready to submit to the board this fall.

New Mill for Inspiration.

For some time past there have been reports of a consolidation of the Inspiration and Miami properties, or at least of joint operation of a large milling plant. It appears now, however, that no agreement has bee nreached and that the Inspiration company will proceed with the construction of its own concentrator. The overtures for joint operation of a mill are said to have emanated from the Miami company, and the suggestion for consolidation from the Inspiration, but the various propositions did not meet with the favor of the two interests.

The capacity of the new Inspiration mill will be 5000 tons per day. The present developed tonnage of ore is 24,000,000 tons and new drill work is adding to this amount monthly

The Iron and Steel Market.

The steel market has again gone contrary to expectations. July and August proved to be months of fair activity, when extreme dullness had been expected, while September has shown a decline in prices and in activity when improvment was confidently expected. There have been no disorderly breaks in prices, but instead a constant though very gradual sagging, somewhat more rapid in September than in July and August, which was quite natural in view of the reduced demand.

The decrease in activity has been most marked in respect to new buying. Specifications on contracts have been fairly satisfactory, though hardly at as good a rate at the close of September as during August. Buyers are pursuing an extremely conservative course, keeping their stocks down to the lowest possible notch.

A very unfavorable feature of the situation from the standpoint of mill operations is the fact that buyers are ordering and specifying in very small quantities, and the mills themselves have no accumulation of specifications. As a result rolling schedules must be made up almost from day to day, roll changes are frequent and tonnage outputs at the mills are disappointing while the production cost of course is increased. Buyers cannot be induced to place larger specifications, for as it is specifications are obtained only with difficulty, and mills cannot accumulate so as to arrange specifications to advantageous rolling because when material is once specified the buyer insists on the earliest possible delivery.

A rough estimate would indicate that during September the independent steel interests operated at an average of 65 per cent. of their capacity and the United States Steel Corporation at an average of 75 per cent. of its capacity, which means that the domestic business was divided approximately according to the respective capacities, while in addition the steel corporation produced its usual large tonnage of export material.

Finished steel products, with the exception of rails, have receded since July 1 by $1 to $2 a ton, plates, bars, sheets and tin plates having declined $2 a ton and structural shapes, wire and merchant pipe $1 a ton. These recessions have not been by definite drops but by slight gradations. A trifle more than one-third of the total decline occurred in September. During these three months open market conditions have prevailed, practically for the first time in a decade or more.

The steel industry has lost practically all its optimism. There is little if any expectation of an improvement in demand this year. Conditions are regarded as although unsatisfactory, but the blame is attached more to excess of capacity than insufficiency of demand. The country has been making and consuming pig iron at the rate of about 23,000,000 tons a year, ca-

pacity being slightly in excess of 33,000,000 tons. The best output in 12 consecutive months has been about 30,000,000 tons, from August 1, 1909, to August 1, 1910, but that output was not all consumed. The best previous record was 27,000,000 tons in the 12 months ended October 31, 1907. These are now recognized to have been exceptional periods. The recent rate of output represents the maximum capacity in so recent a year as 1905, when this was the production with all capacity employed and conditions generally satisfactory.

While the continued decline in prices has not brought the average level of finished steel products down to the historic low level, the present average is not more than $4 a ton above that level, made in 1897 and 1898, at a time when wage and commodity rates were very much lower than at present. The advances in these items amount to much more than $4 a ton, so that as the steel industry was making very little money in 1897 and 1898 it would be losing heavily at this time were it not that various economies have been introduced. It is recognized that with prices as low as they are now, and showing prospects of further declines, a general reduction in wages can hardly be averted, and reductions averaging 10 to 15 per cent. are likely to be ordered before the close of the year.

PIG IRON.

Southern iron has again become easily obtainable at $10, Birmingham, a price which almost disappeared in August. Sales are light, consumers having bought more heavily than necessary considering the rate of consumption in the past two months. Foundry iron in other districts is practically unchanged. In the central west deliveries of merchant basic iron have decreased somewhat. A new low price on basic iron at valley furnaces has been made, 1000 tons being sold for prompt delivery at $13,50, delivered Pittsburgh or $12.60 f.o.b. valley. Prices are quotable as follows: Birmingham, $10 for No. 2 foundry; delivered Philadelphia, $15 for No. 2X foundry, $14.75 for basic; f.o.b. Chicago district furnaces, $14.50 for No. 2; f.o.b. valley furnaces, 90c higher delivered Pittsburgh; Bessemer, $15; basic, $12.60 to $13; malleable, $13; forge, $13; No. 2 foundry, $13.50.

STEEL.

The smaller open-hearth steel mills are offering billets and sheets bars at substantial concessions, $2 a ton below prices prevailing early in August. The large mills making Bessemer steel have important contracts based upon arrangements for monthly adjustment of price, and can not afford to disturb such adjustments by selling at prices the limited amount of steel which could thus be absorbed. The result, due to this peculiar market alignment and in no respect to difference in cost of manufacture or availability for finishing operations, is that open-hearth material is quotable at $2 less than Bessemer. We quote open-hearth steel f.o.b. Pittsburgh district mills at $19 for billets and $20 for sheet bars and Bessemer steel f.o.b. mills Pittsburgh or Youngstown districts, $21 for billets and $22 for sheet bars. Rods are nominally quoted at $27, Pittsburgh, but could probably be bought at $26.

FINISHED MATERIAL.

Due to open market conditions a considerable range of prices exists on several of the products, according to tonnage and specifications involved in the given order. Prices quoted below are the ordinary minima for desirable orders, but in exceptional cases the prices on bars, plates and shapes are slightly shaded. Small orders and orders for forward delivery generally command slightly higher prices than here quoted. Prices are f. o. b. Pittsburgh, except where otherwise stated:

Rails standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, 1.25 cents for tank quality.

Shapes, 1.30 cents for beams and channels, 3 to 15-in., inclusive, zees, and angles 3 x 3 and larger.

Steel bars, 1.15 cents, base; iron bars, 1.25 cents, Pittsburgh; 1.22 cents, delivered Philadelphia; 1.20 to 1.25 cents, Chicago.

Wire nails, $1.65, base: plain wire, 1.45 cents, base.

Black sheets, 28-gage, 1.90 cents; galvanized, 2.90 cents; blue annealed, 10-gage, 1.40 cents; painted corrugated roofing, $1.35 per square; galvanized, $2.50.

Tin plates, $3.60 per base box.

The Non-Ferrous Metal Market.

The September market was not characterized by anything unusual either in prices or sales. The foreign situation in zinc continues to indicate a shortage there and an advance in price. The domestic lead market at St. Louis has resumed almost normal relation to the New York market. Spelter and copper have been without feature and prices have been about stationary.

Copper.—There has been a disposition on the part of producers to make concessions in order to get business, and the largest transactions have been at receding prices. The market has been almost stationary at 12½ to 12⅝ cents for Lake, and 12.30 to 12.35 cents for electrolytic.

Lead.—The New York market has held steadily to 4.50 cents since our last report. The St. Louis market has declined somewhat on account of unusually large offerings of Missouri lead, and the latest quotations are 4.35 to 4.40 cents, which is more nearly in accord with the New York price.

Tin.—Prices for spot tin have fluctuated widely in the past month. At one time a premium as high as 4 cents per pound was paid for spot tin in New York, but later this receded to ¾ cents. In general, however, the market has been dull and September tin was quoted at about 41¾ cents.

Spelter.—The stocks of this metal seem to be low and consumers are apparently well supplied. Efforts have been made to create an interest in future business, but the only result has been a tendency toward lower prices. The New York market has stood at about 5.83 to 5.90 cents, and St. Louis at 5.70 to 5.75.

CORRESPONDENCE.

Fixation of Atmospheric Nitrogen.

To the Editor of Metallurgical and Chemical Engineering:

Sir—Referring to my letter concerning the Notodden and Rjukan plants for the production of nitrates from air (published in the September issue, page 436), I have just noticed a mistake in the fourth paragraph. This paragraph should read as follows:

"With reference to the chemical part of the Rjukan plant it had been decided some time ago to use the absorption towers of the Badische Company. It has been decided recently, however, that absorption towers of the Birkeland and Edye type will be used at the Rjukan plant" (instead of Notodden plant, as said in the former letter).

Paris, France. A. S. Garfield.

Aluminium in Ferrotitanium.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In reply to Dr. G. B. Waterhouse's criticisms of the content of aluminium in carbonless ferrotitanium made by the aluminothermic process, I would like to make the following remarks:

The addition of aluminium to carbon-free ferro-titanium, containing 24 per cent of titanium, is intentional and decidedly useful. Titanic acid is well known to be very difficult of fusion, and for this reason the titanic acid which is being formed from the ferrotitanium separates out only with great difficulty, if alone present. If, however, alumina is produced together and simultaneously with titanic acid, the two react together at once, forming aluminium titanate, which is much more easily fusible than titanic acid and also more easily fusible than alumina.

Aluminium, used to such a large extent in the steel industry,

has also the disadvantage that its oxide alone is fusible only with great difficulty and does not separate out easily. By microphotographic methods it is easy to show the small particles of alumina enclosed in steel.

A simple experiment shows that aluminium titanate gives a much more fusible slag than either titanic acid alone or alumina alone. Let us try to melt pure alumina alone and pure titanic acid alone in a piece of clay pot or in a small shallow crucible by means of the oxy-acetylene flame. Either oxide alone will refuse to melt, except if the flame is applied for a very long time and concentrated directly on the oxide, but even then we will succeed only in melting together small globules.

But if we now mix the two oxides together, say in the ratio of three to four parts of titanic acid to one part of alumina (that is, in same proportion in which the two oxides are formed in the oxidation of the carbon-free ferrotitanium, containing aluminium and made by the aluminothermic method), the two oxides together will melt in a very short time. This simple experiment shows clearly the advantages of the aluminium content in titanium.

But there is another important point. Aluminium is a more strongly deoxidizing reagent than titanium. It is wrong to say that titanium is a better deoxidizer than aluminium. This is proven by the fact that aluminium reduces titanic oxide and titanium does not reduce alumina.

The use of the titanium has other objects than the aluminium. While the titanium is a strong deoxidizer, it also reacts with the nitrogen and is effective in small quantity as titanium itself.

When, however, the steel contains quantities of oxides which could be deoxidized by means of aluminium, if this was present, the titanium acts as a deoxidizer, and is consumed uselessly as such wholly or partly. In order to avoid this useless consumption of the titanium aluminium is added simultaneously with titanium, so that all the impurities and oxides are first reduced by the aluminium. When this is done the titanium can manifest its own effect more strongly, and *experience* has shown that to produce the same results *smaller quantities* of ferrotitanium containing aluminium are required than of ferrotitanium containing carbon.

The above-mentioned difficulties with alumina alone in steel have not escaped other experimenters who have tried to avoid and overcome them. I call attention to English patent 6323 of the year 1895, relating to the use of an alloy of manganese, silicon and aluminium, instead of pure aluminium. It is emphasized there that this alloy in statu nascendi yields an easily fusible slag. This alloy is made by the world-renowned firm of Friedr. Krupp in Essen, who are the owners of the above patent.

HANS GOLDSCHMIDT.

Essen-Ruhr, Germany.

Ferrotitanium for Rails and Rolls.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The article in your July issue, page 342, by Dr. Hans Goldschmidt is as usual of interest to all who have undertaken to use titanium as a purifier in iron or steel, but I take exception to that part in which the author attributes the success of titanium in rails and rolls to a carbonless alloy.

Statistics compiled by the American Iron and Steel Institute show that 195,940 tons of ferrotitanium rails were rolled during the year 1910. The records of the Titanium Alloy Manufacturing Company show that we shipped a quantity of titanium alloy with carbon to the great steel mills of this country for a considerably greater tonnage than has reached the attention of the Institute.

There are in use in this country to-day very nearly 500,000 tons of titanium rails for which this company has furnished the alloy. Orders on our files indicate that every rail mill in the United States but one purchases the titanium with carbon alloy made by this company and the one exception is a company which assures us that they have not yet used any ferrotitanium beyond the samples furnished by this company.

Orders for titanium alloy made by us have run as high as 900,000 lb. in a single instance with dozens of orders varying from a carload up to 14 carloads as a maximum, the 900,000-lb. order having required 18 cars of 50,000 lb. capacity.

Under these circumstances it seems evident that a free-from-carbon alloy with its necessarily higher price and its greater tendency to cause brittleness, both due to the very considerable aluminium content cannot possibly have been an important factor in the manufacture of steel rails.

We have proportionately strong evidence on the subject of chilled rolls, but there are more manufacturers, so that of this line we cannot speak quite as positively.

CHARLES V. SLOCUM.

Titanium Alloy Mmanufacturing Company, Pittsburgh, Pa.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir: With reference to the letter from Mr. Charles V. Slocum on the subject of ferrotitanium for rails and rolls, I would like to state on behalf of Dr. Hans Goldschmidt, who is at present in Germany and therefore not in a position to write a prompt reply on the subject, that I am quite sure that he had no idea of attributing the success of titanium in rails and rolls only to the carbonless alloy.

What he desired to convey was the fact that for this class of work the carbonless alloy possesses very many advantages over a ferrotitanium containing carbon. As he has brought these advantages out in a reply which he has written to Dr. Waterhouse, it is hardly necessary for me to repeat them.

I would like to say further that in Dr. Hans Goldschmidt's article entitled "Melting Point and Its Relation to Alloying Capacity," published in METALLURGICAL AND CHEMICAL ENGINEERING July, 1911, his remarks were entirely general and scientific and related to his personal experience with the carbonless alloy. No one would be more ready than Dr. Hans Goldschmidt to accord Mr. Slocum credit for his energy in introducing his alloy to the steel trade, but his apparent intention to draw the conclusion that the carbonless alloy has found no field of usefulness in the lines he mentions is hardly correct.

WILLIAM C. CUNTZ.

Goldschmidt Thermit Company, New York City.

The Oil Engine for Electrochemical Plants.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—In your August editorial on "the power problem for electrochemical plants" you have not spoken of the cheapest one of all, namely the oil engine. I have long ago abandoned steam wherever practicable: and the gas engine I have also laid aside after thirty-five years' use, for I find the new F H oil engine built by the De La Vergne Company beats even the gas engine, for with oil at 2.25 cents per gallon, a kw-hour can be obtained for $0.005 and less, and with less labor and trouble than gas.

How dreadfully slow people are to take up new ideas even when based on scientific principles, to wit: in 1876 when the late Sir W. Siemens first melted steel electrically he said to me, "like everything else it will take a generation before this is brought into practical use"; and this is generally true, for I can point out numerous great ideas which took over thirty years to come into use.

In this western country there are natural conditions which would not lie undeveloped a day in Germany, but I have tried in vain to interest Eastern capital. The answer always comes back: "We know nothing of that kind of business." Oh! for a firm out here like Siemens & Halske. There would be something doing.

WALTER E. KOCH.

El Paso, Tex.

Recent Ore Dressing Practise at Silverton, Col.

Editorial Correspondence.

The rugged mountainous district of southwestern Colorado is generally spoken of as the San Juan region, or simply "the San Juan." It is divided naturally by the mountain ranges into districts which are tributary to the important towns of the region, of which Silverton is one of the most prominent. The winning of gold and silver from the mountains in the immediate vicinity of Silverton dates back to 1860, when gulch mining was first attempted by a party of miners headed by John Baker. But it was not until the early eighties that the deposits of rich silver ore in the vicinity of Red Mountain were discovered and successfully worked. At that time ore worth less than $100 per ton could not be marketed at a profit on account of the difficulties of transportation. With the extension of the Denver & Rio Grande Railroad into Silverton in 1882 an impetus was given to the mining and smelting of lower grade ores, but with the waning price of silver the precious metal content of the ores still had to be comparatively high in order to yield a profit.

Crude ore only was shipped from this district up to 1890, when pioneer attempts at concentrating the lower grade ores were made at the Silver Lake and Sunnyside mines. Prior to this time lixiviation plants had in a few instances been successful in extracting silver by a process of chloridizing, roasting and leaching with strong brine.

At the present time, however, attention is being directed to the concentration of the low grade milling ores, which are abundant in the district. These are mixed sulphides of lead (galena), zinc (sphalerite), copper (tetrahedrite and chalcopyrite) and iron (pyrite). Silver occurs mainly with the copper ores and sometimes with the finely crystalline galena. Gold accompanies the pyrite and sphalerite in small quantities. Zinc minerals almost always accompany galena, and in the past have given much trouble in concentrating. In brief, operations in the Silverton district are at present confined almost wholly to the separation of these low grade mixed sulphides into marketable products of lead and zinc, and the future of the mining industry there practically depends on the use of the most modern and improved methods of ore dressing.

The presence of considerable quantities of copper and iron minerals with the lead and zinc complicates wet dressing and calls for accessory treatment such as electrostatic separation. The latter process is not now in use in the district and as a consequence many mines and mills are closed, and only those are operating which can produce marketable lead and zinc concentrates by straight water concentration. At present these are the Silver Lake, Vermillion, Ledge, Iowa and Sunnyside. The Vermillion and Ledge mills may be taken as typical of the best ore dressing in the district.

The Vermillion Mill.

The concentrating mill of the Vermillion Mining Company is situated about 1½ miles west of Animas Forks in California Gulch, near the head waters of the West Fork of the Animas River. The crude ore is a mixture of galena, sphalerite and pyrite finely disseminated in a gangue of rhodonite. An average assay will show about 5 to 6 per cent lead, 10 per cent zinc and 3 to 4 per cent iron. The ratio of concentration is about 5½ to 1, and for the month of July, 1911, the saving was 81 per cent. At present the mill is running below capacity and treating only 75 tons per day. Water is obtained partly from the mine, about one-third the total quantity used being taken just as it flows from the tunnel. The balance is pumped from the creek 146 ft. below by a 7x8 Gould triplex pump operated by a 15-hp motor. Pumping is apparently a necessary expense, as the mill is situated so near the head waters of the creek that a pipe line or flume run at the proper level would not collect a sufficient volume of water. From 12 to 13 tons of water are used per ton of ore treated.

The method of treatment follows: Ore is trammed from the tunnel to the head of the mill and dumped into crude ore bins, first passing over a ⅞ inch grizzly. The coarse ore is reduced to ⅞-inch size in a Blake crusher, 11x18 in., and added to the grizzly undersize. The bin capacity is sufficient for the storage of 200 tons of crushed and an equal tonnage of uncrushed ore. From the second bin the ore is drawn by a plunger feeder and belt conveyer to the roughing rolls. These are 14x27 in., running at 60 r.p.m. and set ½ in. apart. The main elevator, which delivers the roll discharge to the trommel screen line, has 55-ft. centers and 14-in. buckets on a 16-in. belt.

A single line of trommels covered with Tyler double crimped screen prepares the feed for jigging. The first screen is 2-mesh, the oversize of which is reground in a set of 14x27 in. rolls running at 125 r.p.m. and set ⅛ in. apart. This roll product is again elevated to the screens. The second and third screens are 6 and 14-mesh, respectively, and their oversizes are treated in 3-compartment Harz jigs. Owing to the abrasive action of the gangue, screens are rather short lived in this mill. Trommel screens last about three weeks, and it is probable that punched steel screens will be used in place of wire, as local experience indicates that the life of the latter will be about three months. With punched screens the pitch of the trommels will be changed from 1 in. per foot to ¾ in.

Jigging practise varies with the physical condition of the ore. At times the mineral does not occur coarse enough to warrant the use of the coarse jig. With both jigs running, however, the first receives a feed ranging from —2 to +6 mesh, and the second from —6 to +14 mesh. The coarse jig has 5-mesh screen in the first compartment, and 6-mesh in the second and third. It runs at 210 r.p.m. and has a stroke of ⅝ in. The fine jig has 8-mesh screen in all compartments, runs at 235 r.p.m. with a ⅜ in. stroke.

Both jigs make a clean lead concentrate in the hutch and side gate of the first compartment. A shipping grade of zinc concentrate is made in the third compartment of the fine jig only. The middlings and tailings of the coarse and the tailings of the fine jig are reground in a Chile mill, while the middlings of the fine jig are returned to its first compartment for retreatment.

One 6-ft. Chile mill is used for regrinding jig tailings and middlings. This machine is grinding about 40 tons per day through ton-cap screen corresponding to 14 mesh. The life of the screen is but 10 days due to the excessive abrasion of the rhodonite. The die ring is 4½ in. thick, while the roll tires are made 3½ in. thick on the outside and 3¼ in. on the inside. These dimensions are calculated to give about equal life to die and tires. Under the light feed at present given to the mill it is run at only 24 r.p.m.

The Chile mill product is elevated and combined with the undersize of the last, or 14-mesh trommel, and thickened in a 3 x 3 ft. dewatering box preliminary to classification in a Richards 4-in., 6-compartment pulsator classifier. The overflow of the dewatering box flows successively to two Callow tanks, each of which gives a thickened feed to one Card table. The overflow of the second Callow tank runs to waste.

Each of the six products from the classifier is treated on a separate Wifley table. The first three tables make high grade lead and zinc concentrates, a two-inch streak of mixed lead and zinc mineral being returned to the tables by raff wheels. The fourth, fifth and sixth tables give a finished lead product, but the zinc mineral from these is combined with the zinc from the Cards treating the Callow product and retreated on another Wilfley. The coarse middling from the first Wilfley is reground in the Chile mill and the middlings from the succeeding five Wilfleys are combined and treated as a mixed feed on a separate table. This last table also has a raff wheel for the return of its middling product. The slime water from all tables except the zinc retreating table is thickened in three Callow tanks and retreated on a Card table.

Table Speeds and Strokes.

	Speed r.p.m.	Stroke in.
No. 1 Wilfley	245	⅞
Nos. 2 and 3 Wilfley	250	¾
Nos. 4, 5 and 6 Wilfley	255	⅝
Midd. Wilfley	250	¾
Cards	260	9/16

An interesting feature of the table work at this mill is the manner of running the Wilfley tables adopted by Louis D. Bastian, mill superintendent. When the tables were run flat enough to bring all mineral—galena and sphalerite—to the head of the table in the usual manner more or less of the rhodonite gangue appeared scattered throughout the concentrates and the zinc separation especially was poor. Mr. Bastian found that by increasing the pitch of the table so as to bring the zinc mineral over the side, and only the lead over the head of the table, he could increase the grade of zinc concentrate from 37% to 43%, and at the same time throw the rhodonite out of the mineral. Additional wash water is used near the head of the table to assist in washing the rhodonite out of the concentrates, and under present conditions this coarse gangue appears as a narrow streak below the zinc.

Ultimately all concentrates of like character from jigs and tables are mixed for shipment. The average grade of these products is indicated in the following figures:

	Pb %	Zn %	Fe %	SiO_2 %
Lead Conc.	57-62	7	7	2
Zinc Conc.	8	43	7	6-7

Of the iron in the zinc concentrate a little over half is combined in a manner which precludes separation, but the iron in the lead concentrate represents practically free pyrite. Electrostatic separation, therefore, would be of little benefit to this ore, as the added cost of further cleaning the zinc mineral would hardly be repaid.

The Vermillion mill is operated by electric power supplied by the San Juan Water and Power Company. The distribution of motors in the mill follows: 25 hp for crusher; 75 hp for rolls, trommels and elevator; 30 hp for Chile mill; 20 hp for 10 tables, classifier, two centrifugal pumps and middlings elevator; 15 hp for pump. The total power requirement under running load is 90 hp.

Acknowledgment is made to Mr. L. D. Bastian, mill superintendent, for assistance in collecting the data given and to Mr. R. W. Hollis, manager, for permission to publish the same.

The Ledge Mill.

The ore of the Ledge Consolidated Mining Company at Chattanooge, near Silverton, differs decidedly from that of the Vermillion as to the nature of the gangue, although the mineral content is not much different. Instead of the hard rhodonite gangue noted in the Vermillion ore, that of the Ledge is kaolinite, soft enough to be scratched by the finger nail and making an excessive quantity of slime in the milling process. The minerals are galena, sphalerite and pyrite, the percentage of lead being about 10 and of zinc 12. The ore also carries about 3 oz. silver and 0.02 to 0.03 oz. gold per ton. About 150 tons are treated daily. The ratio of concentration is 5 to 1.

The ore as mined is delivered to the mill by an aerial tramway one-half mile long. The buckets have a capacity of about ⅓ ton each, and the daily tonnage can generally be delivered on one shift. The ore breaks easily and preliminary crushing is rather coarse to avoid the production of slime. The first break is made to about 1½ in. size in a Blake 10x20 in. crusher. This product is further reduced in rolls, 16x36 in., running at 60 r.p.m. and set ¾ in. apart. Water is added at the rolls and the pulp is elevated to the trommel screen line. The elevator has 64 ft. centers and 14 in. buckets on a 16-in. belt. Ample screening capacity is one of the features of the mill, there being a double line of four trommels, each 9 ft. long and 42 in. in diameter, fitted with ton-cap screens corresponding to 4, 6, 8 and 16 mesh respectively. The oversize of the 4-mesh screen is reground in a set of 14x27 in. rolls, running at 100 r.p.m. and set ¼ in. apart and returned to the elevator and screen line. The oversizes of the second, third and fourth trommels comprise the feed for three 3-compartment Harz jigs, and the undersize of the last screen is thickened and classified for table concentration.

The following tabulation gives data regarding the operation of jigs:

	Size of feed Mesh	Screen Mesh	Speed r.p.m.	Stroke in.
Jig No. 1	−4+6	5	180	1¼
Jig No. 2	−6+8	8	220	⅞
Jig No. 3	−8+14	10	240	⅝

Galena concentrate is drawn from the first compartment in all jigs and from the second compartment also of the third jig. No zinc concentrate is made in the jigs. All jig middlings and tailings are dewatered in a trommel and collected in a bin, from which they are fed to a Chile mill. The water recovered from this pulp is stored in a tank and used in the Chile mill as needed. This procedure is preferable to allowing the pulp to flow from the jigs to the mill without dewatering, as a better ratio of water to pulp can be maintained.

The Chile mill runs at 32 r.p.m. and grinds about 75 tons per day through 16-mesh ton-cap screen. These screens last about three weeks. The die ring is cast thicker than the roll tires, so that all have about the same life, viz., three months.

The performance of the Chile mill at this plant emphasizes the difference in the physical properties of the Ledge and Vermillion ores. The quantity reground at the Ledge through 16-mesh screen is almost twice that reground at the Vermillion through 14-mesh, and yet the screens at the Ledge have about twice the life of those at the Vermillion, where the gangue is rhodonite.

The Chile mill product is elevated and combined with the undersize of the 16-mesh trommel, flowing to a spitzkasten, where it is thickened before classification. On account of the large quantity of slime in the pulp which tends mechanically to carry fine mineral to waste, extra large settling capacity is provided so that all slime is ultimately thickened and concentrated. To this end the overflow of the spitzkasten flows successively to Callow tanks, sufficient in number so that the overflow of the last will be practically clear. The underflow of these Callows is combined and classified in a Richards 4-compartment pulsator classifier and treated on four tables. The thick pulp from the spitzkasten is classified in a Richards 6-compartment pulsator classifier, running under a 16-ft. head of hydraulic water and making 160 pulsations per minute. A hutch product of clean galena is made in this machine, amounting to one ton per day.

The Wilfley tables in this mill are arranged on two floors, ten on each floor. Of those on the upper floor four treat the classified fine material from the Callow tanks and six receive the different grades of coarse pulp. All tables on this floor make a finished lead product only. The zinc-iron middling from each of these tables is retreated on a complementary table on the floor below, where zinc concentrates are separated from the iron and small amount of lead remaining with it. By way of further recovering fine mineral now going to waste it is planned to collect the slime water from the tables on the upper floor, thicken it and reconcentrate on additional tables. It is planned also to regrind the coarser middlings from some of the upper tables. When this plan is in operation the only material going to waste will be the tailings from the lower tables and the overflow from Callow tanks.

All concentrates are laundered to boxes at the foot of the mill and dewatered by a screw conveyer working in an inclined launder at one end of the box. The overflow from the

box passes through a series of settling launders before leaving the mill. The lead concentrates as mixed for shipment assay from 55 to 60 per cent. lead, and the zinc concentrates carry from 40 to 45 per cent. zinc.

During the summer season the mill is operated by water power. Two Pelton wheels running under a head of over 800 feet of water furnish all the power required. One 36-in. wheel, with 3/8-in. nozzle, provides power for the concentrating tables only, thus assuring steady operation. The balance of the mill is run by a 56-in. wheel, 3/4-in. nozzle. About 150 hp. is required for the whole plant under running load, or one hp. per ton of ore treated per day. Electric power is available when water is scarce, and only a short time is required to make the change.

For the data and information given herewith acknowledgment is made to Mr. W. Z. Kinney, manager, and Mr. D. C. McNaughton, mill superintendent.

Cost of Alkali Chloride Electrolysis.*

By Victor Engelhardt.

Aside from the electrolytic production of chlorates, there are two main groups of commercial processes for chloride electrolysis:

(a) Direct Production of Hypochlorite Solutions.—For this purpose the products of the electrolysis, chlorine and caustic alkali, are directly combined in the electrolytic cell itself to form bleaching liquor (hypochlorite). There is, therefore, only one final product. Such installations are suitable only in works in which hypochlorite solutions are directly needed.

Since in this case no saturated solutions are employed and the temperature is kept relatively low (as otherwise chlorate would be formed), the power consumption is comparatively high. Further as the ampere-hour efficiency decreases with increasing amount of active chlorine in the hypochlorite solution, it is impractical to decompose the whole chloride so that part of it is wasted.

On the other hand, these installations have the advantage that they need hardly any attendance and that any number of cells can be connected in series so as to suit any given supply voltage. For this reason the cells may be connected to dynamos which are also used for other purposes; for instance, direct-current dynamos which supply the lighting current during the night, may be connected to the bleaching liquor cells during the day.

This last-named practice is especially suitable for the cotton industry and for laundries, where the amount of hypochlorite solution required is relatively small. The electrolytic cells, when operated only during the day, will therefore furnish sufficient chlorine for the requirements of the works, especially as for cotton there is a relatively large saving of active chlorine, if electrolytic hypochlorite solutions are used, as compared with bleaching powder.

On the other hand, in the linen industry and in paper and pulp mills much greater quantities of hypochlorite solution are required, so that in this case it pays in general to install a special dynamo for the electrolytic plant, especially as the latter may then be made of 50 per cent less capacity, being operated continuously day and night. Moreover, in this case there is not such a large saving of active chlorine for hypochlorite solutions as compared with bleaching powder, so that for this reason alone the quantity of hypochlorite solution needed is relatively large.

(b) Production of Chlorine and Caustic Alkali Separately.—To keep the chlorine and caustic alkali separate from each other cells of three different types may be used: (1) Diaphragm cells, yielding chlorine gas and a caustic solution containing some undecomposed chloride, which can be removed completely only with some difficulty during the further treatment; (2) gravity cells (Glockenverfahren) yielding also chlorine gas and generally a caustic solution containing chloride; (3) mercury cathode cells, yielding chlorine gas and a caustic solution free from chloride.

In all processes for the separate production of chlorine and caustic alkali a hot concentrated electrolyte is employed. From the caustic solution produced it is possible in diaphragm and gravity processes to recover the undecomposed chloride during the further treatment; this is unnecessary in mercury-cathode processes. The power consumption is, therefore, lower and the utilization of the chloride is more complete than in the direct electroytic production of hypochlorite.

For the purposes of this article, which compares the different processes from the standpoint of commercial efficiency, the diaphragm and gravity processes may be grouped together, while the situation may be somewhat different, according to the conditions, for mercury-cathode processes. The latter produce a purer caustic alkali, free from chloride, such as needed for some purposes like the manufacture of metallic sodium or potassium, and which may be sold sometimes at a higher market price. As an offset there are higher capital investment and also mostly higher power consumption, due to the use of higher voltages.

On a large commercial scale processes of all three groups are employed for the production of chlorine and caustic alkali. For industrial works which need chlorine and caustic alkali for their own purposes, diaphragm cells are probably most suitable, since they are simpler and of less delicate construction. Such industrial works may proceed further by either of the following two methods:

(1) If the producer has no use for caustic alkali and no facilities for marketing it, he can combine the chlorine and the caustic outside of the electrolytic cell and produce alkali hypochlorite.* From a commercial standpoint, this method has the advantages of better utilization of the sodium chloride and smaller power consumption (due to both higher ampere-hour efficiency and smaller voltages) over the direct production of hypochlorite in an electrolytic cell.

(2) If the producer can use himself the caustic alkali or if he can market it readily, it is preferable from a commercial standpoint to combine the chlorine gas not with the valuable caustic soda or potash, but with milk of lime (calcium hydroxide) and form a chloride of lime solution.

Installations for the separate production of caustic soda and chlorine in industrial works are suitable only for relatively large capacities, since in order to reduce the relative cost, the apparatus must be built for high current strengths and a sufficient number must be connected in series. Moreover they must be operated continuously day and night, they require their own dynamo, attendance, etc., and would, therefore, be uneconomical for small capacities.

The power cost is of fundamental importance in chloride electrolysis since it is the most important item in the cost sheet. We will assume two limiting values: a minimum value of 0.5 cent per kw-hour for waterpower plants and by-product gas plants, and a value of 1.25 per kw-hour for steam plants.

On the basis of the above discussion the different cases to

*Translated from *Chemiker Zeitung*, Vol. 35, No. 64 and 65, p. 573 and 582 (1911). Slightly abstracted.

*This method is, of course, possible only if what the producer needs in his works, is bleaching liquor. If he needs chlorine gas, this method cannot be used. At the plant of the U. S. R. & R. Co. at Colorado Springs, Colo., where chlorination of gold ores with cyanidation of the tailings is in very successful operation, the chlorine is produced by a diaphragm-cell process and the caustic soda solution is wasted, since there is not such a ready market for caustic soda in the neighborhood as to justify the erection of expensive and troublesome evaporation apparatus. While the waste of the caustic soda seems peculiar from the standpoint of the Eastern producer of caustic soda and chlorine, it is undoubtedly the logical thing to do under the existing conditions. That the chlorine gas can be produced at very much less cost by electrolysis than from bleaching powder, bought in the market, and sulphuric acid, was proven by extensive tests at this plant, using both methods side by side over a considerable period of time.—Editor.

be treated and compared with each other, may be grouped as follows:

I. Direct production of hypochlorite solution at the consumers works.

IA. Smaller installations (cotton industry), operated only during the day. A saving of 30 per cent of active chlorine, as compared with the use of bleaching powder, is assumed.

IB. Larger installations (linen and paper industry), operated continuously during day and night. A saving of 10 per cent of active chlorine, as compared with the use of bleaching powder, is assumed.

II. Separate production of chlorine and caustic alkali.

II. 1. At the consumer's works.

II. 1A. With recombination of chlorine and alkali, to form alkali hypochlorite. Continuous operation day and night.

II. 1B. With production of chloride of lime solution and separate utilization of the caustic alkali. Continuous operation day and night.

II. 2. Separate production of chlorine and caustic alkali on a large scale for selling both products.

II. 2A. Diaphragm processes with production of a caustic alkali containing undecomposed chloride. Continuous operation day and night.

II. 2B. Mercury-cathode process with production of a caustic alkali free from chloride. Continuous operation day and night.

In order to arrive at a fairly satisfactory comparison between the different processes of alkali chloride electrolysis, attention must be paid to the fact that in the separate production of caustic and chlorine these two products are obtained simultaneously from the same raw material so that it is not possible to charge a definite proportion of the cost of operation against the manufacture of either product. On the other hand in hypochlorite manufacture the bleaching liquor solution only is produced.

For this reason, in considering processes for the separate production of chlorine and caustic alkali it will be more correct to deduct the value of the caustic alkali produced from the total cost of operation and to consider the difference as the cost of the bleaching material produced.

Since in industrial plants sodium chloride will probably always be used, it will be sufficient to give the comparative calculations for sodium chloride only. Further it would be wrong to assume always the same values for cost and selling price of sodium chloride, calcium chloride, and caustic soda, since smaller consumers cannot get as favorable prices as large consumers.

I.—Direct Production of Hypochlorite at the Consumer's Works.

In the main the following three electrolytic hypochlorite cells are now in commercial use, differing from each other by the material used for the electrodes: *a*, Siemens & Halske and Kellner, both electrodes being platinum. *b*, Schuckert, one electrode being of carbon, the other of platinum. *c*, Haas and Stahl, both electrodes being of carbon.

The following calculations are based on the system of Siemens & Halske and Dr. Kellner, with which the author is most familiar.

IA: Hypochlorite production at the consumer's works in small installations of the cotton industry, electrolysis being carried out during the day only.

Assumptions: The electrolytic cell is operated with 120 amp. at 110 volts (the power being, therefore, 13.2 kw), for 10 hours a day. 22.2 kilograms of active chlorine daily, 300 days of operation during the year. $28.75 per 1000 kilograms of chloride of lime at the plant, $3.75 per 1000 kilograms of sodium chloride at the plant. A saving of 30 per cent active chlorine compared with chloride of lime with 35 per cent Cl. (Therefore, about 25 kg of active chlorine in a NaOCl solution correspond to about 100 kg of chloride of lime.)

First Cost:	
Platinum electrodes	$1,350
Accessories (cells, pump, etc.)	1,150
Total	$2,500

	Cost of kw-hour. 0.5 cent.	1.25 cent.
Yearly Cost of Operation:		
13.2×10×300 = 39,600 kw-hours	$198.00	$495.00
170×300 = 51,000 kg Na Cl at $0.38 per 100 kg	191.25	191.25
4% amortisation and maintenance of platinum	54.00	54.00
10% amortisation and maintenance of accessories	115.00	115.00
5% interest on accessories	57.50	57.50
	$615.75	$912.75
Under the assumption of 30% saving of active chlorine, as stated above, the amount of bleaching liquor produced per year is equivalent to 26,700 kg of chloride of lime; hence the cost of the ton of equivalent chloride of lime	$23.05	$34.18

If for small quantities of chloride of lime the ton is bought at the consumer's works at $28.75 as assumed above, the production of the bleaching liquor by electrolysis at the works results, therefore, in a saving of $5.70 per ton of equivalent chloride of lime, if the kw-hour costs 0.5 cent, and in a loss of $5.43, if the kw-hour costs 1.25 cent. Under the above assumptions there will be neither a saving nor a loss if the kw-hour costs about 0.875 cent. In other words, if the power costs less than 0.875 cent per kw-hour, the electrolytic production of bleaching liquor at the works will be less expensive than buying bleaching powder.

It should be emphasized, however, that in this comparison the well-known technical advantages of electrolytic hypochlorite solutions could not be considered, because they cannot be given in figures. Further the present very high market price of platinum must be taken into consideration.

The wages are not considered in calculating the cost of operation since they would represent at least the same amount in bleaching with bleaching powder for the preparation of the bleaching solution, etc.

IB: Hypochlorite production at the consumer's works in larger installations in the paper and pulp industry, with continuous operation day and night.

Assumptions: Electrolytic cells operated at 110 volts, 600 amp (66 kw), for 23 hours a day, 254 kg of active chlorine daily, 300 days of operation during the year, $26.25 per 1000 kg of chloride of lime at the plant, $3.75 per 1000 kg of sodium chloride at the plant. A saving of 10 per cent of active chlorine as compared with the use of chloride of lime with 35 per cent Cl. (About 30 kg of active chlorine in NaOCl solution are therefore equivalent to 100 kg of commercial bleaching powder).

First Cost:	
Platinum electrodes	$6,750
Accessories (as in I A)	5,000
Total	$11,750

	Cost of kw-hour. 0.5 cent.	1.25 cent.
Cost of Operation:		
66×23×300 = 455,400 kw-hours	$2,277	$5,692.50
1900×300 = 570,000 kg sodium chloride at $3.75 per 1000 kg	2,137.50	2,137.50
6% amortisation and maintenance of platinum	405	405
10% amortisation and maintenance of accessories	500	500
5% interest on accessories	250	250
Total	$5,569.50	$8,985
Under the assumption of 10 per cent. saving of active chlorine, as stated above, the yearly amount of bleaching liquor produced is equivalent to 255 tons of chloride of lime; hence the cost of the ton of equivalent chloride of lime	$21.83	$35.23

Larger installations of this kind can buy the chloride of lime at less cost than assumed before, say at about $26.25 per ton at the works. Then the production of bleaching liquor at the works by electrolysis results in a saving of $4.42 per ton of equivalent chloride of lime if the kw-hour costs 0.5 cent, and in a loss of $8.08 per ton if the kw-hour costs 1.25 cents. Under the above assumptions there will be neither a saving nor a loss if the kw-hour cost 0.75 cent. For power costs be-

low this rate the electrolytic manufacture of bleaching liquor will be less expensive than buying bleaching powder.

With respect to the technical advantages of electrolytic bleaching liquor and wages the same remarks may be made here as were made in IA.

II.—Separate Production of Caustic and Chlorine at the Consumer's Works.

Only diaphragm cell processes are here considered. The cell should be of compact construction and it should be possible to build and operate it economically in relatively small capacities. A cell which has proven successful on a large scale in the chemical manufacturing industry as well as in plants of smaller capacity is the Billiter cell* which has been developed by the Siemens & Halske Company, together with the Kaliwerke Aschersleben. It is probably the latest diaphragm cell which has proven successful in practice. The following figures relate to this cell.

II.—1A: Separate production of chlorine and caustic with subsequent recombination to hypochlorite, with continuous operation in paper and pulp mills.

Assumptions: Electrolytic cells operated with 500 amp at 120 volts (60 kw), 8 kw for other purposes (hence total power 68 kw), 24 hours per day, 337 kg of active chlorine in the hypochlorite (figuring a loss of 10 per cent in absorption, etc.). Eight thousand hours of operation per year, $26.25 per 1000 kg of chloride of lime at works, $3.75 per 1000 kg of sodium chloride. A saving of 70 per cent of active chlorine as compared with the use of bleaching powder of 34 to 35 per cent Cl. (About 337 kg of active chlorine in NaOCl solution, are, therefore, equivalent to 1100 kg of chloride of lime).

First cost:	
Electrolytic cells	$7,000.
Conductors and switchboard	525.
Absorption plant and pipe system	1,562.50
Miscellaneous	912.50
Building, foundations	$10,000.
Concrete tanks	7,500.
Total	$17,500.

	Cost of kw-hour	
Cost of operation:	0.5 cent.	1.25 cent.
68x8000 = 544,000 kw-hours	$2,720.	$6,800.
350,000 kg. sodium chloride	1,312.50	1,312.50
Wages 5 men at $300	1,500.	1,500.
10% amortization and maintenance on $10,000 (electrodes, diaphragms, etc.)	1,000.	1,000.
3% amortization of $7,500	225.	225.
5% interest on $17,500	875.	875
Total	$7,632.50	$11,712.50
Under the assumption of 10% saving of active chlorine, the hypochlorite produced is equivalent to 366 tons of chloride of lime per year; hence the cost of the ton of equivalent chloride of lime is	$20.85	$32.

As in IB, the price at which chloride of lime can be bought may be assumed here as $26.25 per ton at the works, so that the production of bleaching liquor at the works by electrolysis results in a saving of $5.40 per ton of equivalent chloride of lime, if the kw-hour costs 0.5 cent, and in a loss of $5.75 per ton, if the kw-hour costs 1.25 cents. There will be neither a saving nor a loss if the kw-hour costs 0.85 cent. For power costs below this rate the electrolytic manufacture at the works will be less expensive than buying bleaching powder.

The use of hypochlorite solutions produced in this way also involves technical advantages. The expense for license for using the patented process is not included in the operating cost in this and in the following cases, since it is not possible to give for this a generally valid figure.

II.—1B: Separate production of chlorine and caustic with subsequent production of chloride of lime solution and use of the caustic soda, with continuous operation in paper and pulp mills.

Assumptions: Electrolytic cells operated with 500 amp at 120 volts (60 kw), 12 kw for other purposes (hence total power 72 kw). Twenty-four hours' operation per day, 337 kg of active chlorine in the bleaching solution (figuring a loss of 10 per cent in absorption, etc.). Eight thousand hours of operation per year, $26.25 per 1000 kg of chloride of lime at works; $3.75 per 1000 kg of sodium chloride at works; $5 per 1000 kg of burned lime, $4.50 per 1000 kg coal (evaporating seven times its weight of water), if bought in relatively small quantities.

The caustic soda is figured at only $3.50 per 100 kg NaOH in form of a solution of 35° Be. on account of the higher selling expenses with such comparatively small plants. There is no saving of active chlorine in bleaching. (Hence 337 kg of active chlorine in CaO_2Cl_2 solution are equivalent to 1000 kg of chloride of lime.) Partial recovery of the undecomposed sodium chloride during evaporation.

First cost:	
Electrolytic cells	$7,000.
Conductors and switchboard	525.
Absorption plant and pipe system	1,562.50
Evaporation plant and tanks	2,000.
Miscellaneous	912.50
	$12,000.
Buildings, foundations and concrete tanks	8,000.
	$20,000.

	Cost of kw-hour	
Cost of operation:	0.5 cent.	1.25 cent.
72x800 = 576,000 kw-hours	$2,880.	$7,200.
250,000 kg. sodium chloride	937.50	937.50
200,000 kg. lime	1,000.	1,000.
166,00 kg. coal	750.	750.
Wages, 6 men at $300	1,800.	1,800.
10% amortization and maintenance on $12,000	1,200.	1,200.
3% amortization on $8,000	240.	240.
5% interest on $20,000	1,000.	1,000.
	$9,807.50	$14,127.50
Less selling price of 140,700 kg. of caustic soda	4,925.	4,925.
Cost of 333 tons of equivalent chloride of lime	$4,882.50	$9,202.50
Cost of 1 ton of chloride of lime	14.68	27.65

The price at which chloride of lime can be bought may again be taken as $26.25 per ton, so that the electrolytic operation at the works in the present case results in a saving of $11.57 per ton of equivalent chloride of lime, if the kw-hour costs 0.5 cents, and in a loss $1.40, if the kw-hour costs 1.25 cents. There will be neither a saving nor a loss if the kw-hour costs 1.175 cents. For power costs below this rate the electrolytic manufacture at the works will be less expensive than buying bleaching powder.

Compared with the former examples, the power may cost considerably more if electrolytic manufacture at the works is to be profitable. This is due to the utilization or sale of the caustic soda.

II.—2. Large-scale separate production of chlorine and caustic soda for sale of the products.

Since in this case we have to do with large plants, relatively low prices may be assumed for the purchase of the raw materials. On the other hand a somewhat smaller selling price must be assumed for chloride of lime. The following figures relate to the Billiter cell for diaphragm-cell processes, while platinum anodes are assumed for the mercury-cathode processes, since figures from experience with the mercury-cathode process with carbon anodes are not yet available.

II.—2A: Diaphragm process with chloride of lime and solid caustic soda (containing some sodium chloride) as final products with continuous operation in large-scale plants of the chemical industry.

Assumptions: Power consumption: 1200 kw for electrolysis, 30 kw for chloride of lime apparatus, 30 kw for evaporating plant, 20 kw for the pumps, ventilators, etc., 70 kw for lighting and miscellaneous, hence total power consumption 1350 kw. Voltage 3.75, amp-hour efficiency 90 per cent, 24 hours of operation during the day, 8000 hours per year. Selling price of chloride of lime in large quantities $20 per ton. Selling price of solid caustic soda (95 per cent), $47.50 per ton, including packing. Purchase price of sodium chloride $3, lime $4.50, coal $4.50 per ton.

*See this journal, Vol. VIII., p. 704 (1910).

First cost.	
Electrolytic cells	$125,000
Copper conductors	13,500
Pipes	13,500
Chloride of lime apparatus	62,500
Evaporating plant	20,000
Melting pots	5,000
Screening and lime burning plant	3,000
Plant for purification of water, solution of salt and lye transport	5,750
Cranes and other transportation facilities	1,250
Packing material factory	4,000
Workshop, motors and gears	7,750
Sum A	$261,250
Lighting and heating plant	$4,500
Equipment of office and laboratory	4,000
Sum B	8,500
Buildings, C	87,500
Building erection, superintendence and miscellaneous, D	30,250
Working capital, E	37,500
Total	$425,000

(Without cost of land and railroad track connections.)

Cost of operation:		Cost of kw-hour 0.5 cent.	1.25 cent.
(a) Power:			
1350x800 = 10,800,000 kw-hours		$54,000	$135,000
(b) Raw materials:			
5505 tons of salt	$16,515		
6086 tons of lime	27,387		
6000 tons of coal	27,000		
	$70,902	71,000	71,000
(c) Renewal and maintenance:			
Electrodes	$14,000		
Diaphragms	2,000		
Electrolytic cells	1,250		
Chloride of lime plant	4,375		
Evaporating plant	2,000		
Melting pots	5,000		
Buildings	1,750		
Balance of equipment	4,375		
	$34,750	35,000	35,000
(d) Wages:			
58 men at $375	$21,750		
3 foremen at $750	2,250		
	$24,000	25,000	25,000
(e) Auxiliary materials:			
Oil, lubrication, belts		6,000	6,000
(f) Packing, excl., wages:			
Caustic soda	$7,000		
Chloride of lime	20,000		
	$27,000	27,000	27,000
(g) General:			
Wages	$6,250		
Office and laboratory	3,000		
Insurance	2,500		
	$11,750	11,750	11,750
(h) Capital amortization: Average, 7%, since all amortization has already been taken into account.		28,500	28,500
(i) Interest: 5% on $425,000		21,250	21,250
Total (not including patent license)		$279,500	$360,500
The yearly production of caustic soda is 3,560 tons (95%)		$169,100	$169,100
Balance for the yearly production of 8,690 tons of chloride of lime		$110,400	$191,400
Or per ton of chloride of lime		12.70	22.33

If, as assumed, the selling price of the ton of chloride of lime is $20, there is a profit of $7.30 per ton of chloride of lime for a power cost of 0.5 cent per kw-hour, and a loss of $2.33 per ton for a power cost of 1.25 cents per kw-hour, assuming an interest of 5 per cent only on the capital. For a power cost of 1.075 cents per kw-hour there will be neither profit nor loss, that is, just the 5 per cent of interest on the capital will be realized.

II.—2B: Mercury-cathode process, with chloride of lime and solid caustic soda (free from sodium chloride) as final products with continuous operation in large-scale plants of the chemical industry.

Assumptions: Power consumption 1600 kw for electrolysis, 150 kw for other purposes, together 1,750 kw. Voltage (at least) 5 volts. Ampere-hour efficiency, 90 per cent; continuous operation for 24 hours per day and 8000 hours per year. Selling price of chloride of lime in large quantities $20 per ton. Selling price of solid 95 per cent caustic is to be assumed higher than in II 2A, namely $50 per ton, including packing. Purchase price of sodium chloride $3, lime $4.50, coal $4.50 per ton.

First Cost		
Electrolytic cells without platinum and mercury	$62,500	.....
Copper conductors	13,500	.....
Pipes	13,500	.....
Chloride of lime plant	62,500	.....
Chloride of lime plant	20,000	.....
Melting pots	5,000	.....
Screening and lime burning plant	3,000	.....
Plant for purification of water, saturation, and lye transportation	9,000	.....
Cranes and other transportation facilities	1,250	.....
Packing material factory	4,000	.....
Workshop, motors, and gears	8,750	.....
Sum A	.....	$203,000
Lighting and heating plant	$4,500	.....
Office and laboratory	4,000	.....
Sum B	.....	8,500
Metals (mercury and platinum) C	.....	137,500
Buildings D	.....	87,500
Building erection superintendence and miscellaneous E	.....	26,000
Working capital F	.....	37,500
Total	.....	$500,000

(Without cost of land and railroad track connections).

Cost of operation:		Cost of kw-hour. 0.5 cent.	1.25 cents.
(a) Power:			
1,750 x 8,000=14,000,000 kw-hours		$70,000	$175,000
(b) Raw materials:			
5,505 tons of salt	$16,515	.....	.....
6,086 tons of lime	27,387	.....	.....
5,300 tons of coal	23,850	.....	.....
	$67,752	$67,750	$67,750
(c) Renewal and maintenance:			
Metals	$5,500	.....	.....
Electrolytic cells	1,250	.....	.....
Chloride of lime plant	4,375	.....	.....
Evaporating plant	2,000	.....	.....
Melting pots	5,000	.....	.....
Buildings	1,750	.....	.....
Balance of equipment	6,150	.....	.....
	$26,025	$26,000	$26,000
(d) Wages (as in II. 2 A)	.....	25,000	25,000
(e) Auxiliary materials (as in II. 2 A)	.....	6,000	6,000
(f) Packing (as in II. 2 A)	.....	27,000	27,000
(g) General (as in II. 2 A)	.....	11,750	11,750
(h) Capital amortization in the average 7%, with the exception of platinum and mercury which keep their value, hence on about $375,000	.....	26,250	26,250
(i) Interest of 5% on $500,000	.....	25,000	25,000
Total (not including patent license)	.....	$284,750	$389,750
Sales price of 3,560 tons of 95% caustic soda at $50 per ton	.....	178,000	178,000
Balance for yearly production of 8,690 tons of chloride of lime	.....	$106,750	$211,750
Or per ton of chloride of lime	.....	$12.28	$24.38

With a selling price of $20 per ton of chloride of lime and $50 per ton of caustic soda and an earning of only 5 per cent of interest on the capital investment, there is a profit of $7.72 per ton of chloride of lime for a power cost of 0.5 cent per kw-hour and a loss of $4.38 per ton for a power cost of 1.075 cents per kw-hour. For a power cost of 0.075 cent there will be neither profit nor loss, that is, just the 5 per cent interest on the capital investment will be realized.

II.—2. B[1]: If, however, with several large plants of this kind the caustic soda cannot be sold at $50 per ton, but only at $47.50 (as in II 2A), then the results are changed as follows:

	Cost of kw-hour. 0.5 cent.	1.25 cent.
Cost of operation, yearly	$284,750	$389,750
Selling price of 3,560 tons of 95% caustic soda at $47.50 per ton	169,100	169,100
Balance for yearly production of 8,690 tons of chloride of lime	$115,650	$220,650
Or per ton of chloride of lime	$13.33	$25.40

With a selling price of $20 per ton of chloride of lime and an earning of 5 per cent interest on the capital investment, there is a profit of $6.67 per ton of chloride of lime for a power cost of 0.5 cent per kw-hour, and a loss of $5.40 for a power cost of 1.075 cents per kw-hour. For a power cost of 0.925 cent per kw-hour there will be neither profit nor loss, that is, just the 5 per cent interest on the capital investment will be realized. It appears, therefore, that the mercury-cathode processes can complete with diaphragm processes only at very low power cost, say about 0.25 cent per kw-hour.

The results of the preceding seven cost calculations are summarized in form of the adjoining diagram. This diagram gives the cost of the production of the ton of chloride of lime

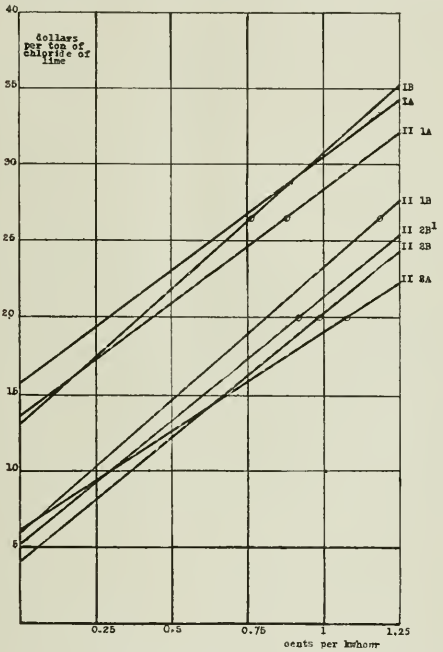
(or in cases IA, IB, and II 1A, of the equivalent ton of chloride of lime) as function of the power cost.

The ordinates give the cost of the ton of chloride of lime in dollars, the abscissas the cost of the kw-hour in cents.

The curves have been plotted by using the following figures which give the end results of the preceding calculations.

	0.5 cent.	1.25 cents per kw-hour.
I A	$23.05	$34.18 per ton of chloride of lime.
I B	21.83	35.23 " "
II 1 A	20.85	32.00 " "
II 1 B	14.68	27.65 " "
II 2 A	12.70	22.33 " "
II 2 B	12.28	24.38 " "
II 2 B	13.33	25.40 " "

The points in the different curves for which under the assumptions made in the different cases we break even financially with alkali chloride electrolysis, are indicated by little circles.



COST OF ALKALI CHLORIDE ELECTROLYSIS PROCESSES AS FUNCTION OF POWER COST.

The parts of the curves at the left of these circles, therefore, represent the conditions under which alkali chloride electrolysis is financially profitable.

If it is desired to use other values than those assumed for the cost of the raw materials or the selling price of caustic soda, the above calculations can be easily changed accordingly.

As the market conditions are at present [in Germany] for the raw materials and final products, we can sum up our results in the following conclusions:

1. According to the different processes, the maximum power cost for which alkali chloride electrolysis is profitable, varies between 0.75 and 1.125 cent per kw-hour.

2. In the smaller installations of the textile industry the direct electrolytic production of hypochlorite (electric bleaching) is in general financially profitable only if a certain saving of active chlorine is figured on. But even if the cost of producing the bleaching liquor by electrolysis is somewhat higher than with the use of bleaching powder, the other technical advantages of electrolytic hypochlorite (less wear and tear of the fibres, smaller acid consumption, etc.) still justify the use of the electrolytic hypochlorite process.

3. In paper and pulp mills as well as in larger installations of the textile (especially linen) industry, it will in general be more profitable not to produce directly a hypochlorite solution, but to produce separately chlorine and caustic soda and make use of or sell the caustic soda. With satisfactory power prices, the erection of such plants is justified *per se*, as against the purchase of chloride of lime.

4. In the large-scale chemical industry and under average conditions the diaphragm processes are superior to the mercury-cathode processes with respect to both first cost and cost of operation. In the case of erection of a new plant, the use of a mercury-cathode cell will be justified only if there is a sure market for the caustic soda, free from sodium chloride, at a higher selling price, and if very cheap power is available.

My thanks are due to Mr. Ebert and Dr. Carl for their help in collecting the data and in checking the numerical calculations.

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Gold Milling in California.

By William H. Storms.

At the time of the discovery of gold in California the methods and means of milling gold ores were of the crudest possible description. In fact, aside from the operations of Mexicans, with arrastras, and South Americans, with their Chile mills, about all that was known of the science of recovering gold from its ores by wet processes was the extremely inefficient methods that had been in vogue in the region about Dahlonega, Georgia and at a few other places in the South, where mining had been attempted in a most primitive way.

Whatever extremely limited ideas the early California miners may have applied to the conduct of their business were patterned after either the practice of the Southerners or that of the Mexicans or Chilenos. Among the first mills built and operated in California in 1850-51 was one at, or near, what is now Nashville, in El Dorado County. This affair consisted of a battery of five stamps. The stems were square and were made of wood. These were lifted by nearly straight cams which operated in vertical slots cut in the stems, the involute cam having not yet been discovered by the mill men. The lower ends of the stems were fitted with iron-shod square shoes. The mortar consisted of a plate of iron laid in the bottom of a wooden box, iron-bound and secured with bolts. This was placed on a foundation of timbers laid at right angles, forming a solid crib. The power was supplied by a small steam engine.

No copper plates were used, the gold being caught in riffles and on heavy blankets, which latter were washed frequently. It is almost needless to say that this crude affair was a most unsatisfactory gold-saving contrivance, nor that for the greater part of the time it was hung up for repairs.

About the time that this mill began operations a party of prospectors discovered a rich vein outcropping a few miles up the canyon from the mine at Nashville. The owners of the mill above described had a duplicate of it at their place but had not set it up, and when the discoverers of the new mine heard of this they at once opened negotiations for the purchase of the outfit, which it appears the owners were glad enough to sell. The mill was moved up to the new discovery and there set up, but not without sundry improvements over that down the canyon.

The purchasers of this mill, who were the owners of what has since been known as the Union mine, took no pains to deny that they were proud of their new acquisition. This mill, although set up with all the skill, care and knowledge that their limited experience suggested, proved to be but little, if any, better than that down the gulch at Nashville.

Soon after starting up all sorts of unsatisfactory conditions

developed. A set of square iron shoes were made for it at Sacramento, but were found to be no better than the others. One of the men suggested an iron stem and shoe, which idea was favored by the other members of the company, but the departure was too radical from the established practice to take the risk of fitting up the entire battery in this new fashion, so they compromised by arranging for one stamp to be placed in the center between the wooden ones.

This iron stamp was made from a piece of shafting and was so evidently superior to the wooden affairs that it was promptly decided to replace all of the wooden stamps with those of iron, but no suitable bars of round iron could be obtained in either Sacramento or San Francisco at the time, and it was several months later before the necessary iron arrived around the Horn at San Francisco.

When the new battery of iron stamps, with round bosses, shoes and dies was finished and set to work, it proved to be such a marked improvement over any of the cumbersome wooden affairs that all mill builders promptly adopted the innovation. In this new mill the wear and tear of cams and tappets was excessive, as the stamp was prevented from turning by a device arranged for the purpose.

It was then suggested by one of the owners, a Mr. Sanford, I think, that it would be an improvement if the stamp were allowed to turn as it was raised by the cam. Some thought not, but the experiment was tried and of course worked to perfection. The cam "dope" was pine tar, which was smeared on with a stick and quickly spread evenly over the under face of the tappet.

Thus one detail after another in the primitive mill was improved as the necessity for improvement appeared. Each detail was worked out by experiment, there being no other school in which to learn than that of hard and expensive experience. An innovation in any mill that proved a success was promptly copied in every mill along the lode, until, in the course of a few years, the California stamp-mill, as we know it to-day, was evolved by these indefatigable pioneers in the business.

After a time copper plates were tried and proved a great success, but it was not until after 1860 that silver-plating of the copper plates was introduced, the first of these, I think, being on the Comstock Lode. Self-feeders were first attempted about 1858, and although they were a great improvement over hand feeding, it was many years before they reached anything like their present perfection.

The mechanism of the stamp-mill has undergone a great change since those early days—far more than the mill practice in California, at any rate, for, with few exceptions, the mill arrangement and daily routine is much the same to-day as it was 30 or more years ago. One would naturally suppose that after all these years of experimentation and varied practice we would have tried out now about every device and that we should have long since been able to plan the ideal gold mill, a mill that all would be willing to agree was superior to any other, yet such is not the case.

The ideal mill is rarely seen, for the ideas of mill builders—not so much the mill founders as those who buy mills and have them built after their own plans and specifications—are almost as widely different as they ever were.

Even to this day can be seen mills where the rock-breaker sets above the level of the floor, so that every pound of ore that goes through the machine must be lifted on a shovel by the attendant. Still more rarely, fortunately, may be seen the spectacle of a man feeding the battery with a shovel. It is but fair to say of this, however, that it nearly always is due to lack of means to purchase an automatic feeder, rather than to the old-fashioned notion that the work could be done better by hand.

It is generally acknowledged that the rock-breaker should be placed outside the mill building and on a separate foundation, but if the situation be such that the breaker must be placed at the mill it is considered good practice to at least provide it with an independent foundation and separate framework of timbers entirely detached from the mill structure. This I have seen done at no small expense, and the independent timber frame then solidly connected with the mill structure by means of timbers and bolts, thus undoing all the advantage that might have been derived from the separate breaker-foundation arrangement.

Ore bins have come in for their share of discussion, some contending that the bin with a flat floor is the best, as it admits the storage of a greater or less amount of ore and also tends to give steadiness to the mill structure. Others maintain that the bin with the sloping floor is best because it supplies the feeders with ore by gravity, still others take the middle course and favor a bin having a bottom which is flat at the front and sloping at the back.

It is my opinion that the proper place to store broken ore is either in the mine or in a bin outside the mill and that the ore bin in the mill should have a sloping bottom from the front backward at an angle of at least 45 deg. to the back of the bin, from which level it may very properly be vertical. In a bin of this shape all the ore delivered to it will run by gravity to the chute doors, excepting, possibly, a very small amount that may lodge in the angle at the bottom between the chutes.

Flat bins permit the storage of ore, it is true, but when this stored ore has to be drawn upon, whenever the natural slope-angle down which the ore will run by gravity has been reached, men must be put into the bin to pick and shovel the ore to the chute doors, and this can rarely be done under an average cost of 20 cents per ton, and it is generally more. Flat bins may have some advantage for the purpose of storage where the ore is high grade, but where the matter of 20 cents per ton is a serious thing for the manager of a low-grade mine to consider, as it is at most low-grade mines, the flat ore bin has no place in arrangements looking toward economy.

On the other hand, few managers of high-grade mines care to keep a lot of rich ore tied up any longer than necessary, so there seems to be no legitimate place for flat ore bins. I can see no advantage of the bin built with a narrow flat section longitudinally of the bin just back of the chute doors. It looks like a compromise between two extremes with nothing whatever to recommend it.

Mortars vary greatly in shape. Some are wide, some are narrow and there is considerable difference in the slope-angle of the interior of mortars near the bottom. All of these variations have a greater or less effect on the rate of discharge, and also to a great extent they determine the condition in which the pulp is discharged from the mortar. A narrow mortar will discharge more rapidly than a wide one, and so, also, will a flaring one. The wide, deep mortar, with high discharge, will slime successfully if this is what is desired, but if a more granular pulp is wanted then the narrow mortar of what is commonly known as the Homestake pattern is the thing.

All ores do not amalgamate with equal facility, and when the type of mortar to be placed in the new mill is to be decided upon this important matter should receive due consideration. If the ore amalgamates readily and most of the gold present is free, then a mortar that will discharge rapidly is necessary and amalgamation may be carried on both inside the mortar and on the outside plates. In some mills large capacity is the main object and no inside amalgamation is attempted. With some exceptionally free-milling ores this is entirely satisfactory, but with other ores is less so.

Ores which are to be amalgamated and the sulphides concentrated after amalgamation should be crushed only sufficiently fine to free the particles of gold and the sulphides from the gangue, further crushing results in the creation of an undesirable quantity of slimes, which will interfere with both amalgamation and subsequent concentration.

All of these peculiarities of ores should be thoroughly in-

vestigated before the type of mortar is decided upon, but in many cases the old-time mill man argues: "A mortar is a mortar, and so long as it is heavy enough to stand the blows of the falling stamps, what more can you ask?" I should say very much more, as I have endeavored to show above. I once had a millman say to me, after an argument over his method of working, "I have been here thirty-five years, and I guess I know what I am doing." This was an unanswerable argument, so I could only venture that he had been at it long enough, and that it was time for him to retire and give some other poor fellow a chance.

Among the numerous other subjects that have been up for controversy is that of battery foundations, and even now, after sixty years of wrangling, and no end of practical experience at home and abroad, not only the old-time millman, but even the engineers, are warmly discussing the various kinds of battery foundations, their shape and size. I think it may be safely said that, except perhaps with the ultra-conservative, concrete foundations are admitted to be superior to those of any form of wood construction.

But at the present moment it is the manner of placing the anchor bolts in these foundations that is being threshed out. I think anchor bolts should be of ample size, say, at least 1½ in. diameter, and supplied with heavy threads that will not strip, these to be so placed in the foundation as to be always easily accessible for tightening, and so arranged as to direction that a broken or worn bolt may be withdrawn at any time and replaced by a new one. To accomplish this its direction in the foundation must be such that it is not obstructed by any portion of the mortar nor by the woodwork of the battery frame. If this be done there appears to be little left to dispute about, as far as I am able to see.

The arrangement of battery posts and their braces is sometimes influenced by the topographical situation, which in some measure may determine the position of the engine, or other motive power. What is known as the "back-knee" construction is the most generally in use, and is the most satisfactory, under usual conditions, but occasionally it is more convenient to place the line shaft on a level with the cam-shaft floor and in front of the line of battery. Here the "front-knee" frame is employed. It should be avoided wherever possible, in favor of the back-knee frame, which is more stable and gives greater rigidity to the battery frame throughout. The front-knee frame is seldom seen in new mills, its undesirable features having been recognized by mill builders generally.

The manner of distributing power from the engine, waterwheel or motor varies, but is of less importance than some of the foregoing considerations. Line shafts are found below the mortars, both behind and in front of the batteries, and at various elevations within certain limitations. Occasionally it is seen on the cam floor, as previously stated, in front of the battery. The best arrangement I have seen was where the line shaft was back of the battery foundations and below the mortars, being supported on a line of independent concrete piers, at a height of about 20 in. above the floor.

Where concrete foundations are to be used I can see no objection to providing a buttress-like projection at the back of these foundations, and at a proper height, to carry the line shaft of the mill. This will render the shaft always easily accessible and will also permit the desirable downpull on the cam shaft pulley, and should prove an altogether satisfactory arrangement.

I know of but one mill in California where the line shaft has been placed higher than the cam shafts, and this is in the North Star mill, near Grass Valley. There the shaft has been placed at the top of the front side of the concrete ore bin. I fail to see that this presents any advantage over that of placing the shaft below the battery at the back, if, indeed, the North Star idea is as good as the latter.

There is little material difference in the arrangement of the amalgamated copper plates in front of the batteries these days. Formerly plates divided by a central strip, thus making two sluices each about 22 in. wide, were much in vogue, but these have given place to wider plates, which are far more satisfactory. Even now in many mills the plate area is too limited for the best economic results.

It may safely be said that as long as gold can be collected at the foot of a run of plates it is good business to supply additional plate surface. Where tailings are as low as 25 or 30 cents per ton, additional plate area may reduce the loss sufficiently to justify the expense of their installation, and the extra work of caring for them, but no amount of additional plates will be of service unless they be kept in proper condition.

Every square foot of plate in the mill should be kept as bright and clean as industrious rubbing will make them. Some millmen have a stated time for doing all millwork. This is a mistake. The time to dress plates is when they need it, no matter whether it be twelve hours after the last dressing or in half an hour after. This will make extra work for the millman, but that is what he is there for.

Any portion of a mill plate covered with sand, or that is discolored by mineral stain of any sort, is not catching a particle of gold, and the mill may as well be without that plate surface thus affected.

A 5-stamp battery, crushing 5 tons daily per stamp, will in a year crush 9000 tons of ore. If an additional saving of 5 cents per ton can be effected by the provision of more plate area, it should be done, as it would mean a net saving of $450 per year for each 5-stamp battery.

In nearly all California gold mines the gold occurs only partly free, the balance of it being associated with the sulphides of the base metals, iron and copper pyrite, and to a less extent with arsenopyrite, galena and blende, and still more rarely with tellurium. This usually necessitates the concentration of the auriferous sulphides.

Occasionally mines are found where the amount of sulphides present is either too small, or of too low a grade to justify their concentration. At the Yellow Aster Mine, in Kern County, for instance, the quantity of sulphides is both too small and of too low grade to warrant their being concentrated; the Sheep Ranch Mine, which at one time was also a large producer, is another example where the sulphides were too poor to concentrate, although the ore averaged $15 per ton in gold.

The ore of the Golden Gate Mine, near Sonora, in Tuolumne County, on the other hand, carried nearly all the value in the sulphides, amalgamation in the mill yielding only about 50 cents per ton on an ore worth from $5 to $20 per ton. The sulphides from this ore were concentrated on vanners and canvas tables and the gold extracted by the Plattner chlorination process.

It is in the process of concentration after amalgamation that the California millman is, generally speaking, behind the times. There are a few exceptions, but they are lamentably few, indeed. However, there appears to be a disposition to investigate other methods, with a view to improvement. This may be, and probably is, due to the entry of younger men into the profession. These younger men are not yet in the rut, and their schooling is very likely to keep them out of it. As a class they are ambitious, energetic, and seeking methods to bring about needed reforms in an antiquated practice, which all who are unbiased will admit are needed.

Ordinarily, from the mill plates the pulp flows by launder to mercury traps, and from these again by launder to the concentrating machines, which are of many makes, but mostly either vanners or tables of the Wilfley type. Some mills have both. In nearly every mill the practice is to split the stream of pulp from the trap below each 5 stamps, into two equal streams, each of these two streams going to a separate vanner or table. No classification of any kind is attempted, and the result is that there is invariable loss.

This loss may be low, it is true, perhaps not more than 30 to 60 cents per ton, but on ore as easily treated as the most of that on the Mother Lode of California this loss is too high, and it can, in probably every case, be lowered if the pulp be classified before going to the concentrating machines.

Two or even three classes of material should be made by the classifying cones, spitzkasten, or whatever device is employed, and each class sent to a separate belt machine, or table, that is carefully adjusted as to vibration, grade, speed of travel, and volume of pulp to water, so as to save the highest possible amount from each class of material under treatment.

This is a physical impossibility in the present practice, where no classification whatever is even attempted. The additional expense would be in the first cost and in maintenance, but the additional saving, in most cases, and I believe in every case, would amply justify the cost.

The average California mill has not been planned with this object in view, and as a result there is generally too little head room between the plate floor and the concentrator floor of the mill. The difference in height rarely reaches 15 ft. and very often it is less than 10 ft., and these heights do not permit the placing of satisfactory classifying devices of any sort in their proper place.

This, however, does not make proper classification in such a mill an impossibility, for there are numerous mechanical devices for cheaply elevating mill pulp, and where there is insufficient head room the difficulty can be overcome by employing one of these in connection with a system of suitable launders to distribute the classified material from the classifiers to the concentrating machines. In Colorado this very important part of gold milling is much better understood than in California, and in the former State may be seen many mills which would serve as an excellent copy for the mills of California.

There is one innovation in mill construction to which I wish to call attention, and this is found in the mill of the Original Amador Gold Mining Company, at Amador City. In this mill, which was built in 1900, there are twenty stamps in two units of ten stamps each. The separate units consist of five stamps arranged on either side of the cam-shaft pulley, this requiring four battery posts to each ten stamps, and a cam shaft a little longer than usual, but the even distribution of stress more than makes up for the strangeness of the new departure, and I think that as soon as the advantages of this style in construction are appreciated there will be many more mills built along similar lines. In this mill also are found classifying devices—one of the very few places in California where they can be found.

Berkeley, Cal.

The process of making sand-lime brick was invented in Germany, where it probably has had its greatest development up to the present.

Observations made there seem to prove that sand-lime brick is a building material which will resist shining soot. Three years ago an inn was built in Greifenhagen, in which sand-lime brick was used for exerything except the lining of the chimney flues, which were built of common brick because at that time the question whether sand-lime brick was equally resistant to fire had not been settled. In spite of the fact that lignite briquets were burned, shining soot was found in the flues in a short time, and the flue bricks were permeated as high as the rafters of the ground floor. When the flues were pulled down, however, it was found that the sand-lime brick were unharmed and white in color.

This experience seems to indicate that sand-lime brick will be found in general practice the best building material to withstand shining soot.

The Determination of Gold and Silver in Copper.

BY EDWARD F. KERN AND ALBERT A. HEIMROD.

The determination of gold and silver in copper bullion is accomplished by either the direct-fire assay method or else by a combination wet and and fire method. There seems to be no conventional standard method, as at some plants the former is used whereas at other plants the latter one is used, or else a more or less modification of either.

The authors, having undertaken an investigation which required a standard method for the determination of gold and silver in copper-gold and copper-silver alloys, encountered the difficulty of finding a method which would give duplicate check results. The reason for the lack of agreement in the assays, which were made by these methods, could not be charged against anything definite, so the working out of an accurate method was undertaken.

As the chances for losses of gold and silver by volatilization are greater by the direct fire method, and because the proper temperature conditions of scorification are difficult to control, the combined wet and fire method was favored for the determination of the gold, and an entire wet method for that of the silver.

Assay of Copper-Gold Solutions.

One of the main sources of error in the determination of gold in copper bullion (or an alloy), by the combined wet and fire method is the loss of some of the finer particles of gold when the bulky copper solutions are filtered. A means of preventing this loss is to add to the solution some salt that will produce a heavy precipitate and which while precipitating will collect all of the suspended particles of gold. The salts generally used for this purpose are either lead acetate which added to the solution containing sulphates gives a precipitate of lead sulphate, or else silver nitrate which added to the solution containing a chloride gives silver chloride. Whether the one collecting agent possesses any advantages over the other was not known, so the first series of assays was conducted with the idea of finding out which salt is the more suitable one to use.

With this point in view, a solution of gold and copper was made up, each cubic centimeter of which contained 0.17654 gram of copper and 0.000205 gram of gold. Samples of 50 c.c. each of this standard solution were taken and assayed. The general method of procedure, after the sample had been measured out, was to add 10 c.c. of concentrated sulphuric acid and evaporate to dryness. The mass was then taken up with distilled water and diluted to 250 c.c. Ten grams of ferrous sulphate crystals ($FeSO_4.7H_2O$) dissolved in distilled water was added, which precipitated the gold in solution as finely divided metallic particles. In order to collect these particles, the first reagent used was from 10 c.c. to 20 c.c. of a saturated solution of lead acetate which gave a heavy precipitate of lead sulphate. The solution was filtered and the precipitate washed, dried, and scorified with 20 to 40 grams of test lead. The button was then cupelled and the gold bead weighed. Results of these assays are given in Table I.

It was found, however, that more satisfactory results were obtained when a silver nitrate solution was used instead of lead acetate, and the silver precipitated as a chloride. The beads obtained by scorification and cupellation, when silver nitrate was used, were parted, and the gold annealed before weighing.

The table (No. 1) shows the amount of collecting reagent used and the results obtained. Although the results on the whole are fair, the errors are not constant; the greater variations occurred when lead acetate was used, and the lesser variations when silver chloride was the collecting reagent. The results, however, gave assurance that for the assay of copper-gold alloys, the latter collecting agent would give the most ac-

Table I. Assays of Standard Copper—Gold Solutions.

Assay Number	c.c. of Lead Acetate Solution	c.c. of Conc. HCl Added	Grams of Ag NO_3 Added in Solution	Grams of Test Lead for Scorification	Theoretical Amount of Gold in Solution Gram.	Weight of Gold in Gram Obtained	Difference in Weights Gram.	Percentage Error
1	20	...	....	20	0.01026	0.00816	0.00210	20.4
2	30	...	....	20	0.01026	0.00982	0.00044	4.28
3	10	...	....	25	0.01026	0.01026	0.00000	0.0
4	10	...	....	25	0.01026	0.01004	0.00022	2.14
5	10	...	....	25	0.01026	0.01024	0.00002	0.19
6	...	5	0.5	25	0.01026	0.01010	0.00016	1.50
7	...	5	0.5	25	0.01026	0.01026	0.00000	0.0
8	...	5	10.0	40	0.01026	0.01017	0.00009	0.87
9	...	5	10.0	40	0.01026	0.01015	0.00011	1.08
10	...	5	10.0	40	0.01026	0.01015	0.00011	1.08
11	10	...	....	40	0.01026	0.01013	0.00013	1.26
12	10	...	....	40	0.01026	0.01000	0.00026	2.53
13	10	...	....	40	0.00718	0.007	0.00018	2.50
14	10	...	....	40	0.01026	0.1020	0.00006	.59
15	10	...	....	40	0.01026	0.01016	0.00010	0.97

curate results. With this as a starting point, the working out of a standard method for the determination of gold in copper was undertaken.

Preparation of Standard Copper-Gold Alloys.

The standard alloys were prepared by melting from 1500 to 2000 grams of electrolytic copper in a graphite crucible, the copper being completely covered with charcoal. A calculated amount of gold was added to five to ten times its weight of molten copper in a small graphite crucible. After maintaining the alloy in a molten condition for a few minutes, it was poured into the larger crucible containing the major portion of the copper. The larger bulk of copper was first thoroughly poled to tough pitch and then the rich alloy added. The alloy was well stirred with a stick of graphite, and then poured from an elevation of from 6 to 8 feet into a tub of cold water, having a depth of 12 inches. It was found that by granulating the alloy, segregation of the gold (and of silver) was overcome. These granules, varying from less than 1 mm. to 1 cm in diameter, were the standard alloys used in the experiments.

Assay of Copper-Gold Alloys.

An alloy (No. 1), containing approximately 100 oz. of gold to the ton of copper was used, for the first series of assays. As close to an assay ton (29.166 grams) as possible was weighed out, and dissolved in 90 c.c. of concentrated nitric acid. The residue of gold particles were then dissolved by adding 2 c.c. of a mixture of 4 parts nitric acid and 1 part hydrochloric acid (aqua-regia). When all the metal was in solution, 37.0 c.c. of concentrated sulphuric acid were added and the solution carefully evaporated to dryness. The mass of crystals was dissolved in distilled water, and the solution diluted to 300 c.c., then 10 grams of ferrous sulphate crystals ($FeSO_4.7H_2O$) dissolved in water was added in order to precipitate the gold.

In the third assay, of Table II, the fine particles of metallic gold were collected in filterable form by adding 10 c.c. of a saturated solution of lead acetate and 0.5 grams of silver nitrate dissolved in water. This gave a mixed precipitate of lead sulphate and silver chloride. In the other experiments the gold particles were collected by adding 0.5 grams of silver nitrate dissolved in water and precipitating the silver with a solution of hydrochloric acid. In all cases the solutions were filtered, the precipitate washed, dried, and together with the filter paper were scorified with 45 grams of test lead. The resulting buttons were cupelled and the beads parted with nitric acid. Results are tabulated in Table II.

The errors in this series of assays (No. 1 to 7) were attributed to the fact that the ferrous sulphate did not throw down all the gold, as it was partially oxidized by the presence of aqua-regia. The reason for adding aqua regia was in order to get the alloy entirely in solution, thus finally obtaining a complete separation of the gold from the copper. By following this procedure it was thought that if no copper was present with the gold during cupellation, the errors of assaying would be reduced.

Table II. Assay of Standard Copper—Gold Alloys.

Assay Number	Alloy Number	Weight of Sample in Grams.	Grams of $FeSO_4.7H_2O$ Added	c.c. of HCl Added	Grams of $AgNO_3$ Added	Grams of Test Lead for Scorification	Weight of Gold in Gram.	Oz. Gold per Ton
1	1	29.1667	10	5	0.5	45	0.09577	95.77
2	1	29.1709	10	5	0.5	45	0.09482	94.80
3	1	29.1696	10	5	0.5	45	0.09758	97.57
4	1	29.1660	10	5	0.5	45	0.09622	96.22
5	1	29.1660	10	5	0.5	45	0.08932	89.32
6	1	29.1712	10	5	0.5	45	0.09780	97.77
7	1	29.1685	10	5	0.5	45	0.09725	97.24

In the next series of assays (No. 8 to No. 13), the gold was not dissolved in aqua-regia, and only silver nitrate used as the collecting agent for the gold particles, it being precipitated as chloride with hydrochloric acid. The errors in this series of assays were accounted for by the presence of aqua-regia formed by free hydrochloric acid and traces of nitric acid in the solution. Table III shows the tabulated results. Alloy No. 1 contained approximately 100. oz. of gold per ton, and alloy No. 2 approximately 10. oz. of gold per ton.

Table III. Assay of Standard Copper—Gold Alloys.

Assay No.	Alloy No.	Weight of Sample in Grams.	c.c. Conc. HNO_3 for Dissolving	c.c. Conc. H_2SO_4 for Evaporation	Grams of $FeSO_4.7H_2O$ Added	c.c. Conc. HCl Added	Gram of $AgNO_3$ Added	Grams of Test Lead for Scorification	Gram of Gold Obtained	Oz. Gold per Ton
8	1	29.1711	90	37	10	5	0.5	45	0.09912	99.10
9	1	29.1708	90	37	10	5	0.5	45	0.09929	99.27
10	1	29.1680	90	37	10	5	0.5	45	0.08936	89.35
11	2	29.1697	90	37	10	5	0.05	45	0.00990	9.90
12	2	29.1723	90	37	10	5	0.05	45	0.01008	10.07
13	2	29.1722	90	37	10	5	0.05	45	0.01030	10.29

The next set of experiments gave us the method which was adopted for the determination of gold in copper-gold alloys.

Standard Method for Determining Gold in Copper-Gold Alloys.

The method finally adopted and used for assaying copper-gold alloys is as follows: About 1 assay ton (29.166 grams) of the alloy was carefully weighed, placed in a 900 c.c. beaker, and 20 c.c. of distilled water added. The beaker was covered with a watch glass and 90 to 100 c.c. of concentrated nitric acid (sp. gr. 1.42) was added in small quantities of 20 to 25 c.c. from time to time. After all the alloy was dissolved, 40 c.c. of concentrated sulphuric acid (sp. gr. 1.84) was added in order to transform all the copper nitrate into copper sulphate. (0.87 c.c. conc. H_2SO_4 theoretically required for 1 gram of copper). Light blue needle-like crystals were formed by the addition of the sulphuric acid, but were easily broken up by stirring and by subsequent evaporation of the supernatent liquid to sulphur trioxide fumes.

The solution was evaporated to dryness and fumes of sulphur trioxide by placing the beaker on a sand bath, so regulating the heat as to prevent bumping. This left a mass of copper sulphate crystals absolutely dry with part of the water of crystallization driven off. The beakers and contents were allowed to cool, and 500 c.c. of distilled water was added, which readily dissolved the salts on heating. Then 10.0 grams of ferrous sulphate crystals ($FeSO_4.7H_2O$), dissolved in water, was added for the purpose of destroying any free nitric acid present and to precipitate any gold in solution. Twenty-five c.c. of a saturated solution of sodium chloride, and enough silver nitrate to make a bead weighing about eight times the weight of gold present were next added. If a greater amount of silver nitrate is used, the bead obtained by scorifying and

cupelling will be too large to part, that is, the gold will be left as a powder instead of a single porous mass.

1.5 gram $AgNO_3$ is equivalent to 1.0 gram Ag.

The silver nitrate was added when the copper sulphate solution was hot, and the solution rapidly stirred in order to make the silver chloride coagulate and settle, thus collecting all the finely divided gold in the precipitate on the bottom of the beaker. The solution was filtered and the precipitate washed until all copper salts were removed from the filter paper. The beaker was then thoroughly wiped with a piece of filter paper which was added to the main precipitate.

The silver chloride precipitate containing the gold, together with the filter paper, and about 20 grams of test lead were placed on a litharge glazed scorifier. The filter paper was slowly burnt off, and about 20 grams more of test lead added, making a total of 40 grams. A pinch of borax glass was placed on top of the lead and the scorification made in the usual manner. The resulting lead button was cupelled on a medium hard bone-ash cupel, and the gold-silver bead parted with dilute nitric acid (1 part conc. HNO_3 and four parts water). The residue was washed with distilled water, annealed and weighed.

Glazed scorifiers were used in order to prevent loss of gold by particles of the ignited filter paper becoming attached to the sides of the scorifier which prevented their collecting in the lead button. The melting of the glaze washed the particles of ash into the lead bath, which collected any gold which it contained. The glazing was done by putting from 10 to 15 grams of litharge on the scorifier, placing same in a muffle furnace and heating to fusion of the litharge; the excess of litharge was then poured off and the scorifier allowed to cool for subsequent use.

Results of assays made by the above method are given in Table IV. Alloy No. 2 contained approximately 10 ounces gold per ton, alloy No. 3 approximately 36 ounces per ton, and alloy No. 4 approximately 96 ounces per ton.

Table IV. Assay of Standard Copper—Gold Alloys.

Assay No.	Alloy No.	Weight of Sample in Grams.	c.c. Conc. HNO_3 for Dissolving	c.c. Conc H_2SO_4 for Evaporation	Grams of $FeSO_4 \cdot 7H_2O$ Added	Gram of $AgNO_3$ Added	Grams of Test Lead for Scorification	Gram of Gold Obtained	Oz. Gold per Ton
14	2	29.1652	90	40	10	0.08	40	0.01044	10.44
15	2	29.1594	90	40	10	0.08	40	0.01040	10.40
16	2	29.1676	90	40	10	0.08	40	0.01042	10.42
17	2	29.1630	90	40	10	0.08	40	0.01040	10.40
18	3	29.1534	90	40	10	0.00	40	0.03568	35.69
19	3	29.1511	90	40	10	0.30	40	0.03560	35.62
20	3	29.1538	90	40	10	0.30	40	0.03566	35.67
21	4	29.1601	90	40	10	0.80	40	0.09583	95.87
22	4	29.1598	90	40	10	0.80	40	0.09583	95.87
23	4	29.1627	90	40	10	0.80	40	0.09584	95.85

Determination of Silver in Copper-Silver Alloys.

The standard copper-silver alloys were prepared in the same manner as the standard copper-gold alloys, as outlined previously.

Samples of the alloy weighing approximately five, ten and twenty grams were used for the assays. The amount of alloy taken in each case depending upon the quantity of silver the alloy contained. The samples were dissolved in concentrated nitric acid (3 c.c. per gram of alloy) in a covered beaker, the excess of acid was evaporated off, and the evaporation continued until copper nitrate commenced to crystallize out of the hot solution. At this point all of the oxides of nitrogen, which are harmful to the precipitation of the silver by means of alkali sulphocyanate, were removed. The solution and salts were taken up with 400 c.c. of cold distilled water, and 25 c.c. of a saturated solution of ammonium sulphyocyanate (25 grams per 1 liter) was added to the cold solution as quickly as possible and with constant stirring. The silver in the form of silver sulphocyanate coagulated and settled rapidly to the bottom of the beaker. The solution was filtered, and the precipitate washed free from copper salts by means of distilled water. The filter and precipitate were placed in the original beaker in which the precipitation was made and digested with 80 c.c. of concentrated nitric acid. The beaker was covered with a watch-glass and placed on a hot plate and allowed to remain until all of the paper was decomposed and the precipitate was entirely dissolved. The solution was evaporated to small volume before all of the oxides of nitrogen were driven off, which is essential for the success of the analysis (did not take to dryness). After cooling, the salts were taken up with 300 to 350 c.c. of distilled water, and 20 c.c. of a saturated solution of barium nitrate was added, which precipitated the sulphuric acid present:

$$Ba(NO_3)_2 + H_2SO_4 = BaSO_4 + 2HNO_3.$$

The presence of sulphuric acid is the result of oxidation of silver sulphocyanate by nitric acid.

About 3 cu. cm. of a saturated solution of ferric alum was added, as an indicator, and the solution titrated at ordinary temperature with a standard solution of ammonium sulphocyanate (3.2 grams NH_4CNS per liter). The standardization was obtained on samples of pure electrolytic silver, dissolved in nitric acid, boiled, and the solution diluted to about 100 c.c.

1.0 c.c. NH_4CNS solution was equivalent to 0.004465 gram silver.

$$AgNO_3 + NH_4CNS = AgCNS + NH_4NO_3.$$

The end point is readily distinguished by the appearance of a faint orange color, which the solution assumes as soon as all the silver is precipitated, due to the formation of ferric sulphocyanate.

The accuracy of this method, as described, was first tested on standard copper-silver solutions as follows, and later on on copper-silver alloys. About 60 grams of the standard alloy was dissolved in 180 c.c. of concentrated nitric acid in a covered beaker, and the oxides of nitrogen expelled by evaporation. The solution was diluted to 1000 c.c. with distilled water and three samples of 300 c.c. each were accurately measured out and assayed as already explained. These solutions were titrated with two standardized ammonium sulphocyanate solutions, namely: A solution 1.0 c.c. of which was equivalent to 0.004465 gram of silver was run in to within about 1 c.c. of the end point, then the titration was finished with a second standard solution, 1.0 c.c. of which was equivalent to 0.000584 gram of silver.

It was thought that by using the two standard solutions the determinations could be made more accurately. This procedure was, however, found to be unnecessary, so in the later titrations only the stronger standard solution was used (1.0 c.c. equivalent to 0.004465 gram silver). The results of this series of determinations are recorded in Table V.

Table V. Assay of Standard Copper—Silver Solutions.

Assay Number	Grams of Standard Alloy in 1 Liter Solution	c.c. of Solution Titrated	NH_4CNS solution		Gram of Silver in 300 c.c. Solution
			1.0 c.c. = 0.004465 Gram Ag	1.0 c.c. = 0.000584 Gram Ag.	
1	60.0083	300	18.9	12.0	0.09140
2	60.0083	300	18.9	12.3	0.09157
3	60.0083	300	18.9	12.1	0.09146
4	60.0091	300	19.0	10.8	0.09120
5	60.0091	300	19.0	10.7	0.09108
6	60.0091	300	19.0	10.8	0.09120

The results of determinations made on the copper-silver alloys are given in Table VI. The method of analysis was carried out as described.

In order to obtain accurate results by this method of assaying copper-silver alloys, the following precautions should be observed:

If all the oxides of nitrogen are not expelled, and if the solution is too hot during the precipitation of the silver, then the solution will not retain a green color after the excess of

Table VI. Assay of Copper—Silver Alloys.

Assay Number	Alloy Number	Grams of Sample Taken	NH_4CNS Solution. 1.0 c.c. = 0.004465 Gram Ag.	Gram of Silver in Sample	Oz. of Silver per Ton
7	6	20.0279	17.50	0.078140	113.76
8	6	20.0187	17.45	0.077914	113.51
9	7	10.0023	40.40	0.180386	525.99
10	7	10.0036	40.45	0.180609	526.57
11	8	5.0306	50.60	0.225930	1309.90
12	8	5.0213	50.60	0.2.5930	1311.50

ammonium sulphocyanate has been added, when the silver is precipitated from the copper solution. In the first case the solution will change to the original color, namely, blue, at the same time brown fumes of nitrogen tetroxide (N_2O_4) will be given off, causing some of the silver to go back into solution. In the second case on adding the sulphocyanate to the hot solution, copper is reduced and a precipitate of cuprous sulphocyanate forms which interferes with the accurate titration of the final silver solution.

If the final solution, obtained by dissolving the silver sulphocyanate and the filter paper in conc. nitric acid, is evaporated too far (during the operation of expelling the nitrogen oxide fumes) a brown coating of burnt organic matter from the filter paper forms on the bottom of the beaker, and when the solution is diluted it assumes a brownish color which interferes with detecting the end point of the titration.

The dissolving of the alloy, the evaporation of the resulting solution, the dissolving of the precipitate, and the final evaporation of this solution should be carried out carefully in a covered breaker, in order to prevent error as result of spattering.

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Exploitation of Scandinavian Water Powers.—The London *Electrical Review* of April 1 reports from Berlin that negotiations are being conducted by an Anglo-German-Scandinavian financial group with regard to the consolidation into a company of a number of Norwegian and Swedish water-powers, under the title of the Hydraulic Power and Smelting Corporation, and with a share capital of $5,000,000 and debentures of $5,000,000. The scheme relates to water falls already in use and yet to be used, as well as to some of the establishments supplied by them. The assets to be brought into the undertaking include the zinc smelting works and zinc refinery at the Trollhattan Falls and in Sarpsborg, and the water-power of 126,000 hp at the mouth of the Tysse mountain stream in the Hardanger Fiord. The latter installation is already equipped or in process of being equipped, for 57,000 hp and the turbo-generators have been used for years past for supplying the carbide and cyanamide works in the neighboring Odda, in which the German Siemens group is interested. As concerned with the scheme in Germany are mentioned the Allgem. Elek. Ges. the Siemens-Schuckert Works, and the Berlin Electrical Enterprises Co., together with the banks associated with them under the lead of the firm of S. Bleichröder.

Nevada Consolidated Copper Company.—In the quarterly report of this company covering the second quarter of 1911 it is shown that an unusually low grade of ore was handled owing to the advisability of milling a large tonnage of ore removed for the purpose of arranging permanent tracks into Eureka pit. The grade of ore milled during the quarter averaged 1.59 per cent copper. During June all operating reverberatories were fired with oil, except one which was fired with coal and oil. The results have fully justified the use of oil instead of coal at the Steptoe plant. The first basic lined converter was scheduled to go into commission about Aug. 1.

Notes on the Chemistry and Metallurgy of Vanadium—III.

By Warren F. Bleecker.

The Metallurgy of Vanadium Ores.

In the United States the metallurgist may be called upon to treat the following vanadium bearing ores:

Carnotite.
Impregnated Sandstone.
Patronite.
Roscoelite.
Vanadinite.

In addition to the above there have been described a number of vanadium minerals which have not as yet been found in commercial quantities. These are:

Brackebuschite, a lead-zinc vanadate; Dechnite, a lead vanadate Pb (ZnCu) V_2O_6; Descloizite, a lead-zinc vanadate with formula 4 (PbZn) O . V_2O_5 . H_2O; Endlichite, a lead vanadate with V : As : : 1 : 1; Eusynchite $(PbZn)_3V_2O_8$, massive descloizite; Tritochlorite, a lead-copper vanadate; Mottramite, a copper-lead vanadate with formula $(CuPb)_5V_2O_{10}$. $2H_2O$; Psittacinite, a pulverulent copper-lead vanadate; Pucherite, a vanadate of bismuth with formula given as $BiVO_4$; Ramirite, a lead-copper vanadate, and Volborthite, a copper-calcium vanadate with formula given as $(CuCa)_3V_2O_8$.

The pure carnotite mineral is said to contain 18 per cent V_2O_5 and 56 per cent U_3O_8. It occurs as impregnated sandstone, the ore being usually yellow or yellowish green. If the vanadium content of the ore is low the color is canary yellow, while if the vanadium content predominates (which often occurs) the color may be a dark green, almost black.

Carnotite ores are by far the most soluble of vanadium bearing ores under metallurgical treatment. Decomposition may be effected by roasting with sodium chloride or fixed alkalis, after which a large percentage of each metal is soluble in water. The ore is quite soluble in boiling alkali or alkaline carbonates, there being a patented process based upon this property. Decomposition of the ore also may be effected with mineral acids, sulphuric being preferably used. A very comprehensive report of a proposed process for the treatment of carnotite with sulphur monochloride has been published. The difficulties of treatment are therefore not in decomposition, but in the final separation of the uranium and vanadium values and the economical production of compounds or alloys suitable to the consumer. Although it is not the purpose of this paper to discuss the metallurgy of uranium, it may be said that the behavior of the two elements with reagents is in many respects so different that very little study should at least result in an efficient method of separation; and with the uranium once freed from vanadium there will be little difficulty in getting it into a desirable form.

Sandstone impregnated with vanadium has become an important source of vanadium. The ore is somewhat complex and a dried specimen disintegrates with ease into three general components. The first is a green fine substance which has been carefully studied by Hillebrand; second, a coarse white quartz product, apparently containing some vanadium, and third, a further cementing material consisting of vanadium compounds containing lime, silica, alumina and fixed alkalis, together with probable traces of other elements.

In metallurgical processes the usual procedure is to grind to 20 or 30 mesh and roast with sodium chloride. The best results are apparently obtained by roasting at a temperature of 1100-1200° Fahr. Depending upon the oxidizing conditions in the roast, the product, when cold, may be colored yellow, red, green or even black. If the temperature is allowed to run too high the mass will fuse together, with the result that vanadium compounds insoluble in water are formed. The roasted ore is leached with water by a counter-current system of enriching weak liquor by passing it over fresh ore, and finally

washing the ore with fresh water. The solution cannot be enriched in this particular manner beyond a certain limit, about 8g per liter, because of reprecipitation brought about by the soluble basic matter in the ore. I have obtained the best results by keeping the solutions alkaline. One of the methods I use consists in adding a small quantity of crude lye to the sodium chloride before mixing with the raw ore. This has the disadvantage, however, of decomposing a considerable amount of silica, which must be accounted for later on.

Sodium carbonate may be used for decomposition instead of sodium chloride, but does not give good results unless the ore is of especially high vanadium content.

Merely washing the roasted ore with water will not result in complete extraction in any event. The reasons for this are apparent if it is remembered that the ore also contains a small amount of iron, quantities of lime, alumina and silica. Iron vanadate, calcium vanadate and aluminium vanadate are all insoluble in water, while sodium vanadate will unite with soluble silica to form a compound insoluble in neutral solutions. I arrived at the latter conclusion by mixing solutions of sodium silicate and sodium vanadate and neutralizing with hydrochloric acid. A white precipitate results which, while somewhat gelatinous, may be washed quite readily. Repeated washing did not remove the vanadium.

A good commercial extraction of the vanadium values is obtained if the water leaching is followed by washing with either dilute hydrochloric or sulphuric acid. This has the disadvantage of extracting large quantities of lime and alumina.

It may be said that the fine green substance mentioned above as one of the three components of the ore is much more difficult to decompose than the other two, although the fine is originally the richest in vanadium.

I have read several statements relative to the roaster gases containing chlorine and vanadium. A number of careful examinations of the gases from a muffle furnace under different conditions of oxidation failed to show the presence of either free chlorine or vanadium in any form. Free hydrochloric acid, ferric chloride and sodium chloride were the principal compounds evolved during the roasting when sodium chloride alone was used.

In the treatment of vanadiferous sandstone the chief difficulties are overcome if a good extraction has been obtained, and this may only be done by long and careful experimenting with the particular grade of ore to be treated, because the varying amounts of impurities in the different grades of ore will modify the extraction to a greater extent than might at first be imagined.

Patronite is a carbonaceous sulphide of vanadium occurring with iron, nickel, silica and small amounts of other impurities. There is said to be a large body of this ore in Peru, and it is now being economically treated in the United States by the American Vanadium Company. I have recently received a small sample of non-carbonaceous vanadium sulphide from San Miguel County, Colorado. Analysis showed the presence of other vanadium mineral in addition to the sulphide, since the vanadium content was 15.08 per cent V_2O_5, while the sulphur content was 4.30 per cent.

The sulphide ore, after roasting to the oxide, may be decomposed with mineral acid, preferably sulphuric. Only a fair extraction is probably obtained, but since the ore is of such high grade the present methods may be the most economical in the long run. Extraction also may be made by roasting the oxidized ore with crude lye and leaching with water. The problem in this case will be to keep the solutions strongly alkaline without using an excessive amount of alkali, although this may be done by any one of a number of regenerative methods.

Where the metallurgist is having difficulty with extraction, I think it is a good rule to use alkalis, since the behavior of the alkaline vanadates is much more uniform than that of the vanadyl compounds, and in most cases the alkaline vanadates are much more soluble in water, while in dilute solutions vanadyl compounds may be more readily precipitated by impurities than the alkaline vanadates.

Roscoelite is an impossible sort of a mineral to work with commercially. The only agent which I have found to give good results in effecting decomposition is sodium peroxide. Commercially this is out of the question. The best extraction with allowable reagents I obtained by using a mixture consisting of 50 per cent each of sodium chloride and crude lye. Three hundred pounds of this mixture per ton of ore resulted in an extraction of 60 per cent of the vanadium. The raw ore assayed about 3.50 per cent V_2O_5. The mineral is but slowly attacked by concentrated mineral acids, aqua regia not effecting a complete decomposition upon several hours' boiling. Among the reagents tried with roasting were sodium chloride, sodium hydroxide, nitre cake and sodium carbonate. If there were not much greater quantities of more easily treated ores, or if the available quantities of roscoelite were greater a method of treatment would surely be developed, but at the present time it hardly seems worth while. An additional difficulty in the treatment of roscoelite is presented in the saving of the noble metal content, which in Boulder County ore is often of greater value than that of the vanadium.

Vanadinite is a more or less important mineral, the formula being given as $3Pb_3 \cdot V_2O_5 \cdot PbCl_2$. This is the only vanadium ore (at least of present commercial importance) which can be treated by concentration. A company proposing to operate at Cutter, N. M., claims to have considerable quantities of vanadinite, the concentrates containing 15 per cent $BaSO_4$. I am indebted to Mr. Herman A. Prosser for this information.

Lead vanadate may be decomposed by concentrated sulphuric acid with the formation of divanadyl sulphate under the commercial conditions of treatment. It is apparent that the extraction will not be complete and that there will be large quantities of impurities in the tank liquors. The difficulties of treating ore with sulphuric acid in large quantities render such processes unattractive, although it is claimed by those who are more familiar with the sulphuric acid methods of treatment than I that no difficulties are met with and such processes are satisfactory. In order finally to get rid of the greater part of the impurities, the concentrated vanadyl sulphate solutions are evaporated and finally calcined with sodium carbonate with the formation of sodium vanadate. A good extraction may be obtained by treating vanadinite with strong, hot solutions of lye. The ore must be ground to extreme fineness and the leaching continued for some time. Lead and other impurities are more or less soluble in the mill solution, but the matter of purification is very simple as compared with the purification of acid liquors.

Sodium chloride does not seem to give good results as a decomposing agent for vanadinite, although a fair extraction may be obtained by chloridizing in a muffle furnace.

Numerous patents have been taken out both in this country and abroad for the extraction of vanadium from the various ores and slags in which vanadium occurs. I have never read of a process that seemed more than moderately efficient under the conditions described by the author. Since vanadium has many laboratory peculiarities that are not apparent on a mill scale, most of the processes devised are almost totally inoperative. For example, one inventor has tried to patent a method for recovering vanadium from acid solutions by boiling. The phenomenon of vanadic acid precipitating from a boiling acid solution is of course familiar to all chemists, but those who have taken the trouble to compare the amount of vanadium in solution to that precipitated have not wasted money on trying to patent a process based on the reaction. Again, there have been patented processes for the extraction of vanadium by boiling the ore with sulphuric acid. As a metallurgical operation this has never appealed to me on account of the mechanical difficulties; easy enough in a test tube, perhaps, with five times as much acid as ore, but less easy in tanks of

standard materials and with the human equation of the workmen forever to be taken into account.

Smelting methods have been patented, but since vanadium is essentially a slag-forming element there does not seem to be much hope in that direction, especially in the treatment of ores carrying as low as 1 per cent V_2O_5. Although there have been patented processes for the extraction of vanadium by fusion with various agents, such as nitre, potassium bisulphate and the like, the tremendous excess of reagent required, together with the additional heat to effect fusion, and finally the difficulties of dissolving the fused material, should make one very cautious in recommending such a method.

Chemical Methods Necessary.

After the entire field of theory, backed up by thorough experimental evidence, has been canvassed, the metallurgist must be convinced that the sole method of attacking the problem of vanadium extraction is chemical. Expensive reagents cannot be used, nor can cheap ones be used in large excess unless a well tried method of recovery of used materials is without question available. While acid processes of extraction may at present be used profitably in treating low-grade vanadium ores, competition may, in the near future, cause valuable property based on these methods to become idle and worthless.

As before stated the greatest difficulty of vanadium production will have been overcome when the vanadium is in solution and free from gangue. The removal of impurities from this point on may in some cases be more expensive per ton of material handled, but the vanadium will be in a condition in which it is readily handled, either by precipitation or otherwise. Specifically I shall discuss only vanadium solutions obtained from the leaching of vanadiferous sandstone, because those interested in solutions obtained by acid leaching will find data bearing directly upon the treatment of such solutions as well.

Nature and Composition of Mill Solutions.

The compounds present in mill solutions will be sodium vanadate and salts of lime, alumina, silica, titanium and sodium, particularly sodium chloride. In addition, some of the vanadium will be combined with each and every other element present, probably including vanadium itself. In neutral solutions there will be a little, and of course in acid solutions there will be a great deal.

In beginning to leach the first solution from the ore will be very low in vanadium, 1.1 g per liter, and will be rich in calcium chloride, calcium carbonate and sodium chloride. This solution is usually colorless and its content depends upon many conditions, such as character of roasting process, fineness of ore, rapidity of leaching, etc. Following the clear liquor will percolate a yellowish solution containing a maximum amount of vanadium, together with sodium chloride and all of the other impurities. Treating one ton of ore with a ton of water, approximately 10 per cent of the solution will be of the first variety, 20 per cent of the second and the remainder which will percolate through will have a decreasing vanadium content following the parabolic law of leaching. The richest liquor should have a vanadium content of about 8 g per liter under favorable conditions. If the liquors are allowed to mix as they come from the ore precipitation of calcium vanadate will occur in the receiving tank. This vanadate of lime usually carries about 18 per cent V_2O_5 and may be filtered and washed.

The greater part of the basic impurities contained in the solution may be removed by the careful addition of a slight excess of sodium carbonate and heating the solution to 90° C. Enough carbonate must be added to make the solution distinctly alkaline. The impurities will settle to the bottom of the tank and the purified solution may be removed by decantation. The residue is of a dirty gray color and contains usually about 10 per cent vanadic acid. While practically all of the lime and alumina may be removed in this manner, the silica usually remains in solution. I have found no certain method of removing the silica and except where products of exceptional purity are required there is no necessity for removing it. As a matter of fact there is no need of removing any of the impurities unless a very pure product is to be finally extracted from the solution.

In case the solutions are acid with sulphuric acid, the principal impurity will usually be alumina, the other impurities being obviously lime and iron, together with any fixed alkali which may have been present in the ore either originally or added for purposes of extraction. The vanadium is present as vanadyl sulphate, at least 15 different forms of which have been studied, representing the various states of oxidation of both the vanadium and the sulphur. In mill practice the sulphuric acid solutions of vanadium are either green or blue, which colors show a lower state of oxidation of the vanadium than the pentoxide. If to such an impure solution peroxide of hydrogen is added a yellow precipitate is thrown down, which, if filtered off and dried, turns red. Analysis shows it to be a vanadate of iron. This same precipitation may also separate after the solution has stood for several days in an open tank. There is of course some significance to be attached to this reaction, but it is too remote to be readily observed. It is only mentioned here to illustrate the tendency of vanadium compounds to do the unusual thing, as compared with the better known elements.

The removal of the impurities from an acid solution without the addition of alkali in excess is difficult, if not impossible. Sulphate of calcium, insoluble under most conditions, is here held in solution in considerable quantities and may be removed only by an expensive electrolytic method.

If the acid of the solution in hydrochloric, the vanadium is present as vanadyl chloride, in which two atoms of chlorine are united with one vanadyl group. The same impurities are present, lime, of course, being in considerable quantity.

It should not be forgotten that most vanadium ores contain traces of titanium which is extracted with alkali and remains with the vanadium through the various stages of purification, and in some instances the removal of this element may necessitate a modification of the process.

It is of course obvious that it is as necessary to get the vanadium out of solution completely and efficiently as it was to obtain a complete extraction. Not only is it desirable to precipitate all of the vanadium from the solution, but it is necessary that it should be in a chemical combination and physical state suitable either for further refinement or for use by the ultimate consumer. Upon these conditions depends the method of precipitation or whatever refinement of the liquor is necessary.

Since the most important vanadium product is ferrovanadium, the various methods of preparing vanadate of iron, from which ferrovanadium may easily be made, will be discussed.

Preparation of Vanadate of Iron.

The method now in general use for the production of vanadate of iron is to add ferrous sulphate to a neutral or nearly neutral solution of sodium vanadate. The compound is precipitated as either a yellow or dark green, bulky mass, which settles to the bottom of the tank, and from which about 75 per cent of the liquor may be decanted. The usual practice is to filterpress the sludge and dry the cakes in a suitable oven.

There are a number of things about this method which are undesirable. In the first place vanadate of iron, like all other vanadates, is soluble in mineral acids. The first requirement, therefore, is that no free acid be present in the solution or be formed there during the precipitation. The sodium sulphate formed in the solution by substitution is a powerful solvent for the vanadate of iron unless the iron sulphate be present in considerable excess, and even then it is doubtful if a quantitative reaction takes place. If the solution should be slightly alkaline the hydroxide of iron is formed at the same time, thus lowering the grade of the product. Another serious objection to the method depends upon the fact that most of the vanadium in the mill solution is reduced; that is, there are vanadic and hypovanadic salts combined with the sodium after roasting, and

these salts are apparently unchanged in the leaching process. Again, the iron is added in a ferrous condition which does not improve matters. The result is that a small per cent of colloidal precipitate is formed, which makes filtering a difficult problem.

At the same time there are formed certain sulphates of vanadium and iron which are more soluble in water than in the mill solution, so that after the washing has reached a certain stage, provided the precipitate can be washed at all, the vanadium begins to appear in the wash water. This would not be so serious if it were not for the fact that, unless removed by washing, the impurities remaining in the vanadate of iron mean havoc in the electric furnace at the time of final reduction. The vanadate of iron as removed from the filterpress carries in the neighborhood of 90 *per cent of the mill solution.* It is obvious that the dried, unwashed product must contain an enormous amount of the salts originally contained in the mill solution.. These consist mostly of sodium chloride, sodium sulphate, calcium salts, titanium salts and salts of silicon and aluminium. As stated above, some of these salts are in combination with the vanadate of iron and some held mechanically by the sludge. That the presence of these salts is detrimental to electric furnace reduction is easily shown by the heating of the impure product in an open crucible; indeed, it is claimed that a loss of 10 per cent of the vanadium has been known to occur in the simple process of driving off the moisture at a moderate temperature. An instance was cited me in which the appalling loss of 70 per cent of the vanadium value was experienced in trying to reduce some vanadate of iron to ferrovanadium.

Unless special precautions have been taken to remove them, silica and alumina as well as lime are always present in the commercial product. Producers of ferrovanadium say that small quantities of these impurities do not affect the value of the vanadate.

Electrolytic Methods of Precipitation.

While it is a well-known fact that salts in solution can be acted upon and changed to desirable forms by the proper application of the electric current, the importance of this in metallurgical operations has not until recently been fully realized. The deterioration of anodes is one of the serious defects of many proposed metallurgical schemes, but there are many instances in which this fact may be used to advantage, and there are processes in operation at the present time in which highly desirable anode products are economically and efficiently turned out.

If two iron plates with a difference of potential of two or three volts are suspended in a receptacle containing vanadium liquor, immediate precipitation of iron vanadate takes place at the anode and continues until all of the vanadium is precipitated. The reaction is efficient and occurs with great rapidity, as will presently be noted.

In case vanadate of iron is to be made from the mill solutions, the acid wash water, if such is used, is led into the same storage tank as was the rich water solution. The water solution is but very slightly alkaline and the mixture is practically neutral, but the fact of its being either slightly acid or alkaline does not seriously affect the electrolytic precipitation. There will separate out and settle to the bottom of the storage tank a yellow vanadium compound containing iron and calcium. This may be allowed to run into the precipitating tank, as lime is not a deleterious impurity in vanadate of iron.

The precipitating is done in wooden tanks 8 ft. long and 4 sq. ft. in cross-section. Each tank is provided with copper busbars running along the top just inside and supported by porcelain insulators. These bars are supplied with current generated at a pressure of from four to five volts. The electrodes are all of iron, although the material of the cathodes may be copper, or, in fact, any other metallic conductor, even carbon. The anodes are made of cast iron, the dimensions being 18 in. x 18 in. x 0.5 in. They may be made from scrap if desired. The cathodes are also 18 in. square, but are only 1/16 in. thick. Since the thickness is not important, so far as their work is concerned, they should be as thin as possible on account of the weight and expense.

In the electrolytic production of vanadate of iron from mill solutions the phenomena have been studied at some length It has been observed that the presence of acid radicals other than vanadium aids the precipitation very materially by exerting a solvent action on the anode during electrolysis. In this case a salt of iron is formed which immediately reacts with the sodium vanadate, with the chemical precipitation of vanadate of iron as would be the case if the same iron salt were added from the outside. If there are no other acid radicals than the vanadium present the precipitation occurs more slowly and the precipitate adheres quite firmly to the anodes, necessitating removal either by scraping or violent agitation. The presence of iron oxide on the anode is beneficial because of the ease with which it is attacked by the vanadic acid radical, and in this case the precipitate is the yellow vanadate of iron instead of the green vanadate. A portion of the lime contained in the solution will be precipitated during electrolysis as vanadate of lime, and this is recovered with the iron vanadate. Some of the calcium is precipitated on the cathode, partly combined with vanadium and partly combined with carbonic acid. In all cases of moderate anode density the precipitate is very easily filtered and washed, but if a very high anode density be used the precipitate will be colloidal.

As stated above, the precipitation begins immediately upon the passage of the current, and under favorable conditions proceeds with extreme repidity. It is possible to precipitate 95 per cent of the vanadium content from a ton of solution carrying 5 g per liter in five minutes with a current of 300-400 amp at a difference of potential of four volts. It is neither necessary nor desirable to remove all of the vanadium from the solution, because the solution which has been electrolyzed is more active as a solvent upon the ore than pure water.

The temperature at which the reaction takes place most efficiently is about 40° C., although good results have been obtained at 0° C. and at the other extreme of 100° C. The distance between the electrodes may be a matter of convenience, although a distance of about 4 in. was used throughout most of my experiments.

The impurities in the vanadate of iron will be sodium chloride, salts of calcium, iron, aluminium, silica and titanium. As a rule any impurity present in the solution will be found in part in the product. If an especially good grade of vanadate is to be made it would be necessary first to purify the solution, which may be done with little expense.

It will be noted that vanadium is always an anode product in hydroelectrolytic operations. Use is made of this property in several methods for producing vanadium compounds besides vanadate of iron. From a sodium vanadate solution the vanadates of almost all of the metals may be made, including vanadate of gold, but not platinum. Copper vanadate is very easily produced, although conditions are much more limited than when precipitating vanadate of iron.

Copper vanadate is either blue or green, and is usually diluted with oxides of copper. It may be filtered and washed in wooden tanks and dried in copper pans. The compound fuses at about 700° C. and becomes very fluid unless a portion of the vanadium is reduced by the action of the graphite of the crucible used for melting. In this case the melt becomes very pasty with the separation of metallic copper. In this connection it is interesting to note that the copper so reduced carries but a very small percentage of vanadium and the vanadium (copper vanadate) carries much carbon. Copper vanadate is the basis of two processes for the production of vanadic acid.

Production of Vanadic Acid.

One of these methods is electrolytic only in the recovery of

the copper to complete the cycle of reactions. I shall describe the method because it affords an opportunity to call attention to some difficulties which may not at first be apparent.

The vanadate of copper is freed as much as possible from solution by washing on canvas filters. It is then placed in suitable containers and treated with sulphuric acid (1-1). The mass turns green at first, then brown and finally red. Upon the addition of the proper amount of acid, which is a matter of experience, the residual vanadic acid is filtered on canvas filters. It may be washed with pure water until the washings show no copper. If an insufficient amount of acid was used the residue will contain copper, while if too much acid was used the recovery of vanadic acid will be small. Not more than 25 per cent of the vanadic acid content of the copper vanadate can be recovered by the first operation.

The solution containing sulphate of copper and vanadyl sulphate is neutralized by passing it over fresh copper vanadate and is then subjected to electrolysis for the recovery of the copper. The copper is precipitated on plates of size suitable for anodes to be used in the precipitating tanks. The solution containing vanadyl sulphate and free sulphuric acid is used for the final decomposition of copper vanadate, and the cycle of reactions is complete. The process is not particularly efficient for several reasons, which, as stated above, are not obvious. The precipitation of copper from copper-vanadyl sulphate solution is not efficient, although perhaps it could be made so, but the mechanical difficulties of handling are great and so numerous that the method is too expensive. The greatest difficulty is in the accumulation of impurities in the vanadyl sulphate solution, the greatest of which is sulphate of calcium. This salt is very soluble in the copper-vanadyl sulphate solutions and soon loads up so that fresh acid has to be used. The process is operative and is actually continuous as outlined, but it is doubtful if it can ever be used commercially.

If an ingot of fused vanadate of copper is attached to a source of electricity so that it becomes the cathode in a simple cell, in which a dilute solution of vanadic acid is the electrolyte, the vanadium tends to pass to the anode, while the copper tends to remain at the cathode in the metallic state. The copper will remain more or less firmly on the cathode, but the vanadic acid in excess of saturation of the electrolyte falls to the bottom of the cell, owing to its inability to adhere to the anode. The anode must be inert, preferably platinum. Any mineral acids present in the solution exert a deleterious action and for this reason the copper vanadate should be as pure as possible. Pure water may be used in the cell because the vanadate of copper is soluble enough to allow the action to proceed to some extent, and as the water becomes saturated with vanadic acid the current passes very readily and the precipitation takes place rapidly. As the action in the cell begins spots of metallic copper appear on the surface of the cathode and the solution becomes yellow, due to the dissolved vanadic acid. As soon as the solution becomes saturated a film of the yellow pentoxide appears on the anode and soon the acid begins to settle out of the solution on the anode and on the walls of the cell. The precipitate is of a rich yellow color and settles to the bottom of the cell in a curdy mass, in which condition it may easily be removed by filtration. No washing of the precipitate is necessary, since the solution contains no impurities if a pure copper vanadate was used. The vanadic acid obtained in this way is easily melted and cast into ingots and should assay 100 per cent V_2O_5. The method is expensive as compared with other electrolytic methods for the production of vanadic acid on account of the large number of operations necessary to prepare the copper vanadate for electrolysis. However, if a pure product is required the method is far cheaper than any method of chemical separation. It must be remembered that vanadic acid may be obtained pure and yet be contaminated during the final operations of getting it into the desired physical state. This is especially true if it is to be melted and cast into ingots.

Any method of producing vanadium compounds by strictly chemical means is uncertain and expensive. I refer to methods of which the general public, including myself, are aware. Vanadic acid, for example, may be produced chemically in a number of ways, of which I shall mention two or three as examples.

If a dried sample of vanadate of calcium be treated with the proper quantity of hot hydrochloric acid, either concentrated or dilute, there results a separation of the red pentoxide of vanadium, which may be filtered and washed first with hydrochloric acid and finally with pure water. The vanadic acid made in this manner will contain a small quantity of lime salts and traces of any other impurities that were originally present in the calcium vanadate. It will also contain a considerable quantity of hydrochloric acid combined with the vanadium as vanadyl dichloride. The chlorine content as well as the 10 per cent moisture present may be removed by fusing the product in either platinum or fused silica. Depending upon the experience of the chemist a practically pure vanadic acid may be made from vanadate of lime.

Small quantities of an excellent grade of vanadium pentoxide may be made by precipitating the vanadium with a salt of mercury and finally driving off the mercury by heat. The only waste of mercury occurs in the handling of the precipitate, since all of the metal may be recovered as in the practice of gold retorting.

Perhaps the oldest and best known method of making vanadic acid is by first making ammonium metavanadate and driving off the ammonium salts by heat. The operation is conducted by having a concentrated solution of sodium vanadate which has been previously freed from a large percentage of the impurities originally present. To the sodium vanadate solution is added solid ammonium chloride, previously purified, to supersaturation. The white, crystalline ammonium-vanadium salt separates out and may be filtered and washed. If care is used in preparing the solutions the first metavanadate obtained will yield about 97 per cent V_2O_5 upon ignition. By dissolving the ammonium metavanadate in caustic soda and reprecipitating the vanadic acid may finally be obtained practically pure. In this method the expense of the reagents is prohibitive, besides, the reactions are slow and not quantitative.

Vanadic acid can be prepared by the continued boiling of acid solutions containing vanadium salts. The vanadium separates out of the solution in small quantities at various concentrations. Starting with a given amount of vanadium, perhaps a maximum of 10 per cent may finally be recovered by continued boiling a dilute solution. The pentoxide so recovered is not pure and the method apparently has no commercial value.

Vanadic acid may also be prepared by the treatment of dried sodium vanadate with acid, but as in the above instances, the yield is so small as compared with the expense that totally new methods must be devised if vanadic acid is demanded as a commercial product in large quantities.

Author's Electrolytic Method for Vanadic Acid.

I have devised an electrolytic method which is operative and which may be depended upon for economical production of a very high grade oxide. The method consists in taking advantage of the different solubilities of sodium vanadate and vanadic acid in the same bulk of water, and the procedure is as follows:

Sodium vanadate liquor which does not contain chlorides is purified by heating to 90° C. and making slightly alkaline with sodium carbonate. This liquor is decanted into an evaporator and concentrated to 30° Baumé. At this degree of concentration it will contain about 200 g per liter vanadic acid, depending upon the amount of sodium salts present. This concentrated solution is electrolyzed in a compartment cell consisting of a porous cup with a capacity of 8 liters inside a suitable jar or tank with a capacity of 5-10 gal. The vanadium liquor is poured into the porous compartment and water containing a little lye is poured into the outer compartment. The anode is of platinum foil 6 in. x 6 in., and the cathode is of either iron or copper, preferably the latter. A difference of

potential of 6-8 volts, depending upon the resistance of the porous cell, is taken from a low-voltage generator. A current density of about 30 amp per square foot at the anode will result. The sodium ions pass through the porous cell into the cathode compartment, leaving the vanadium ions in the anode compartment. Since sodium vanadate is very soluble in water, and vanadic acid is almost insoluble in water, a precipitation results upon the removal of the alkali. The presence of mineral acids interferes seriously with this reaction, since the vanadium is soluble in dilute solutions of mineral acids. If a small amount of chlorine is present it is evolved partly at the anode and that which is not evolved causes the formation of various complex compounds which result in a very low grade product. The presence of sulphuric acid in the anode compartment causes the precipitation of sodium vanadyl sulphate containing 73 per cent V_2O_5.

Any silica or lime salts not removed by the preliminary purification treatment will precipitate in part in the anode compartment just as the solution becomes neutral and may be removed by filtration. In spite of every precaution some impurities will find their way into the product, and not a little of this comes from the disintegration of the porous cell. The greatest objection to this process is the necessity for using platinum in such large quantities, one ounce of platinum being required for every pound of vanadic acid produced per day of 24 hours. There is no loss of platinum, however, and it is probable that platinum plate electrodes would greatly reduce the actual weight of platinum required.

The vanadium pentoxide produced by this method is of the red variety and presents a tough leathery appearance, and may be filtered and washed with ease. Without taking more than the usual precautions, the grade should be 98 per cent V_2O_5, the impurities being principally lime and silica. The current efficiency is as high as could be expected with a compartment cell. There is no apparent reason why the units may not be of any desired capacity, and it is very probable that the process could be made continuous.

From the foregoing statement, that chlorine interferes with the recovery of the vanadium pentoxide, it is obvious that the salt roast is out of the question. Therefore if vanadic acid is the required product another method of extraction from the ore will have to be used. A modification of the sodium carbonate roast will be found satisfactory, although the extraction will not be so high, nor will the operation be as cheaply performed. A small amount of chlorine will be present even in this case because the commercial sodium carbonate carries about 1 per cent sodium chloride and the commercial sodium hydrate sometimes carries as high as 3 per cent.

Presence of Titanium in Vanadium Ores and Products.

Since titanium is present in almost all vanadium ores, although the percentage is usually very small, its extraction is excellent. This element resembles vanadium in a large number of reactions, and it is natural to expect that titanium should be found as an impurity in commercial vanadium products. I have found titanium in vanadic acid of supposedly high purity, and in some cases the percentage has been higher than one would suppose. The fixed alkalis are good solvents for titanium and this element may in some cases interfere with assays for vanadium. From alkaline or neutral solutions lead acetate precipitates lead titanate, which exactly resembles lead vanadate excepting that the precipitation is not quantitative under the same conditions that lead vanadate is. This precipitate is soluble in nitric acid and in sulphuric acid solution is apparently reduced to sulphur dioxide to the same extent that iron would be if present. The orange red color of titanium with hydrogen peroxide in nitric acid solution may in some cases so nearly resemble the vanadium color that the chemist is at a loss to understand why he cannot precipitate all of the vanadium out of the solution.

In connection with the presence of titanium in vanadium products, the phenomenon of the evanescent end point when titrating with potassium permanganate may be due to this element.

Vanadium for Steel Making.

Since practically all of the vanadium produced at the present time is finally to be used in making vanadium iron and steels, it is proper to say a word regarding the preparation of the vanadium for use by the steel maker. The vanadium must be added in a metallic form because a portion of its value lies in its property of removing traces of oxygen and nitrogen and uniting with carbon to form carbides in the metal. Any vanadium present as oxide will probably be totally lost in the slags. The form in which vanadium is now added to the steel is an alloy of vanadium and iron carrying from 25 per cent to 50 per cent metallic V. There are difficulties in the production of ferrovanadium which at the present time have not been overcome.

Ferrovanadium may be produced from either vanadate of iron or vanadate of calcium (and of course vanadic acid) by reduction with carbon at a temperature of about 1500° C. The reduction is most efficiently carried on in an electric furnace of any type that will give the proper temperature. The alloy, which melts between 1300° C. and 1400° C., depending upon the percentage of vanadium, settles easily to the bottom of the crucible, and may be tapped and cast into ingots. Ferrovanadium made in this way will contain a high carbon content, some silicon and aluminium. Steel makers do not want ferro with carbon over 1 per cent at the outside, while the alloy obtained by the above method will contain much more than this.

A passable grade of ferro is being made by reduction with metallic aluminium. The process is necessarily expensive on account of the price of the reducing agent and certain losses not incident to other processes.

Although I have never seen the process in operation, the reduction of vanadium by metallic silicon or ferrosilicon appears to me to be one of the best yet devised. Mr. F. M. Becket, of Niagara Falls, is the inventor of the process. Mr. Becket was the first man in the United States to produce vanadium alloys in commercial quantities, and he probably knows more about this end of the game than any one else. Therefore I have no hesitation in accepting his statements relating to his process, which is based upon sound metallurgical principles.

A recent process patented by Mr. Boericke ought to give a very high-grade alloy and should be efficient. Mr. Boericke uses either metallic manganese or ferromanganese as a reducing agent, thus avoiding the presence of carbon, silicon or aluminium in the final product.

Numerous other processes have been patented, but none appear to be of any considerable importance.

To sum up, it may be said that great advances have been made during the last few years in every branch of vanadium metallurgy. Efficient processes of extraction, precipitation and producing alloys have been developed and these, at the present prices of vanadium products, allow the various ores to be worked at a profit. As with problems concerning other metals, none of the processes are entirely satisfactory in the fullest sense. Owing to the fact that vanadium seems to be firmly established in the steel industry, and promises to be of great value in other lines as well, and since competition is going to force the prices steadily downward, one is compelled to predict that processes of extraction and refining of high efficiency will be developed in the near future. The time when operators can slop a little acid on picked ore and make a profitable extraction, or throw away 1 per cent roasted tailings, or gracefully accept a penalty for uranium values in vanadium products will soon be past, and so much the better for metallurgical intelligence and really careful experiment.

I wish to thank my assistant, Mr. Arnold J. Guerber, for months of painstaking work on the problems here discussed and for carefully prepared analytical data upon which many

of the conclusions are based. In papers of this character, because of the newness of the subject, there must have crept in some misstatements and hasty conclusions. I shall be very glad to have attention called to those which may be observed.

Boulder, Col.

The Electric Pig Iron and Steel Plant at Trollhättan, Sweden.

In the July issue, page 368, and in the September issue, page 459, the first parts were published of the important paper presented by Mr. J. A. Leffler to the Jornkontoret (the Swedish Association of Iron Masters) at their meeting at Stockholm on May 31 on the electric iron ore reduction plant at Trollhättan, Sweden. In the following we give an abstract of the concluding portion of Mr. Leffler's paper, together with an abstract of a paper by Mr. E. Odelberg on the conversion of the electrically reduced pig iron into steel.

Electrodes and Electrode Consumption.

Part of the electrodes used at the works were supplied by the Planiawerke, Ratibor, Silesia, and a part by the Höganäs-Billesholm Company, Sweden. The purpose in using both was to obtain a comparison between the well known Planiawerke electrodes and electrodes of Swedish manufacture. Results of the analyses of the electrodes are shown in Table I:

TABLE I.

Analysis of Electrodes.

	Electrodes from Plania.	Höganäs.
Ash	2.80%	3.96%
S (total)	0.79%	1.06%
P	0.27%	0.16%
Composition of the Ash:		
SO_3	0.44%	0.97%
P_2O_5	0.627%	0.37%
SiO_2	37.80%	42.00%
$K_2O + Na_2O$	1.21%	0.45%
CaO	6.08%	10.20%
MgO	2.62%	2.16%
Fe_2O_3	28.04%	21.70%
Mn_3O_4	0.52%	0.38%
Al_2O_3	21.22%	19.80%
Total	98.557%	98.03%

The rather large difference in the ash analyses is probably due to the difficulty to burn the last traces of carbon by ignition.

From the above will be seen that the Höganäs electrodes contain a somewhat high percentage of ash and sulphur. In producing iron of high quality it is, of course, necessary that especially the percentage of sulphur should be kept as low as possible. With the electrode consumption found in actual operation and assuming that all of the sulphur combines with the iron, 1 per cent sulphur in the electrodes corresponds to 0.005 per cent to 0.006 per cent sulphur in the iron.

The electrodes in the furnace are four in number. Each electrode is built up of 4 cartons, 2 m long and 330 x 330 mm in cross-section arranged to form an electrode of 660 x 660 mm cross-section. Before being bound together the sides forming the joints are planed to a flat surface and the joint is made with a thick paste of molasses and graphite, so that the closest joint possible is formed. The upper part of the electrode is covered with asbestos paper and thin sheet iron and the top surface is covered with a thick layer of ground asbestos and silicate of potash.

The sides of the electrodes are carefully planed smooth at the upper part for a distance of about 250 mm in order to make a good contact surface for the electric contacts. Between these and the electrode is also placed a fine mesh copper wire cloth.

In order to observe the quality of the different makes, electrodes were built up of carbons from the two factories and each kind used under similar conditions. No difference in durability could be discovered.

FIG. 1.—FURNACE HOUSE AT TROLLHÄTTAN.

Table II gives the complete record for two electrodes (called No. 1 and No. 2), also the average for ten electrodes.

TABLE II.

Electrode.	Working Time. Hours.	Min.	Weight in Kg. Before.	After.	Consumption of Electrode. Total Kg.	Per Cent.	Consumption per Ton of Iron. Gross.	Net.
No. 1	1041	37	1321	557.5	763.5	57.79	2.09	1.21
No. 2	1115	31	1325	743	582	43.92	1.85	0.81
Average of 10	906	6	1313.3	635.8	677.5	51.61	2.28	1.16

The Charges.

During the time which this report concerns 29 different qualities of charge were used, which, however, may be regarded as four main groups, viz. unroasted Tuollavaara, roasted Tuolluvaara, Borgvik and Uddeholm ores.

Referring to Table III it will be noticed that the variations in the first group mainly concerned modifications of the silica contents of the slag and also varying proportions of Tuolluvaara ore in the charge.

In the third group a complete change was made in the mixture of the ore at the request of the iron works from which

TABLE III.

SUMMARY OF RESULTS AT THE WORKS OF "JORNKONTORET," TROLLHÄTTAN

Total Hours During Which Same Charge Used	Total Charges Worked	Used Ore															Lime-stone	Total Materia	Charged Along the Sides of the Furnace Per cent. of Total Materials	Used				Obtained		
		Kilogrammes																		Coke	Elec trode Car-bon	Charcoal		Iron		Slag kg
	at 0.65 cu. m	Tcolluvaara		Kirrgrufvan (Stråberg)	Stallberg	Klacka-Lerberg	Haggrufvan	Grindgrufvan	Persberg		Taberg	Nordmarken	Finnmossen	Langban	Total Ore				kg		cu. m.	kg	Tap ings	kg	
		Un-roasted	Roast-ed						Ore	Concen-trates																
Group I																										
.....	...	3,027	...	177	354	...	...	...	...	...	...	...	...	...	3,558	305	3,863	.....	3,100	380	...	...	1	2,310	900
88.30	211	74,891	...	4,432	8,844	...	...	...	...	...	...	...	...	...	88,167	3,524	91,691	100	...	...	137.15	24,668	1—19	54,263	...
287.30	574	189,087	...	11,203	22,406	...	...	...	...	...	...	...	...	...	222,696	13,483	236,179	100	...	...	373.1	65,678	19—58	151,235	...
287.00	312	102,181	12,367	11,972	...	...	...	...	...	...	...	...	...	...	126,520	7,245	133,765	100	...	223	202.8	35,534	58—80	86,560	...
7.35	4	1,292	...	196	152	...	...	...	...	...	...	...	...	...	1,640	92	1,732	100	...	...	2.6	412	80	1,010	...
13.10	14	4,522	...	826	532	...	...	...	...	...	...	...	...	...	5,880	322	6,202	100	...	...	9.1	1,549	80—81	3,630	...
5.50	7	3,133	...	458	359	...	...	...	...	...	...	...	...	...	3,950	207	4,157	100	...	...	4.55	809	81	1,820	...
11.00	20	7,140	...	1,040	820	...	...	...	...	...	...	...	...	...	9,000	460	9,460	100	...	...	13.	2,295	81—82	4,675	...
39.20	85	31,332	...	4,536	3,612	...	...	...	...	...	...	...	...	...	39,480	1,932	41,412	100	525	156	55.25	9,572	82—88	21,765	...
98.25	294	93,858	...	14,214	11,028	...	...	...	...	...	...	...	...	...	119,100	6,762	125,862	100 och 90	...	111	191.1	32,744	88—104	73,970	...
1,017.20	2,314	734,852	...	121,524	84,555	...	...	...	...	...	...	...	...	...	940,931	56,282	997,213	100, 90 och 83	...	475	1,504.1	256,340	104—230	619,625	...
240.40	467	181,380	...	13,158	8,982	...	...	...	...	...	...	...	...	...	203,520	8,122	211,642	100	...	114	303.55	50,370	231—255	135,990	...
2,096.20	4,302	1,423,668	...	183,954	153,262	...	...	...	...	...	...	...	...	...	1,760,884	98,431	1,859,315	.....	525	1,079	2,796.3	479,971	1—255	1,154,543	236,575
Group II																										
193.10	503	5,163	194,677	14,029	9,757	...	...	...	...	...	...	...	...	...	223,626	8,931	232,557	100	...	100	326.95	54,754	256—282	145,495	32,645
Group III																										
49.00	120	...	...	...	...	15,600	3,120	21,840	21,840	...	...	...	...	...	62,400	11,880	74,280	100	...	...	78.	12,811	283—289	30,655	...
32.50	66	...	...	...	...	7,970	1,594	11,158	11,158	...	...	...	...	...	31,880	5,386	37,266	100	...	...	42.9	7,122	289—292	16,315	...
85.35	113	...	...	...	...	7,444	2,931	16,244	31,471	...	...	...	...	...	58,090	7,557	65,647	100	...	...	73.45	12,586	292—298	30,300	...
176.55	253	...	...	...	...	17,458	6,958	38,258	74,086	...	...	...	...	...	136,760	20,673	157,433	100	...	296	164.45	29,024	298—309	58,470	...
254.30	294	...	...	...	...	18,210	7,452	39,786	77,412	...	...	...	...	...	142,860	24,354	167,214	100	...	98	191.1	32,739	310—326	71,825	...
63.20	150	...	...	...	...	8,116	3,631	20,977	36,306	...	...	...	...	...	69,030	13,806	82,836	100	...	...	97.5	16,263	327—336	40,455	...
662.10	996	...	...	...	...	74,798	25,686	148,263	252,273	...	...	...	...	...	501,020	83,656	584,676	...	...	394	647.4	110,545	283—336	248,020	193,510
Group IV																										
89.45	262	20,940	...	...	...	...	...	...	...	...	30,130	8,051	41,624	3,955	104,700	7,398	112,098	100	...	...	170.3	27,931	337—349	72,060	...
15.45	48	3,840	...	...	...	...	...	...	...	...	5,520	1,488	7,632	720	19,200	1,920	21,120	100	...	...	31.2	5,022	350—352	13,520	...
73.00	206	16,160	...	...	...	...	...	...	...	8,904	23,084	6,242	31,782	2,868	89,040	8,904	97,944	100	...	...	133.9	21,757	352—363	46,065	...
30.30	90	6,291	...	...	...	...	...	...	...	7,908	9,165	2,427	12,582	1,167	39,540	3,954	43,494	100	...	...	58.5	9,233	363—367	18,390	...
21.10	63	4,221	...	...	...	...	...	...	...	5,292	6,111	1,638	8,442	756	26,460	3,150	29,610	100	...	...	40.9	6,203	368—371	14,764	...
41.00	110	6,219	...	...	...	...	...	...	...	13,149	8,766	2,547	12,069	1,080	43,830	5,247	49,077	100	...	...	71.5	11,065	372—376	25,908	...
3.55	12	498	...	...	...	...	...	...	...	2,490	720	192	984	96	4980	594	5,574	100	...	...	7.8	1,199	376—377	3,002	...
95.10	245	10,244	...	...	...	...	...	...	...	51,220	14,898	3,875	20,243	1,960	102,440	10,244	112,684	100	...	...	159.25	25,006	377—389	57,070	...
24.45	60	2,100	...	...	...	...	...	...	...	15,840	3,000	840	4,200	420	26,400	2,100	28,500	100	...	...	39.	6,022	389—392	14,405	...
51.05	106	2,704	...	...	...	...	...	...	...	32,032	3,663	1,144	5,697	520	45,760	3,640	49,400	100	...	100	68.9	10,647	392—397	19,175	...
24.40	32	1,888	...	...	...	...	...	...	...	4,032	.....	933	6,334	253	13,440	1,088	14,528	100	...	...	20.8	3,065	397—399	6,736	...
54.05	144	40,320	...	...	...	...	...	...	...	17,280	...	...	...	...	57,600	4,320	61,920	100	...	...	93.6	14,408	399—406	41,033	...
525.20	1,378	115,425	...	...	...	...	...	...	...	158,147	105,057	29,377	151,589	13,795	573,390	52,559	625,949	...	...	...	895.7	141,556	337—406	332,128	151,965
Grand Total																										
3,477.00	7,179	1,544,256	194,677	197,983	163,019	74,798	25,686	148,263	252,273	158,147	105,057	29,377	151,589	13,795	3,058,920	243,577	3,302,497	.....	3,625	2,053	1,666.35	786,826	1—406	1,882,496	615,595

TABLE III—(*Continued*)

Per cent. Iron in Ore Charge	Slag per Ton Iron kg	Material Charged per 100 L. Charcoal kg	Charcoal per Ton Iron Containing kg				Time Used For Working Hours	Time Used For Working Min.	Time Used For Interruptions Hours	Time Used For Interruptions Min.	Time Used Total Hours	Time Used Total Min.	Electric Power Average Load kw	Kw-hours Used Total	Kw-hours Used Pr Ton Iron	Iron Produced per Kw.-Year Ton	Consumption of Electrodes kg Gross	Consumption of Electrodes kg Net	Consumption of Electrodes Pr Ton Iron Gross	Consumption of Electrodes Pr Ton Iron Net
			cu. m. Water	Gases	Ash	C Coke														
Group I																				
Total 65.57	205	66.49	2,422 69.1	41.7	11.8	293.1	415.7	2,009	56	105	39	2,115	35	1,319	2,651,029	2,296	3.82	13,012	6,743	11.24	5.83
Group II																				
Total 65.50	224	71.13	2,247 50.8	36.9	11.0	277.6	376.3	184	32	4	58	189	30	1,694	312,601	2,149	4.08	1,578	763	10.84	5.24
Group III Total 49.50	780	90.31	2,610 59.8	49.3	13.2	323.4	445.7	639	18	20	57	660	15	1,017	650,480	2,623	3.34	2,281	1,121	9.19	4.52
Group IV Total 57.92	458	69.88	2,697 40.2	43.1	17.2	325.7	426.2	506	34	22	11	528	45	1,733	877,706	2,643	3.31	2,474	1,285	7.45	3.87
Grand Total 61.54	327	70.77	2,479 60.9	42.9	12.8	301.4	418.0	3,348	10	153	45	3,501	55	1,344	4,500,596	2,391	3.66	19,345	9,912	10.28	*5.27

*Net—51.24% of gross.

it was sent and a gradual increase of lime was necessitated on account of the sulphur contents of the iron.

The fourth group shows a gradually increasing percentage of concentrates from Persberg which was mixed with the ore.

Table III forms a summary report of the various quantities of ore, charcoal, energy and electrodes used for respective charges and also the amounts of pig iron and slag produced from four groups of charges.

The Durability of the Crucible and Roof.

The roof of the crucible was at first considered as being a weak point of the construction, but it may be stated that it has stood the test in an excellent manner, as only two repairs of any importance have been necessary.

In the neighborhood of the electrodes the arch became red hot in a few places after protracted working, but it generally proved possible to prevent the bricks from burning through by cooling them with an air blast.

It is probable that the roof would have lasted still better if the furnace could have been worked the entire time with both phases. The second phase was, however, at times out of action. In this way the heating of the roof became uneven and it was damaged by cracks resulting from the stresses.

In the morning of April 6 when the charge contained 70 per cent concentrates a settling occurred from the shaft into the crucible which was accompanied by so violent a development of gas that the roof was damaged between two electrodes. At the time the loose bricks closing the holes in the roof were so tightly sintered into their places that they did not act as safety valves. Repairs were, however, effected in less than two hours. Once the iron cut through the lining of the hearth and the iron shell, causing an interruption of about two hours.

Iron and Slag.

Analysis of the iron was made from each tapping and analysis of the slag and the gas at very frequent intervals. The accompanying tables IV and V show four different analyses which may be taken as averages for those taken and the average of all the analyses are also given. The complete tables are of great interest as showing the continuous and regular working of the furnace, but are too extensive to publish in these columns.

TABLE IV.
Analyses of Iron.

Date.	Tapping.	Composition in Per Cent. C	Si	S	P
Jan. 3, 1911	No. 138	4.19	1.35	0.90	0.004 0.021
Jan. 14, 1911	No. 171	4.04	0.75	0.82	0.005 0.020
Jan. 16, 1911	No. 324	3.0	0.14	0.08	0.028 0.019
Jan. 30, 1911	No. 372	3.1	0.45	0.40	0.014 0.010
Min.		2.30	0.03	0.08	0.004 0.010
Max.		4.50	2.21	1.14	0.094 0.026
Average		3.2	0.3	0.3	0.028 0.015

Analyses of Slag.

Date, 1911.	Tapping	Composition in Per Cent. SiO_2	Al_2O_3	TiO_2	FeO	MnO	CaO	MgO	CaS	P_2O_5	Silicon*
Jan. 3	138	44.76	1.20	2.58	1.75	1.35	31.70	15.10	0.077	Trace	1.61
Jan. 14	171	41.60	6.85	2.72	1.49	1.48	28.91	16.70	0.063	....	1.70
Jan. 16	324	46.82	5.06	...	6.89	0.23	33.24	7.97	0.023	0.041	1.91
Jan. 30	372	37.98	6.98	0.37	1.28	0.52	27.98	23.45	0.123	....	1.42
Min.	.....	36.40	1.20	0.36	0.50	0.08	25.00	3.55	0.014	....	1.31
Max.	.....	49.20	12.85	5.92	10.92	4.48	35.00	23.45	0.217	....	1.94
Average	.....	43.00	5.00	3.00	2.00	2.00	30.00	15.00	0.050	....	1.60

* SiO_2, TiO_n, AL_2O_3

FeO, MnO, CaO, MgO

A subject of great interest is the question whether by this process iron of a uniform composition can be produced for any desired length of time.

As no O and C, except what is contained in the ore, the fuel, and the electrodes, is introduced into the furnace, it is evident that the process is to a large degree dependent on the relative proportions between the quantities of O and C so introduced. It is also apparent that these proportions are apt to vary owing to variations of the fuel and ore and the gas circulation. These conditions must be watched carefully and the charges proportioned accordingly. Irregularities of work-

ing can also be remedied by additions of ore and charcoal through the openings in the roof of the crucible.

The problem of even gas circulation cannot yet be considered satisfactorily solved. It is apparent that the CO_2 percentage of the escaping gas is largely dependent upon the gas circulation and for same reason if the latter is disturbed, as for instance by choking charcoal or ore dust in pipes, etc., the CO_2 percentage will change with consequent effect in the supply of C.

In studying the iron and slag analysis these facts should be taken into consideration and the results appear better.

The analyses of slag show that these have been kept more siliceous than desirable for the basic lining of the hearth. This has, however, been done for the purpose of obtaining results as closely comparable as possible with the treatment of the same ores by the ordinary blast-furnace process.

A few observations of the temperatures of iron and slag taken by means of an optical pyrometer are given in Table IX. In the table is also shown the percentage of Si and Mn in order to show the connections between these and the temperatures.

TABLE VI.—TEMPERATURES OF IRON AND SLAG.

Tapping.	Per Cent Si.	Per Cent Mn.	Temp. in deg. Cent. Iron.	Temp. in deg. Cent. Slag.
169	0.80	0.76	1380	1420
174	0.81	0.83	1330	1375
228	0.14	0.26	1270	1290
401	0.13	0.12	1280	1370
Min.	0.04	0.06	1270	1270
Max.	1.26	0.77	1380	1470
Aver.	0.40	0.30	1300	1370

TABLE VII.—ANALYSES OF GAS.

Date. 1911.	Composition in Per Cent of Volume CO_2	O	CO	H	CH_4	N	Dry Gas. Weight in kg. per cu. m.	Dry Gas. Heat Value per cu. m.
Jan. 3	28.2	..	..	..	..	..	..	..
Jan. 14	27.2	..	57.5	14.8	..	0.5	1.276	2113
Jan. 16	24.4	..	56.4	10.0	1.8	7.4	1.302	2112
Jan. 30	21.8	..	65.4	10.4	..	2.4	1.288	2239
Min.	3.2	0	46.2	2.5	0.8	0.4	1.171	2018
Max.	31.2	1.2	82.0	20.2	9.1	14.3	1.377	3030
Aver.	23.0	0	60.0	10.0	2	5	1.225	2500

In spite of the large number of analyses made it would be premature to draw any definite conclusions concerning the action of the gas under different conditions. It is, however, apparent that currents of varying composition are apt to pass through different parts of the shaft so that, for instance, the gas at the centre of the furnace and at the side in the same section may show a different composition.

The outside temperature of the furnace was kept constant and normal but in a couple of cases when some damage was done by overheating.

An average of 110 liters per minute of cooling water was used for cooling the electrodes with a loss of an average of 97.8 kw, which is nearly 7 per cent of the total load on the furnace.

Working Results.

While it may not be advisable to draw final conclusions on the basis of the results obtained it is nevertheless of interest to examine these results with a view to find indications for the continued development and improvement of the process.

As a definite and highly satisfactory experience it should be noted that the need of repairs of the crucible has been very slight. During a period of nearly 5 months of continuous working only about 18 hours have been required for repairs which must be regarded as an eminently satisfactory result.

It is also satisfactory to note that the height and other dimensions of the shaft have in the main proved correct for normal conditions for fuel and ore.

If, however, earthy ore or concentrates or too fine fuel is used, or if the charge contains too much moisture the working is injuriously affected as would equally be the case with an ordinary blast furnace, and the design of the shaft does not appear to be well suited for such conditions. This is because it is then especially important to maintain a strong circulation of gas.

If this cannot be done the moisture is not evaporated, but gradually penetrates further down in the shaft causing a lowering of the temperature. The reducing power of the CO is then decreased and the percentage of CO_2 in the resulting gas decreases. But this percentage of CO_2 is the indicator of the degree of ultilization of the fuel used and must, therefore, under all circumstances be maintained as high as possible.

For the conditions mentioned a shaft with a wider neck would be more suitable, as it leaves a larger space for the gas to pass up and a reduction of the temperature would be counteracted. Possibly it would also be an advantage to make the shaft wider and lower.

In order to produce an effective and uniform circulation of the gas it is in the first place necessary to purify it by washing, so that the frequent interruptions for cleaning may be avoided. Further the gas pipes must be quite tight so that the gas drawn by the fan is really blown into the crucible, which must also be made as tight as possible. Alterations in these respects are at present in progress at the works.

The circulation of gas which is so useful and necessary for the purpose indicated above and which moreover is a protection for the roof, has, however, the disadvantage that the electrodes are attacked by the carbonic acid in the gas. Nevertheless, the electrode consumption has been low. The average for the entire time per ton of iron has been 10.28 kilogrammes gross and 5.27 kilogrammes net. Electrodes with screw joints are now being adapted for the furnace and the gross consumption may then be reduced—under most favorable circumstances to equal the net consumption.

The experience with this furnace appears to indicate that with this size four electrodes is a minimum number. It would probably be better to use six electrodes with three-phase currents. This would distribute the heat more uniformly and the risk of disturbances of the work on account of defects in the transformers would be reduced. Losses through cooling water would in this case probably be increased, but this would undoubtedly be more than counterbalanced by the advantages gained.

During the time to which the report refers only unburnt limestone has been used. In view of future comparisons with ordinary blast furnace results it was desired to maintain the conditions very much similar to those under which the same ores are usually treated. It is, however, evident that by the use of burnt lime the economy of the process will be improved and this improvement would increase in proportion as the quantity of lime required is increased. Whether unburnt or burnt limestone is used the percentage of CO_2 in the gas remains the same. This proves that the CO_2 derived from the limestone may be replaced by CO_2 formed by the reduction of the ore by CO. The power required for burning the limestone is, therefore, saved and the same quantity of fuel reduced more ore.

* * *

While Mr. J. A. Leffler's paper dealt with the reduction of pig iron from iron ore in the electric furnace at Trollhättan, Mr. E. Odelberg presented a paper on the results obtained in the conversion of this electrically produced pig iron into steel. An abstract of Mr. Odelberg's paper is herewith given.

As the manufacture has been in progress only for a short time, and as further the appearance and analysis of the pig iron were in some respects not the best that could have been desired during the first period of working, it was with some hesitation that the steel works received and treated the product.

The results up to the present time are, therefore, few and of indifferent interest and perhaps somewhat one-sided.

The Behavior and Qualities of the Electric Pig Iron in the Open-Hearth Process.

For acid furnaces it is desirable to have a pig iron with 0.4 to 1.0 per cent. Si or more, and many manufacturers consider that they require at least the same quantity of Mn to obtain a fully satisfactory product.

At the Trollhättan plant there was at first produced a considerable quantity of iron which as regards surface and fracture had the characteristic appearance of spongy iron and of which the analysis showed only a very small fraction of Si and Mn. At that time there was, therefore, some hesitation about the treatment of this iron, the material for which was the same as ordinarily used for the production of a high-grade steel.

The first test of using this iron at Degerfors was made in a basic furnace with addition of ordinary grey iron and a smaller quantity of scrap than usual in order to make sure of an acceptable product of hot iron. The result was, however, so satisfactory that already at the third charge only white electric pig iron was used, together with 50 per cent. scrap. The somewhat surprising observation was then made that the expected boil of the material immediately after the melting did not occur.

The tests with melting of spongy pig iron were then transferred to an acid furnace. Small quantities were used at first but these were very soon increased to the ordinary proportions of 64 per cent. pig iron and 36 per cent. scrap.

As anyone acquainted with the open-hearth process is aware there occurs immediately after, and even during the melting, a violent boil of the material. This is especially the case in an acid furnace if iron too poor in silicon is used (even if this is not what is called spongy iron). A similar partial boil occurs again on teeming the charge into the ladle and moulds and this is invariably the case with hard steel. It was, therefore, surprising that this partial boil did not occur, but instead the regular "boil" commenced immediately after the material was melted and before the oreing.

The charges of such iron were finished in considerably shorter time than the ordinary (about 8 hours). They also required considerably less oreing, but nevertheless were quite hot and liquid when teemed. With equal additions of Mn and Si as ordinarily a product was obtained with the same percentage of Mn and Si as is generally obtained with pig iron of normal analysis from an ordinary blast furnace.

When, however, the same open-health furnace was charged with electric pig iron of correct appearance and normal percentages of Mn and Si, together with the same proportion of scrap as before the treatment required considerably longer time and more ore than can be regarded as desirable.

These conditions are elucidated in the working results of Degerfors which show that the time and oreing of the charges vary generally in accordance with quite small variations of the Si and Mn contents in the ore.

At Degerfors a decided opinion was soon arrived at that electric pig iron of spongy nature was that most suitable for the open-hearth process, whereas the refining of pig iron of normal fracture and analysis proved costly and requires too much time. In regard to the finished product from these apparently so different qualities of pig iron no difference has been found at Degerfors either in teeming or in use.

On the strength of this experience and as on account of decreased output it proved more difficult and costly to produce in the electric reduction furnace pig iron of so-called normal silicon contents, it was decided that the manufacture should be arranged for producing spongy pig iron and that there should be no hesitation about producing the spongy quality, which it is considered desirable to avoid in ordinary blast furnace practice.

Various explanations have been put forward to explain the cause of the rather surprising result of the refining of electric pig iron, which in appearance as well as composition, has been the opposite of pig iron produced in the ordinary blast furnace.

The true explanation is probably that the electric pig iron, even of the white spongy quality, is practically free from rests of ore and iron reoxidized by the blast. This is also proven by the fact that the fracture of very thick flowing spongy iron from the electric furnace is perfectly white and free from tarnished bubbles, while the bubbles in spongy iron from the ordinary blast furnace usually show all the colors of the rainbow. On account of this freedom from oxygen the partial boil in the open-hearth furnace during and after the melting does not occur and it is possible to at once decarburize the charge by oreing. The presence of silicon in such pig iron is therefore unnecessary during the first portion of the open-hearth process.

The slowness of the open-hearth process with electric pig iron of normal appearance is due to the silicon not being oxidized at once by oxides in the iron, so that the refining must be accomplished principally by means of ore and only after this is accomplished the decarburizing proper can commence.

In proof of the above it may be mentioned that for a charge of pig iron with 0.61 per cent. Si and 0.68 per cent. Mn a time of 50 minutes was required to reduce the Si and Mn to the low percentages necessary for beginning the decarburization. If the Si and Mn contents are reduced below these limits in an acid furnace, it would appear that they are again absorbed and not again reduced during the decarburization. The carbon contents in the pig iron used for the charge then determines the time and the quantity of ore required for the decarburization to obtain the desired product. As a higher, and according to the ordinary rules, normal percentage of silicon cannot occur in spongy iron from an electric reduction furnace, the Si contents alone has been referred to in this connection as determining the quality of the pig iron. But this iron is also distinguished by being so low in carbon as to approach steel and it should be remembered that the higher percentage of carbon which occurs at the same time with a higher percentage of silicon also has the effect of prolonging the time for decarburizing in the open-hearth furnace.

The charge above referred to, the pig iron for which was mottled and contained 3.69 per cent. C, 0.61 per cent. Si and 0.68 per cent. Mn, required a time of 6 h. 25 min. for a product with 0.60 per cent. C. Another charge for which the pig iron (spongy iron) held 2.75 per cent. C, 0.04 per cent. Si and 0.19 per cent. Mn required, however, a time of only 4 hours for yielding a product with 0.45 per cent. C. In these times mentioned the time for cleaning and repairing the hearth is not included.

A higher percentage of silicon than 1 per cent. in pig iron from an electric reduction furnace is, therefore, if not detrimental to the quality of the final product, an obstruction to the progress of the open-hearth process, and to a certain extent the presence of manganese in such iron has a similar effect.

It is well known that a certain amount of silicon (many manufacturers also demand a considerable percentage of manganese) is necessary in pig iron from an ordinary blast furnace in order to make the iron suitable for the open-hearth process. It is also well known that if the Si and Mn contents in the iron are too high the open-hearth process is protracted, which is undesirable from an economical point of view. The quantities of Si and Mn required for carrying through the process with pig iron from an ordinary blast furnace, are just those needed during and immediately after the melting of the charge to free the iron from the oxides and gases formed by the influence of the blast. Whatever surplus of Si and Mn may be present only protracts the open-hearth process unnecessarily.

The partial boil frequently mentioned in this connection is a premature and gradually increasing partial decarburization of detrimental effect, but it does not occur if the pig iron contains sufficiently large quantities of Si and Mn. The fact that it does not occur in the case of "electric" pig iron with practically no Si and Mn must be explained by this pig iron being free from oxides.

It may therefore be maintained on the strength of the experience gained with electric pig iron treated by the open-hearth process that high silicon contents are detrimental rather than advantageous, while with ordinary pig iron a certain quantity is necessary to neutralize the defects of a reduction process less perfect and ideal than the electric process.

Steel Produced from Electric Pig Iron.

As already mentioned, the steel made at Degerfors, whether produced from pig iron very poor in silicon or from what may be called more normal pig iron from the Trollhättan furnace, has always proved of very good quality when teemed. Such has also always been the case on rolling as indicated by forging and annealing tests as well as by the analyses.

The first product which was rolled to wire was sent to the Blombacka A. B., Lindfors, where tests were immediately made and the quality found to be very good in every respect.

Garpshytte Fabriks, A. B., have also tested a few small lots of rolled wire and obtained very satisfactory results.

A lot of 20 tons with 0.75 per cent. C was sent to one of Degerfor's largest customers for rolled wire for testing. They wrote as follows:

"We have tested the 0.75 per cent. C material you sent us and found that the wire is perfectly satisfactory, but we have not found any particular difference between this material and your usual quality with 0.75 per cent. C which we are in the habit of buying. Kindly note that we have no objections against the electric steel with 0.75 per cent. C. It is very satisfactory, but we cannot say that it is superior to your usual 0.75 per cent. steel."

A quantity of files and axes have been made from steel with 1.25 per cent. C and 0.55 per cent. Mn. The files are in use at Degerfors and have proved to be of good quality. The opinion of the file makers in regard to the behavior of the steel in working was that it was soft and good both for forging and cutting, but required very little annealing, although the carbon was as high as 1.30 per cent., and they were of opinion that 1.40 per cent. C would probably have been more suitable.

Fuel-Oil Specifications.

In consequence of the success that has attended the use of heating-value specifications in buying and selling coal there has arisen a demand for similar specifications, based on careful tests, to govern the purchase of fuel oils and the refined products of petroleum.

The Bureau of Mines has therefore issued Technical Paper No. 3, by Irving C. Allen, on "Specifications for the purchase of fuel oil for the Government," with directions for sampling oil and natural gas.

"These specifications, to be satisfactory," says the author, "should establish not only the heating value of the oil, and thus show, like the specifications for coal, the number of heat units the purchaser obtains for a given price per unit quantity of fuel, but also the physical character of the oil, its flash point and burning point and the quantity of extraneous matter it contains.

"It is evident that an essential feature of any such plan of purchasing oil in bulk is an equitable method of sampling deliveries—that is, a method that insures the taking of representative samples.

"The Bureau of Mines has received many inquiries from Government bureaus and from private concerns regarding fuel-oil specifications and the sampling of purchases. In reply to these inquiries the bureau issues this paper, which, although intended primarily for the guidance of Government officials, may be of service, the bureau trusts, to all persons who buy or sell fuel oil."

General specifications for the purchase of fuel oil are given as follows:

In determining the award of a contract, consideration will be given to the quality of the fuel offered by the bidders, as well as the price, and should it appear to be to the best interest of the Government to award a contract at a higher price than that named in the lowest bid or bids received, the contract will be so awarded.

Fuel oil should be either a natural homogeneous oil or a homogeneous residue from a natural oil; if the latter, all constituents having a low flash point should have been removed by distillation; it should not be composed of a light oil and a heavy residue mixed in such proportions as to give the density desired.

It should not have been distilled at a temperature high enough to burn it, nor at a temperature so high that flecks of carbonaceous matter began to separate.

It should not flash below 60° C. (140° F.) in a closed Abel-Pensky or Pensky-Martens tester.

Its specific gravity should range from 0.85 to 0.96 at 15° C. (59° F.); the oil should be rejected of its specific gravity is above 0.97 at that temperature.

It should be mobile, free from solid or semi-solid bodies, and should flow readily, at ordinary atmospheric temperatures and under a head of 1 foot of oil through a 4-in. pipe 10 ft. in length.

It should not congeal nor become too sluggish to flow at 0° C. (32° F.).

It should have a calorific value of not less than 10,000 calories per gram (18,000 British thermal units per pound); 10,250 calories to be the standard. A bonus is to be paid or a penalty deducted according to the method stated under section 21, as the fuel oil delivered is above or below this standard.

It should be rejected if it contains more than 2 per cent water.

It should be rejected if it contains more than one per cent sulphur.

It should not contain more than a trace of sand, clay or dirt.

Each bidder must submit an accurate statement regarding the fuel oil he proposes to furnish. This statement should show: The commercial name of the oil; the name or designation of the field from which the oil is obtained; whether the oil is a crude oil, a refinery residue, or a distillate; the name and the location of the refinery, if the oil has been refined at all.

The fuel oil is to be delivered f.o.b. cars or vessel, according to the manner of shipment, at such places, at such times, and in such quantities as may be required.

Should the contractor, for any reason, fail to comply with a written order to make delivery, the Government is to be at liberty to buy oil in the open market and charge against the contractor any excess of price, above the contract price, of the fuel oil so purchased.

Copies of this Technical Paper may be obtained by addressing, the Director of the Bureau of Mines, Washington, D. C.

Regulations on Papers for the Eighth International Congress of Applied Chemistry, 1912.

In the following we give the set of tentative rules on papers to be presented before the Eighth International Congress of Applied Chemistry. The Congress will be held in this country in 1912. Since the final rules are to be adopted during December of this year, any criticism of the tentative rules given

below should be made before Dec. 1, 1911, and should be sent to the secretary of the Congress, Dr. B. C. Hesse, 25 Broad Street, New York.

1. All papers must be in duplicate and legibly written, preferably typewritten.

2. Each sheet must be written on one side only and *not* on both sides.

3. Each paper must be accompanied by an abstract thereof, also legibly written, preferably typewritten, and which must be in duplicate.

4. Papers and their abstracts, both in duplicate, must be in the hands of the American Committee not later than June 30, 1912. All papers received prior to that time and accepted will be printed in their respective Sectional Volumes and distributed to such of the attending members of the Crongress as may desire them at or before the opening of the Congress. Papers received after that time, if accepted, will be printed, but may appear in an appendix which may or may not be ready by the opening of the Congress;. the Congress cannot then undertake to print them along with the papers of those sections to which they may be assigned.

5. All papers or like contributions must be as concise as possible, and must contain the full name and post-office address of their respective authors; further, what number, if any, of reprints of the paper or like contribution is desired.

6. Papers or other like contributions must be original and not elsewhere read or published, nor contributed or offered to any other society, association or publication for presentation or publication. The offer of any paper to the Congress is a tacit and understood agreement to the foregoing requirement. No paper should deal with historical matter any more than is needful for a proper understanding of the new subject matter presented, which subject matter, as far as practicable, should be of a date subsequent to June, 1909, the date of the Seventh International Congress of Applied Chemistry, except by special request. Nonconformity to this requirement may be a reason for rejection; a remedy herefor is offered in paragraph 18.

7. All authors, as a matter of course, agree not to publish their accepted papers in any other publication except as hereinafter provided, and, further, they automatically agree to abide by any final decision of the Congress with respect to such paper or papers, their presentation, discussion or printing.

8. The Congress obligates itself to have its final Report and Proceedings, including subject and authors' index, completed and ready for distribution on or before Dec. 31, 1912; in case those reports and proceedings be not ready for distribution by that date, authors of all papers received and accepted after June 30, 1912, may then publish in any journal or publication that they may elect. This refers only to the report and proceedings bound in paper; members desiring cloth-bound sets can obtain them at an advanced charge over the $5.00 membership fee; such advanced charge will be announced later, but will probably be $2.50; delivery of these cloth-bound sets will be about 90 days later than of the paper-bound sets. Authors of papers received before the close of June 30, 1912, may publish those papers in any publication they may elect after the paper is read or after the Congress has adjourned.

9. Authors of papers accepted and printed in full or in abstract will receive free of cost and all delivery charges, not to exceed fifty (50) reprints of each paper or abstract; additional copies of reprints can be had upon payment of the prices for such copies, which prices will be announced later. The Congress cannot undertake to furnish reprints of papers if the order for such reprints is not attached to the paper or abstract when received by the American Committee.

10. No paper offered to and accepted by this Congress can be at any time published elsewhere without giving credit to this Congress for such article or publication.

11. The Congress reserves the right to reject any paper or other contribution that may be offered to it.

12. Rejections by Sectional Committees will not be final; their decisions will be reviewed by the Committee on Papers and Publications, but rejection by that Committee will be final.

13. Authors of finally rejected contributions will be notified in writing of such rejection immediately after it has been made, and, as far as the Congress is concerned, such final rejection is strictly secret and confidential.

14. Any paper which is of a pronounced polemical, advertising or personal character may be thereby disqualified and for that reason alone rejected, regardless of whatever merit the paper may otherwise possess.

15. The Congress reserves the right to print the full paper only, or the abstract only, or the title only, in each case with the author's name and post-office address.

16. The Congress will not publish a list of rejected papers nor state what papers have been rejected; directly after the closing of the Congress all records relating to rejected papers and like contributions will be destroyed; any and all proceedings as to rejected papers or like contributions, so far as the Congress is concerned, will be strictly secret and confidential.

17. Authors are requested to state on the papers themselves their preferences for the sections in which they wish them to be read; the Congress will respect that request wherever practicable, but reserves the right to assign the paper to any other section that may be deemed more appropriate, and such disposition is final.

18. Authors will *not* receive printer's proofs of their papers or abstracts; nor will their papers or abstracts be revised after receipt by the American committees, except when express authority so to do is given in writing by the author and time and opportunity offer for such revision; authors of manuscripts so revised waive any and all right to complain of or rectify any such revision. (See paragraph 6.)

19. The time consumed in reading or presenting the substance of any paper by an author or his representative at a meeting of a section must not exceed ten (10) minutes; papers or presentations requiring more time than that must be suitably condensed so as to fulfill that requirement.

20. Anyone reading a paper of another's authorship must be fully equipped and prepared to defend the paper in discussion, and no one else will be permitted to read such a paper; an offer to read another's paper is an implied statement to the effect of such preparation.

21. In the absence of an author or his properly equipped representative the paper will be read by title only, and if there be any discussion it must be based upon the paper as printed, because neither the paper itself nor its abstract will be read; exceptions to this rule can be made only under regulations that may be adopted by each sectional executive committee.

22. Discussions of a pronounced polemical, advertising or personal character may be ruled out by the chair on that ground alone and not permitted to appear in the printed record; the ruling of the chair in such matters is final and is not subject to revision or appeal.

23. Participants in discussion will be given an opportunity of editing the manuscript reports of their remarks, but printer's proofs will not necessarily be submitted to them, although wherever practicable they will be so supplied.

The Reaction Between Hydrogen and Carbon Monoxide and the Effect of Steam on Carbon Monoxide at Elevated Temperatures.

In an interesting paper by Mr. Armand Gautier in the *Comptes Rendus*, vol. 150 (1910), page 1564, the author investigates the question of the possible reduction of CO by hydrogen, with formation of water.

Pure CO, mixed with three times its volume of hydrogen, was heated in a porcelain tube and the temperature recorded

by a Le Chatelier pyrometer. Many tests were made at temperatures from 300° C. to 1250° C.; the mixed and dried gases were circulating with a velocity of 450 cc. to 475 cc. per hour. The water formed was absorbed in a tube filled with P_2O_5, the CO_2 by potassium hydrate.

	H_2O per liter CO.		CO_2 per liter CO.	
Temperature.	mg.	cc.	mg.	cc.
400° C.	4.4	5.4	1.7	0.86
600° "	12.4	15.4	11.4	5.80
900° "	17.1	21.2	20.5	10.4
1220° "	22.2	27.5	19.2	9.7
1200° "	17.2	21.3	4.1	2.1

The results show that by heating CO in presence of hydrogen, water is actually formed. The reduction begins approximately at 200° C., before red-heat. The maximum formation of water is between 1100° and 1200° C.

During all the tests the volume of H_2O did not exceed 0.028 of the volume of CO used. At the same time the formation of CO_2 increases till 900°, and drops down when the temperature is increased above this point.

$$CO_2 + H_2 = CO + H_2O.$$

This equation also explains the stability of the water volume in this high-temperature interval, despite the reverse equation.

$$CO + H_2O = CO_2 + H_2.$$

Between 900° and 1000° the water and CO volumes may follow pretty closely the equation:

$$4CO + 2H_2 = 2H_2O + CO_2 + 3C.$$
$$4 \text{ vol.} \quad 2 \text{ vol.}$$

But no trace of carbon was found.

Two remarkable phenomena may be noticed, the decrease in CO_2 between 1200° and 1220° and a similar one during the period of formation of water.

By using a mixture of $CO + 3H_2$ and a consumption of ½ liter per hour, the author obtained at 900°, under the same conditions, the following results:

		H_2O per 1 CO.		CO_2 per 1 Co.	
	Temperature.	mg.	cc.	mg.	cc.
I.....	900°	17.10	21.2	20.5	10.4
II.....	900°	9.3	11.5	21.4	10.8

Test II gave some carbon according to the equation:

$$3CO + H_2 = H_2O + CO_2 + C_2.$$
$$2 \text{ vol.} \quad 2 \text{ vol.}$$

Test I showed no gain in carbon. During the latter the water volume was nearly twice the volume of CO_2, and the presence of some methan could be detected.

This equation may explain it:

$$4CO + 8H_2 = 2H_2O + CO_2 + 3CH_4.$$
$$4 \text{ vol.} \quad 2 \text{ vol.}$$

By heating a mixture of CO and hydrogen at 400° and above only a few thousandths per cent of methan were obtained whose presence had not been detected till then.

Gautier assured himself that during such a process no gas was formed which could be absorbed by bromine or condensed at a low temperature, and that no ethylaldehyde or formic acid was produced, except some traces of formol.

The test was repeated at 1200° with a mixture of $CO + 6H_2$ (1 liter per hour), and the following result was obtained:

$$H_2 = 99.8 \qquad CH_4 = 0.2.$$

At 1300° and 3 liters per hour:

$$H_2 = 98.65 \qquad CH_4 = 1.35.$$

The velocity of gas passing the tube seems, therefore, to prevent the destruction and prevents the formation of heavy hydrocarbons.

During this last test 8 per cent of the carbon had been transformed into methan. A perceptible formation of other hydrocarbons did not take place.

The results may be summed up as follows:

1. The reduction of CO by hydrogen begins approximately at 400°, resulting in the formation of water, carbon and some methan.

2. The mutual reactions are limited and equilibrium is reached at approximately 1000°.

3. The formation of methan is favored by a higher percentage of hydrogen and by a greater velocity to prevent the splitting off of CH_4 at red-heat.

4. Between 450° and 650° traces of formol are formed, but disappear again at 950°.

In regard to the effect of steam on CO at higher temperatures, Gautier made some preliminary tests, using 100 volumes CO with an excess of steam and received the following results:

$$CO_2 = 37.9; \quad H_2 = 38.9; \quad CO = 22.5.$$
$$N_2 \text{ (from air)} = .7.$$

To prove the probable appearance of formic acid and formaldehyde between 500° and 900° the test was repeated.

The steam condensed after leaving the apparatus contained in 2cc. —0.000005 formol at 550°, at 800° 0.000001 formol, and no formic acid could be found in 2.5 cc. condensed steam.

As to the practical application of the results of Gautier to our metallurgical processes, the reviewer is of the opinion that they may perhaps help to explain the question of formation of hydrogen and methan in blast-furnace gas, which problem has not yet been solved satisfactorily or completely.

They may also be of use in explaining the change in analysis of producer-gas in a long gas main and in the checkers of regenerators. Especially the marked influence of steam on CO seems worth consideration, since a considerable amount of steam is found in all producer-gases.

A similar case is known to the reviewer, where cold coke-oven gas was mixed with hot producer gas in the main of a heating furnace. A report on this is reserved for later.

Sharpening of Tools by Electrolytic Etching*

BY ERICH SCHNECKENBERG.

The attack of an acid on a metal is increased if an electric current is passed simultaneously from the metal into the acid. The current at once carries away all the metallic particles dissolved by the acid, so that the metallic object always presents a clean, fresh surface for the attack of fresh acid.

The current for this purpose may be produced in either of the following ways: 1. A piece of gas retort carbon is placed in the same acid as the metallic object, and the carbon and the metal are connected by a wire outside of the acid, so that the whole forms a short-circuited galvanic cell, in which the current flows from the metal through the acid to the carbon. 2. The metallic object may be connected with the positive pole and another electrode in the acid with the negative pole of some external source of direct current.

In electrolytic etching the attack of the acid upon the tool can be centered and localized on any desired part of the tool, in any direction and strength. All that is needed for this purpose is to form the negative electrode or cathode in a suitable manner and put it in the right position. For in electrolytic etching the quantity of metal etched and removed at any spot is dependent entirely (aside from the time of etching and the concentration of the etching liquid) on the current density at that spot in amperes per square centimeter, and this current density depends on the resistance of the liquid along the various current lines (or tubes of lines) between the positive object and the negative electrode.

For tubes of lines of current of the same cross section this resistance is determined by their length and by the geometrical and physical characteristics of the two opposite surface areas of the metallic object and the negative electrode. These conditions can always be arranged in a suitable manner on the basis of some experiments.

It is possible, for instance, to devise suitable arrangements for sharpening any sort of tool, precision millers, etc. The material for the negative electrode, as well as the acid, must

*Translated from *Werkstattstechnik*, August, 1911.

be chosen accordingly, so as to prevent too rapid destruction of these devices. First of all, I will show how this work should not be carried on.

Several years ago Barthel, in New York, tried the following experiment: He placed some tools, first freed of rust and grease, into a mixture composed of 100 parts of water, 3 parts of 66 per cent sulphuric acid and 6 parts of 44 per cent nitric acid. In the case of tools with two flat surfaces he placed a flat carbon plate opposite to each side surface of the tool. Where round files, millers and drills were to be sharpened he used a cylindrical, hollow carbon tube. When he connected carbon and tool by a wire outside of the acid an electric current passed from the tool through the acid to the carbon.

The etching was, however, different at different spots of the objects, according to the distance and relative position of metal and carbon. Further the carbon also was attacked by the acid; the acid became black in color and the tool was covered with a layer of slime. The tool had, therefore, to be removed after a short time and cleaned, then again placed back and adjusted in position, and etching was resumed, repeating the entire process until the tool had been etched to sufficient sharpness. Then it was rinsed in water, cleaned with a brush, laid in lime water in order to neutralize any possible acid residue and dried with sawdust.

In this form the process was too complicated for workshop use. Moreover, the strength of the current always differed, depending on the surface characteristics, size and number of tools dipped and of the carbon pieces. Since the necessary time of etching depends on the current strength the sharpness of the tool attained by the process can only be estimated from the time consumed, if the current strength is always uniform. The strength of the current could, of course, have been brought to a standard value by connecting several resistances in the exterior circuit, and cutting out some of them when the emf between tool and carbon decreased. But this would have meant that since at the beginning of each etching period the emf was greatest, decreasing then rapidly, a large portion of the emf was to be consumed in the resistance. This, however, had to be avoided since the energy in this cell was anyway rather low.

The introduction of current from an external source (by method two) offers a much simpler and safer process. A storage battery giving 2 to 3 volts is suitable. With this method it is, of course, not necessary to use the most powerful electronegative element, carbon, in order to obtain a sufficiently high emf, but any suitable inert material can be chosen, without any consideration of its position in the contact series. The result is a good, proper etching process for shopwork use.

Grathwol, in Herisau, for example, succeeded in sharpening very dull files within one hour in this way. A patent was allowed him for a file-etching process in which the negative electrode, the cathode, is made of lead, and the electrolyte is sulphuric acid of about 1.18 specific gravity.

Since the quantity of metal etched away at any one place increases with the etching time, it is possible to subject an article to an etching process varying in strength at different places by withdrawing the object gradually from the bath.

Hall and Thornton, in Birmingham, sharpen metal tubes and rods to a point by connecting them with the positive pole of a battery, dipping the end to be pointed into the acid solution connected to the negative pole, and then slowly and gradually withdrawing it from the bath. In this way the rods become thinner and thinner the nearer to the end. This process can be applied to the sharpening of needles, so that thousands of these can be sharpened at one time. All that is needed is a simple trough, and instead of raising the needles gradually from the trough the level of the solution in the trough may be gradually lowered. Another advantage of this method is that the solution in which the metal is dissolved may afterwards be used for electroplating purposes.

Another modification of tool sharpening by electrolytic etching is possible if the tool is brought in contact with the electrolytic acid solution at those places only which are to be etched off, instead of placing the whole tool into the acid solution. This is accomplished in the following way: A mold is made of porous calcined clay corresponding to the desired form of the metal to be etched and is saturated with the etching solution and connected with the negative pole of a battery, and pressed with a slight pressure against the tool, which is connected to the positive pole. After a short time the mold electrode is removed, the tool brushed free of the solution, the mold electrode again dipped into a fresh acid solution, and again pressed against the tool. This method was used successfully by a large electrolytic engraving firm in Leipzig to etch reliefs into metal.

To what accuracy and precision this electrolytic etching process can be brought is proved by the die process of Burdett in Hartford, who etched whole designs, inscriptions and patterns into knives or other metallic surfaces. The metallic surface is covered with a thin layer of a mixture of 10 liters (2.6 gal.) naphthalene, 0.125 kg. (0.27 lb.) carbon disulphide, 2 kg. (4.4 lb.) powdered resin and 1.5 kg. (3.3 lb.) copper chloride. A thin coat of this is put on, and by means of a die which had first been cleaned with a weak caustic potash solution the design is pressed into the coating. Then the particles pressed out by the die are washed off with water; the entire coating is covered with a weak solution of ammonia or brine, a clean metallic surface laid on top and connected to the negative pole of a battery, while the metallic object itself which is to be etched is connected with the positive pole.

Then the electric current passes between the object and the metallic surface, through the ridges cut into the resin coating, so that after a short time the pattern is etched in sharp, clear, deep lines, after which the surface is washed with soda solution. This process also needs only very simple devices to make it fit for operation on a large scale.

Summary.—Electrolytic etching, in which the tool forms the positive electrode for the etching solution in contact with it, permits rapid etching in a simple way, on a definite spot, in a definite direction and strength, depending on the position of the negative electrode, which determines the current flow. It is much better to use an external battery as source of current, instead of generating a current internally from a reaction between tool, acid and carbon. An example for the practical application of tool sharpening by the electrolytic etching method, the electrolytic file sharpening, needle-point sharpening and relief production as well as die sinking work may be mentioned.

Notes on Chemistry and Metallurgy in Great Britain.

(*From our Special Correspondent.*)

Drawn-Wire Tungsten Filament Lamps.

Your correspondent was one of a party of representatives of the technical press who visited the British-Thomson-Houston Company's works at Rugby in August to inspect that company's lamp works and to be one of the intermediaries whereby the announcement was to be communicated to all concerned that drawn-wire tungsten filament lamps were now available in commercial quantities. Possibly a similar announcement and demonstration has been made on your side by the British-Thomson-Houston Company's parent, the General Electric Company.*

*As is well known to our readers, the new drawn-wire tungsten lamp is the outcome of a long series of researches on processes for producing ductile tungsten, carried out at the Research Laboratory of the General Electric Company, at Schenectady, N. Y., under the direction of Dr. W. R. Whitney. The successful work was the result of the efforts of a group of operators and research men, foremost of whom were Dr. C. G. Fink and Dr. W. D. Coolidge.—Editor.

One saw much that was interesting—the new lamps will stand an enormous amount of over-running. It was distinctly novel after one's close acquaintance in the past with carbon filaments, or more recently with squirted tungsten filaments, to see bobbins of tungsten wire from which requisite lengths are cut off as readily as is double cotton-covered wire for the use of armature winders. This is not the place to discuss small structural improvements which, apart from the filament, distinguish this lamp from its predecessors. Neither, and this is the more disappointing, can your correspondent say anything about processes conducted behind locked doors whereby tungsten in its powdered form is converted into a drawn wire. It is a triumph in science, lying on the borderland between chemistry and metallurgy, of great commercial value and more scientific interest. In course of time information, at first no doubt more or less erroneous, must leak out. Meanwhile one can only speculate as to how this has been effected.

A Month of Industrial Unrest.

Early in August the stress of a political crisis masked the symptoms of the labor situation. Unrest and discontent had long been prevalent in South Wales, there had been strikes in the seafaring trade, and employers of labor had recognized that the present trade expansion was liable to be throttled by labor disputes. Just as the Parliamentary situation was clear of all save the recrimination over the *fait accompli*, the trade of London, Liverpool and Leith was paralyzed by strikes of dockers and carmen. On top of this followed a national railway strike, with extensive rioting in South Wales and destruction of property in the Midlands. Considering all the facts of the suddenness of the calamity, the forces of organized civilization in the shape of constabulary and soldiers proved surprisingly adequate. Concessions of increased pay to dock laborers, increased pay and shorter hours for carters and lightermen (certainly justified in these cases) together with the appointment of a Royal Commission to go into the question of the Railway Conciliation Boards have calmed down the situation. The industrial machine therefore is getting into its stride again.

Short though the trouble has been, the loss has been very serious, and will be reflected in balance sheets and trade returns at a later date. Cotton mills and engineering works have been stopped for raw material, contracts have been lost, and the general dislocation will take a lot of adjustment. Of course, the effects of increased wages will lead to a redistribution of the charges incurred thereby; and we shall, unless the index-numbers of the world's food prices take a downward movement, encounter further demands.

The immediate economic questions sink into insignificance compared with the greater sociological questions now partly unveiled to the gaze of the man in the street. To the individualist it is an unwelcome fact that the power of the organized proletariat, by means of a general strike, to attempt to force say state ownership of mines or railways upon the general community brings the climax to the coming industrial struggle thirty years nearer than was anticipated thirty days ago.

The Formation of Sludge in Oil-Cooled Transformers.

An article in the *Times* of the 9th of Aug., by an engineering correspondent, discusses the general aspects of the problem presented to users of oil-cooled transformers, and to the manufacturers of oil for insulating purposes by the deposition of sludge, particularly on the windings and cores. The author says:

"Of the hypotheses advanced to account for the trouble one ascribes most of it to the absorption of vegetable acids from varnishes nominally, but not actually, oil resisting, the balance being attributed to the ingress of dust, which is attracted electrostatically to the windings, and to the thickening of the material through chemical action. The other hypothesis is purely a thermal one, and predicates that a fractional distillation of the oil takes place through high temperatures. In investigating the problem in any case when the deposit has reached such a stage as to call for a renewal of the oil, the difficulty is encountered that very often little or nothing is known of the initial qualities and subsequent history of the oil, while not sufficient is known of the temperature ranges in the transformers themselves. Probably there are cases which the first hypothesis meets, but there are undoubtedly others to which it cannot apply, and for which the thermal hypothesis is necessary in order to afford a complete explanation of the phenomena observed."

He then proceeds to outline the changes produced in transformer oil by baking at temperature from 230° F. to 270° F., details of which can be found in a paper by Digby and Mellis, published in the Proceedings of the Institution of Electrical Engineers, 1910. The results then given were considered to be mainly of academic interest, but, in connection with the thermal theory of formation of sludge, those results, together with further investigation, might be of very practical value.

The time which elapses before any noticeable deposit of sludge occurs varies considerably. The colloidal deposit has been observed after only six months running of a transformer; but generally the accumulation of a serious deposit occupies from three to six years. Observations with thermo-couples have shown that at points where the core-plates are burred the temperatures are from 15° F to 25° F. higher than the warmest portions of the oil remote from such positions; and, of course, the deposition of sludge prevents the normal distribution of heat by convection and gives rise to a largely extended area in which the heat is unduly augmented.

Analyses of sludges have yielded some very interesting information as to their various metallic and organic constituents; and the research is being continued in the hope of elucidating the causes of a trouble the importance of which is now recognized.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the seven months ended the 31st of July gives the value of the imports of iron and steel, including manufactured articles, at £6,501,659 and of exports as £25,373,018, and the respective increases over the value for the corresponding period of 1910 are £1,511,050 and £485,396, but in July, 1911, there was a deficiency of £727,055 as compared with July, 1910. For other metals, including manufactured articles, the imports were of the value £16,044,504 and the exports £6,322,193, and the increases as compared with the value for last year are £2,607,777, and £463,622. Imports of electrical goods, exclusive of machinery and line wires, amounted to £795,065, a decrease of £46,026, while exports were £1,542,133, or a decrease of £420,556. The value of imports of machinery was £3,575,125, with exports £17,952,018, showing increases of £956,676 and £1,100,728, respectively. Importations of new ships were put at £48,203, and exports at £3,220,582, the imports having improved by £40,856, and the exports having decreased by £1,636,275.

The British Association.

At the meeting at Portsmouth on the 31st Aug., Professor H. H. Turner in his presidential address to the Mathematical and Physical Science section on "The Characteristics of the Observational Sciences" declaimed against the dangerous—he might have said pernicious—doctrine that work done before the formulation of a law was in any way inferior: and instanced the valuable results of W. W. Campbell's observations of the line of sight velocities of stars at Lick Observatory, Professor Schuster's analysis of facts bearing on the periodicity of sunspots, Groombridge's observations of stars, and M. Teisserene's meteorological observations with balloons.

In so doing the professor was only voicing the opinion of the practical scientist against the tendency only too prevalent at present among some possessing high scientific repute to theorize on any and every subject, formulate a "law" from their questionable speculations, and then cast about for apparent

facts which may by hook or crook be distorted into some semblance of evidence in support of their precious "law." Science is not gaining anything by the substitution of such methods for the true and nature-guided work of men like Davy, Faraday and Tyndal; or we should not find our ears assailed at assemblages of "scientific" men with such onological absurdities as the expression, pretty frequently used of late, "atoms of electricity." A certain person of considerable reputation as a scientist several years ago at an annual meeting of one of the leading London scientific societies uttered the exhortation "therefore, gentlemen, let us DREAM," and it is only too apparent that there are many who do not need such advice to cause them to give preference to methods bordering on the metaphysical.

A Fourth Recalescence Point in Steel.

John O. Arnold (Intern. Zeitsch. Metallog., 1911, 1, pp. 192-203), as a result of comparing cooling curves of steels of varying carbon content with curves of iron containing no carbon, has found that an evolution of heat occurs with steel between the Ar_3 and the Ar_1 points. The greatest disengagement of heat is obtained with steels having a carbon content of about 0.45 per cent., which would consist of approximately equal proportions of pearlite and ferrite. This recalescence the author supposes to be due to constitutional segregation of the solid solution with formation of microscopic distinct masses, beginning at Ar_3 and ending at Ar_1.

Cementation of Chromium Steels.

F. Giolitti and F. Carnevali (Atti. R. Accad. Sci. Torino, 1911, 46 pu. 558-568) have experimented with a chromium steel containing 2.33 per cent. Cr, 0.41 per cent. C, 0.15 per cent. Si, and 1.02 per cent. Mn, by cementation in the same way as that adopted for nickel steel (Journ. Chem. Soc., 1911. ii., p. 609). The results of five hours' treatment in ethylene at 950°C., and at 1050°C., treatment with carbon monoxide for the same time at the same temperatures, and with various mixtures of carbon monoxide and carbon dioxide at 1000° and 1100° indicate that the differences observed in the action of these cementing media on carbon steels are produced similarly in chromium steel. The presence of chromium tends to augment the maximum concentration of carbon in the zone of cementation in comparison with the maximum in carbon steel similarly cemented.

Determination of Chromium in Chrome Iron Ore.

Otto Nydegger (Zeitsch. Angew. Chem., 1911, 24, pp. 1163-1164) takes 0.5 gramme of the powdered ore with 5 grammes of a fused and powdered mixture of two parts of anhydrous sodic carbonate with one part borax glass, and heats the mixed ore and fusion-mixture in a platinum crucible over a bunsen flame until all the ore is taken up—from an hour to two hours being required. When cool the mass is dissolved out with 300 cc. or 400 cc. of water containing one volume per hundred of sulphuric acid. Any residual unoxidized chromium is converted into chromate with 0.2 gramme of potassic persulphate, which precipitates, as peroxide, any manganese that may be present. Excess of persulphate is decomposed by boiling for half an hour or so, the MnO_2 is filtered off, and the chromium is estimated volumetrically by ferrous ammonium sulphate with a potassium ferricyanide indicator.

Market Prices During August, 1911.

Copper has been fairly quiet throughout the month. Opening at £56.7.6 it rose to £57 per ton on the 14th, since dropped to about £56.5.0, and closes at £56.2.6.

Tin dropped smartly away from its opening price until it reached £189.10.0 on the 14th. Had risen again to £191.0.0 by the 18th; since irregular, and closes at £189.15.0.

Lead has been fairly steady, with rise of 5/- on the 18th. Opened at £14.5.0, closes at £14.10.0.

Haematite shows a sharp rise from £3.1.3 on the 10th to £3.3.0 on the 18th. Since fell again and closed at £3.2.9.

Scotch Pig rose in sympathy with iron generally from £2.12.1 to £2.13.8 by the 18th, closing at £2.13.3.

Cleveland opened at £2.6.10, showed a sharp drop on the 16th to £2.6.3, afterwards rising to £2.7.6, and closing at £2.6.11.

Rubber (Parafine) has not exhibited any marked changes, was at 4/6½ on the 16th and 4/9 on the 23d; closed at 4/10.

Alum, lump, loose, per ton	£5.12.6
Antimony, black, sulphide powder, per ton	22. 0.0
Borax, British refined crystal, per ton	16. 0.0
Copper Ore, 10 to 25.2, per unit	9/6 to 10.0
Copper Sulphate, per ton	19. 5.0
Carbolic Acid, liquid 97/99 per cent, per ton	0. 1.6
Creosote, ordinary good liquid, per gal	0. 0.2⅜
Camphor, 1 oz. tablets	0. 1.9½
Caustic Soda, Ash, 48 per cent, ordinary, per ton	5.10.0
Litharge, flake, per ton	16. 0.0
Mica, small slab, original cases, per lb	0. 2.0
Naptha, solvent, 90 per cent, 160° C, per gal	0. 0.8
Petroleum, Russian Spot	0. 0.5½
Sal Ammoniac, lump, firsts, delivered, per ton	42. 0.0
Sulphate of Ammonia, f.o.b. Liverpool, per ton	14. 5.0
Sulphur, recovered, per ton	5. 0.0
Shellac, Standard T. N., Orange Spots, per cwt	4. 4.0
Tin Ore, 70 per cent, per ton	£120 to 122. 0.0
Hydrochloric Acid, per cwt	5.0
Platinum, per oz	8.12.6

Differences noted are:

Rise.

Creosote, per gal	£0. 0.0⅛
Carbolic Acid, per gal	0. 0.2
Litharge, per ton	0. 5.0
Sulphate of Ammonia, per ton	0. 5.0
Tin, per ton	0.15.0
Lead, per ton	0. 5.0
Haematite	0. 1.4½
Scotch Pig	0. 0.5
Rubber	0. 0.2

Fall.

Copper Sulphate, per ton	£0. 8.0
Naptha, solvent, per gal	0. 0.1
Tin Ore	3. 0.0
Copper, per ton	0. 5.0

Newcastle Meeting of the British Institute of Metals.

The autumn meeting of the (British) Institute of Metals was held at Newcastle-on-Tyne on Sept. 20, 21 and 22; that is, practically simultaneously with the Toronto meeting of the American Electrochemical Society.

We reserve an account of the excursions, visits and social functions for our next issue, but give herewith concise abstracts of the papers presented.

This young but exceedingly active society must be congratulated on the high intrinsic value of the papers presented at its meetings. The society continues to live up to the high standard set right at its start and it will be seen that the last meeting was no exception.

The headquarters of the Institute, with the office of its very active secretary, Mr. G. Shaw Scott, are at Caxton House, Westminster, S. W. London.

Alloys of Copper and Tin.

A paper by Mr. J. L. Haughton and Prof. T. Turner dealt with "volume changes in the alloys of copper with tin"

The work of which Mr. Haughton presented an account may be regarded as a continuation of previous researches on the changes in length which occur in a cast bar during and after solidification. To these investigations Picken, Simpson, Murray,

Hague, Ewen and Coe (former research students in the University of Birmingham) have contributed. The method employed has been that of applying an extensometer to one end of a cast bar while the other end was kept fixed. With a single substance, such as a pure metal or a definite compound, the extensometer does not appear to act until immediately after solidification has occurred, and contraction thereafter proceeds regularly. But where internal changes take place due to such causes as the separation of a constituent, or of a eutectic, from the solid matrix, such changes are indicated by changes in length of the bar. The separation of a constituent is generally accompanied by an expansion, while if the interval between the solidus and the liquidus is considerable the expansion of the cast bar is also generally large, and it extends over a considerable interval of time.

The apparatus employed has been modified by the author in order to allow a single operator to observe both pyrometer and extensometer readings at the same instant and to record these results by means of a chronographer on a paper tape. In a few cases tin-copper alloys have been already examined by Ewen and by Wüst, and it is desirable to note that the agreement between the author and these previous workers is generally quite satisfactory.

The paper clearly shows that there are four maxima in the expansion curve and that three of these, with about 10 per cent, 46 per cent and 65 per cent of tin respectively, agree with the "crystallization interval" as determined from the accepted equilibrium diagrams. The maximum observed at 30 per cent of tin is not easy to explain, though it may be pointed out that the equilibrium diagram at this point is exceptionally complicated and that perhaps the "crystallization interval" should extend down to 500° C., in which case the maximum would fall into proper order. Mr. Haughton records the fact that certain of the intermediate alloys of the series are remarkably insoluble in acids and ordinary solvents. Hence the analysis presented unexpected difficulties. These were overcome by the use of bromine and this simple and direct method of solution may be of use to other investigators of the tin-copper series.

Alloys of Aluminium and Zinc.

A paper by Dr. Walter Rosenhain and Mr. S. L. Archbutt, both of the (British) National Physical Laboratory, dealt with "the alloys of aluminium and zinc."

The paper describes a detailed investigation of the constitution of the alloys of aluminium and zinc which was undertaken in connection with an extended research on the light alloys of aluminium—the latter research being carried out by the authors on behalf of the alloys research committee of the Institution of Mechanical Engineers.

The present paper describes the detailed thermal and microscopic study of the alloys. A long series of cooling curves were taken at a very slow rate of cooling, and in many cases the ingots of the alloys were subjected to prolonged annealing before the cooling curves were taken. The microscopic examination of the slowly cooled alloys was supplemented by the study of specimens which had been annealed and quenched at various definite temperatures.

The results are embodied in an equilibrium diagram which differs considerably from the diagram of Shepherd, which has hitherto been accepted; these differences result from the discovery of the existence of the definite compound Al_2Zn_3 which has a stable existence only between 443° C. and 256° C.; in the majority of the alloys this compound is only slowly formed by a reaction between crystals of zinc and an aluminium-rich mother liquor at 443° C. At 256° C., however, the compound again decomposes into practically pure zinc and an aluminium-rich solid solution; this decomposition is, however, of special interest because it produces a finely laminated constituent closely resembling the "pearlite" of carbon steels. The research has been carried out in the metallurgy department of the National Physical Laboratory.

Non-Ferrous Metals in Railway Work.

A paper by Mr. George Hughes dealt with "non-ferrous metals in railway work."

In this paper the author has placed on record some of the methods of working and uses pertaining to the non-ferrous metals in locomotive and carriage construction.

The use of copper is described in the shape of plates, stays and tubes, and analyses, tests, specifications, together with life of material and instances of wear and failure, are described.

Copper fireboxes on the Lancashire & Yorkshire Railway have a life of from 10 to 16½ years, an average mileage for the older engines with 140 lb. pressure being 310,000 miles. Engines with 160 lb. pressure run from 235,000 to 272,000 miles. Individual tube plates show a life of 7¾ years.

Workshop methods, such as patching of copper firebox plates, brazing copper steam pipes, melting and mixtures of brass alloys, are dealt with, with the object of eliciting a practical and scientific discussion.

The molding and coring of an injector casting are referred to.

The author states that brass slide valves of the unbalanced type will run 1000 miles with a wear of 0.035 in. on the face. Balanced valves wear 0.0053 in. per 1000 miles, while even better results are showing from a modified form of valve for superheat designed by the author.

A short account of the bearing metals used under the name of white metals is included, together with a note of non-ferrous metals and alloys used in the railway carriage department.

Corrosion of Brass.

A paper of Mr. Paul T. Bruhl dealt with "the corrosion of brass, with special reference to condenser tubes."

It is urged by the author that so important a subject as the corrosion of brass by sea water, the neglect of which must mean no slight expenditure, should induce steamship companies to keep records bearing on the subject. It has been this lack of data that has hitherto retarded the solution of the corrosion problem. All the companies with whom correspondence was opened possessed "no definite information."

The conclusions arrived at by the author are:

1. That the presence of air or an increase of temperature up to a certain point accelerates corrosion.
2. That iron, nickel and small amounts of lead are injurious; tin up to 1 per cent, large amounts of lead and aluminium are useful in diminishing corrosion.
3. That the inlet pipe and the condenser plates should preferably be made of brass.
4. That the condenser should be protected against stray currents.
5. Protective coatings are not recommended.
6. The importance of "spills" cannot be exaggerated.
7. That the tubes should be flushed with clean water after use.

Hard-Drawn Copper.

A paper by Mr. D. R. Pye discussed "the mechanical properties of hard-drawn copper."

In this paper the lack of any satisfactory definition of standard hard-drawn copper is pointed out. Although the (British) Post Office specification is highly satisfactory for the sizes of wire used in telephone and telegraph construction, it is quite unsuitable for larger sizes. The various definitions adopted from time to time are reviewed, and it is suggested that the one at present adopted by the Engineering Standards Committee, viz.: "The term hard-drawn copper wire shall apply to copper wire which does not elongate more than 4 per cent on a gage length of 10 in. when broken by tension," though not contradictory of the properties of hard-drawn copper, is unsatisfactory since a specification should preferably fix a *minimum* elongation as a safeguard against the wire becoming brittle. An important point is that tensile strength per square inch and elongation at fracture both depend on the diameter of the wire.

Experiments are described confirming a suggestion made by Mr. A. P. Trotter that this tensile strength per square inch diminishes with increase of diameter according to a linear law. It is also shown that the elongation at fracture for similarly manufactured wires depends very much on the diameter, being considerably greater for larger sizes of wire.

In view of these experiments it is suggested that a satisfactory definition of hard-drawn copper wire would fix a minimum tensile strength per square inch given by the formula $T = 30 - 20\ D$ and a minimum elongation per cent at fracture given by the formula $e = 5\ D$ where D = diameter in inches.

This specification would hold good for all sizes up to 0.5 in. diameter. Finally this suggested definition is compared with the specifications of the American Society for the Testing of Materials, and an appendix is added, giving the results of some recent experiments in America showing the effect of hard-drawing upon the electrical properties of copper.

Electric Conductivity and Constitution.

A paper by Dr. W. M. Guertler dealt with "the electric conductivity and constitution of alloys."

The aim of this paper, which is one mainly of theoretical interest, is to call attention to the property of electrical conductivity in its bearing upon the practical determination of the constitution of alloys.

The so-called "equilibrium diagram" for alloys is now almost universally accepted as the basis for any systematic study of the properties of alloys, and hence it is of great importance to be able to construct it accurately, and to have a thorough knowledge of it. The relationship between concentration, temperature and any given property—electrical conductivity, specific volume, magnetism, etc.—is best and most clearly brought out by the employment of a co-ordinate system in those dimensions. Thus taking concentration and temperature as the base, we can erect ordinates from it whose lengths are proportional to the numerical values of the property under consideration, and so obtain a series of intersecting curved surfaces, each of which corresponds to a particular phase-region of the equilibrium diagram.

The converse problem is one of great importance, viz.: that of obtaining the equilibrium diagram (when it is unknown) by projection from these surfaces on to the basal plane.

The best known method, and one which has been hitherto and is still most universally employed for the determination of the equilibrium diagram, is that known by the name of "thermal analysis," in which special account is taken of the variations in the "total heat" of the systems under consideration.

The author discusses at length the limitations of "thermal analysis," and in particular explains why it fails in those cases where, as so often happens in practice, a state of complete stable equilibrium is unattainable within a comparatively short interval of time.

The respective merits of electrical conductivity and of specific resistance measurements are set forth and illustrated by numerous examples from practice.

Solid Solutions.

A paper by Mr. C. A. Edwards gave "further notes on the nature of solid solutions."

The main object of this paper is to deal with a few points of interest and difference raised in the discussion on a previous paper by Mr. Edwards on the same subject, and also to stimulate further discussion on a matter which is not purely of scientific interest. The main conclusion in the previous paper was that "solid solutions of two metals or intermetallic compounds are intimate crystalline mixtures and while the crystals are so small that the mass appears quite homogeneous they are, nevertheless, sufficiently large to retain their identity." The principal objection to this view was that it is opposed to the phase rule. A general review of solutions—gaseous, liquid and solid—is given in the present paper, after which review the author concludes that the above idea does not conflict with the phase rule, and in fact the phase rule was never intended to extend in its application to the internal nature of a constituent.

As regards solutions in general Mr. Edwards holds that gaseous, liquid and solid solutions are members of the same family, the different states of these members being due to differences in their molecular aggregation.

Failure of a Brazed Joint.

A paper by Dr. H. Louis discussed "the failure of a brazed joint."

The paper gives a brief account of the investigation of the cause of a failure in the braze of a steam pipe on a steamer, undertaken at the instance of the (British) Board of Trade. The author shows that the failure was due to corrosion following certain well defined lines in the brass, and he traces the cause of these lines to the presence of small quantities of lead and tin in the original brazing spelter. The lead-tin alloy separating out between the crystals of brass formed planes of weakness that gave access to the corroding solutions, and this brought about the gradual corrosion of the entire brazing material.

SYNOPSIS OF CURRENT LITERATURE.

Iron and Steel.

Electric Steel.—It is not often that the results of experimental work carried out on the scale of that described by A. Müller in *Stahl und Eisen*, July 20, finds its way into print. The work was that carried out by a German steel company on a three-ton Girod furnace preparatory to equipping their plant with electric furnaces. That connections to such a furnace can play such an important part in the life of the lining and the power consumption of it is well emphasized in the changes made by the author in question on his furnace. Where one connection was made to the top electrode and another to the bottom electrode it was found that the arc was thrown strongly against the opposite side of the furnace to the conductors. The top conductor was then split into two and led to the electrode on opposite sides, a single conductor being used on the bottom electrode. With this system the arc was blown toward the line. The final solution of the problem was to split both the conductors, bring them up to the furnace many feet underneath and then rise vertically, a pair on each side of the furnace, carrying the conductors leading to the top and bottom electrodes just as far as possible side by side. To increase this length the bottom electrodes were in electrical connection with the shell of the furnace so that connection with them could be made high up on the side of the furnace. This splitting of the conductors keeps the arc central and saves half the lining and all trouble with blowing out of the arc. Even the shell of the furnace was finally made of steel of low permeability. The energy saving in the conductor system and shell as revised amounted to 10 per cent over the first installation. The flat circular arc now carried on the furnace means an even greater saving of electrode, as previously when it was thrown off to one side it caused very uneven consumption and consequent waste of electrode.

The present system of working is to charge the furnace with about three tons of open-hearth metal and put on a current of 4000 to 6000 amperes. A highly oxidizing slag is formed and all the phosphorus and some of the sulphur is removed during this oxidizing period. The bath is then deoxidized under a new slag formed of lime, sand, ferro-silicon and petroleum coke or scrap electrode ends. This removes sulphur down to 0.01 per cent. On an average five charges are worked

in 12 hours, though this is dependent on the amount of impurities to be removed.

With the original form of two conductor connections where the arc was thrown to one side, the silica roof over that side, even with air cooling, lasted only 20 heats. With the four conductor system the roof now lasts 60 to 70 charges. Experiments are not yet completed as to the economy of using other materials in place of silica for the roof.

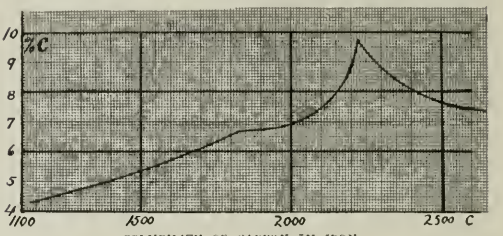
The side walls were first formed by ramming a mixture of magnesia and tar around a core shaped like the inverted frustrum of a pyramid. Such a lining cost $175. It was later determined that dolomite would most economically replace the magnesia so that all linings are now made of this material. They last 120 charges and cost $85. The total cost of lining, including repairs to slag line between heats, amounts to 30 cents per ton of steel. The area of the ends of the steel electrodes in the bottom bears the ratio of 0.05 to 80 to that of the whole bottom.

The bottom electrodes are water cooled in the ends projecting outside of the furnace. The dolomite bottom is rammed in between them with an air tamper, such a bottom lasting 1000 heats in a furnace run very intermittently. A calorimetric determination of the heat carried out in the cooling water of these bottom electrodes gave 10.1 kilowatt-hours for the 130-minute run, or about 1.01 per cent of the total electrical energy supplied, or 2.9 kilowatt-hours per ton of steel produced. The cooling water used in the top electrode carried out 36.7 kilowatt-hours, 3.65 per cent of total energy supplied, or 10.5 kilowatt-hours per ton of steel. The paper contains numerous other determinations of the power lost by stoppage, energy requirements for different weights of charges, etc., most of which are given in the form of charts.

The running of this small experimental furnace has thrown much light on the nature of many reactions in the refining of the steel. In the oxidation period carbon is but slowly removed, manganese more rapidly, phosphorus very rapidly and about 20 to 35 per cent of the sulphur. The sulphur appears to be removed in three ways: In the oxidation period by the action of the iron oxide in the slags; in the deoxidation period by solution in the high lime slags and volatilized as silicon sulphide. It has also been noted that oxides, slag emulsions and gases have a much more important bearing on the physical properties of the finished metal than the sulphur and phosphorus present.

Regenerative Furnaces.—A novel change in the construction of regenerative furnaces and one which has proven successful in practice is described by Fr. Bertiardt in *Stahl und Eisen*, July 3. The flues carrying the gases from the chambers to the furnace ports are led up separately to the furnace, and distinct from each other. The flues resemble flying buttresses so typical of Gothic architecture in the way they stand out distinct from the furnace. The gas is admitted to the hearth of the furnace in a nearly horizontal stream, while the air is brought down vertically through ports in the roof of the furnace. This construction does away with the narrow division wall between the air and gas ports in the usual style furnace, a source of great trouble at all times. A 17-ton furnace which formerly made on an average of 280 to 300 heats before requiring shutting down for repairs when provided with this new construction ran 630 heats. In a 40-ton furnace the new gas and air flues lasted 1500 to 1800 heats. The ports averaged 280 to 330 charges before requiring any repairs, and after that time the roof was in 50 per cent better condition than that of a similar furnace provided with the old-style ports.

Solubility of Carbon in Iron.—A new determination at high temperatures by Ruff and Goecke appears in *Metallurgie*, July 22. Iron was heated in a carbon crucible in a carbon resistance furnace. After standing at various temperatures for some time it was dropped into ice water and the carbon content determined. Temperatures were measured by the Wanner pyrometer. The results are shown in the accompanying curve, the per cent of dissolved carbon forming



SOLUBILITY OF CARBON IN IRON.

the ordinates and the temperature in centigrade degrees the abscissas. The change in direction at 1837° corresponds to the formula Fe_2C, the maximum at 2220° to the formula Fe_2C.

Copper.

Leady Copper Matte.—The treatment of leady copper mattes has always presented a serious problem to the metallurgist. Some new experiments which have led to the introduction of a new process are described by Robert Hesse in the June numbers of *Metallurgie*. The author started on the assumption that by smelting a leady matte with oxides of copper, lead and iron the lead could be removed in the metallic state, leaving the copper in the form of an enriched matte, the reactions involved being similar to those in the roast-reaction processes in use in lead and copper metallurgy. Numerous experiments proved the fallacy of this line of work as the only results were the production of a lead-copper alloy, and even that only when the amount of oxide present was very greatly in excess. However, when silica was added to the mixture of matte and oxides, or of roasted matte, black copper containing little lead could be produced and the larger part of the lead and iron slags off as a silicate. Even when an insufficient amount of oxide is present black copper will be produced together with an enriched matte containing but little lead. Low-grade mattes must be concentrated before the process is applicable, the author having great success with Octavi matte containing 40 per cent copper, 24 per cent lead and 11 per cent iron.

As the method does not require resmelting of anything but the lead silicate slag, which can be done in a blast furnace, it represents a saving of 60 per cent of the present cost of treatment as carried out at Altenau. An interesting conclusion drawn from his work is that with electric energy at a cost of $24 per kw-year the process of fusion can be more cheaply carried out in an electric furnace in Germany than in a reverberatory. The stages of the process recommended for the treatment of the Octavi matte are as follows: Pulverization to 2 mm, roasting to 4 per cent sulphur, fusion with silica. The black copper is refined, the leady slag going to the lead smelter with the silicate slag formed in the first fusion.

Chloridizing Roasting.—The chloridizing roasting of sulphide ores forms the basis of an extended study by Von Kothny in *Metallurgie* for July 8. He observed that when pyrite was heated in a stream of indifferent gas sulphur was first evolved at 200° C. and at 700° the pyrite had been transformed over into simple sulphide of iron. When heated in a current of air it started to transform itself to $FeSO_4$ at 250°, this being the only reaction taking place up to 200°. At higher temperatures it oxidized mainly to Fe_2O_3, though the reaction took place very slowly at temperatures below 500°. Any $FeSO_4$ formed was transformed by the oxygen to basic ferric sulphate which at 530° was decomposed into iron oxide and sulphur trioxide.

By the oxidizing roasting of copper sulphide mixed with iron

oxide and sodium choride practically all the sulphur goes into sulphate and about half the copper is transformed into chloride. Anhydrous cupric chloride mixed with sodium chloride and heated in a current of air to 250° gives off chlorine. The decomposition of copper sulphate by sodium chloride begins at 280°.

In a study of the reactions involved in chloridizing the author finds that the Hargreaves process by which hydrochloric acid is formed plays no part in converting cupric oxide into chloride. Neither does the direct action of chlorine on cuprous sulphide. Ferric chloride converts copper oxide into chloride very rapidly at temperatures around 500-600°. The formation of copper sulphate by roasting pyrite with copper oxide in the presence of sodium chloride plays no part. At temperatures of 300-600° ferric sulphate converts copper oxide slowly into sulphate. The author concludes that the mechanism of the chloridizing of a pyrite cinder containing a small amount of copper and sulphur is as follows: The copper is present largely as sulphide, which by an oxidizing roasting is converted into sulphate and oxide. Sodium chloride acts directly on the sulphate and ferric chloride on the oxide.

To insure chloridizing of such a material it must be finely ground; a large amount of air must be admitted in the oxidizing roasting period and stirring must be resorted to to insure contact with this oxygen; sufficient pyrite must be present to furnish the required amount of ferric chloride and is best added in a weathered form; for 4 per cent copper content at least 7.5 per cent salt must be added; the process should be carried out at temperatures between 500 and 600°.

Gold and Silver

Amalgamation Practise on the Rand.—In our issue for July, 1911, page 322, we gave a review of the proposals of W. R. Dowling regarding amalgamation of banket ore. These involved the removal of the plates from in front of the stamps and a large reduction of plate area used following tube milling. A further suggestion was that all amalgamation be carried on in a special plate house and that shaking plates be abandoned in favor of stationary plates. The reasons for Mr. Dowling's proposed changes were fully given in his paper.

In the June, 1911, issue of the *Journal* of the Chemical, Metallurgical and Mineral Society of South Africa, Mr. James E. Thomas contributes to the discussion of Mr. Dowling's paper and gives some data based on conditions before and after adopting Mr. Dowling's suggestions. These are conveniently tabulated in the following tables and present obvious conclusions.

Under the old system of plates in front of mortar boxes and shaking plates following tube mills the following results were obtained:

Average number of stamps running	235
Average aperture of mill screen	0.0461 in.
Average ratio of water to ore in mill	5.75 to 1
Average duty per stamp per 24 hours	9.459 tons
Average number of tube mills (22 ft. x 5 ft. 6 in.)	7
Average amalgamating plate area in use	5276 sq. ft.

The plates in front of mortar boxes accounted for 3351 sq. ft. of the total, and the shaking plates for the balance. The latter had a grade of 10 per cent and the pulp flowing over them a consistency of 2 to 1. The extraction by amalgamation was 56.8 per cent of the gold in the ore sent to the mill and by cyaniding 84.67 per cent of the gold in the cyanide pulp. The total extraction on the ore was 93.38 per cent.

During January and February of this year the plates were removed from the stamp mills and the shaking plates in the tube mill circuit were replaced by stationary plates. Comparative results follow:

Average number of stamps running	220
Average aperture of mill screen	0.0685 in.
Average ratio of water to ore in mill	5.17 to 1
Average duty per stamp per 24 hours	10.834 tons
Average number of tube mills (22 ft. x 5 ft. 6 in.)	10
Average amalgamating plate area in use	1700 sq. ft.

The plate area consists of three stationary plates per tube mill with a grade of 18 per cent; ratio of liquid to solid in pulp flowing over plates was about 1.1 to 1. The extraction by amalgamation was 55.52 per cent of the gold in the ore, by cyaniding 87.03 per cent of the gold in the cyanide pulp, and the total extraction was 94.23 per cent of the gold in the ore.

Cyaniding Pyritic Concentrate.—The concentration of black sand on amalgamated plates presents a special problem for the recovery of the gold which it contains, usually as particles of metallic gold attached to particles of pyrite. In the July, 1911, issue of the *Journal* of the Chemical, Metallurgical and Mineral Society of South Africa, Mr. R. Lindsay describes the method of treatment followed at the Geldenhuis Deep, Ltd. The concentrate treated results from the daily scrape from the mills and the barrel residue. The material is ground in barrels for four or five hours and then cleaned in the usual way over stationary copper plates. The flow from the plates is roughly divided in a settling tank into coarse and fine pulp, the former grading about 45 per cent — 200 and the later 98 per cent — 200. The former amounts to about 250 lb. daily, assaying about 30 oz. gold per ton. This is ground in a small tube mill to minus 200-mesh and passed into a batea containing a bath of 250 oz. mercury. The batea in this instance is used in place of an amalgamated plate and serves as a good amalgamator besides recovering some amalgam and mercury which escaped the first copper plate. From the batea the pulp flows to a tank where it is combined with the portion which was already as fine as minus 200-mesh.

The concentrate from the mill barrels is similarly first run over a plate and then divided into coarse and fine portions. The coarse is tube milled, combined with the fine, forming a pulp which is about minus 200-mesh in grade.

The combined fine pulps of pyritic concentrate from the plates and barrels is then ready for cyanide treatment. For this work an agitating and decanting tank designed by Andrew F. Crosse is used. This tank and a description of its use and operation were noted in this journal, February, 1910, page 72. The tank is conical, 7 ft. 6 in. in diameter and 8 ft. 4 in. deep, and has a content of 122 cu. ft. It has a capacity of 1800 lb. dry weight of material and there is no difficulty in obtaining a clear solution for decanting while air agitation is in progress.

In the present instance the practice is to agitate the charge with air for 12 hours before beginning to wash with weak cyanide solution. The gold goes into solution rapidly on account of the fine state of subdivision of the material, a charge assaying 250 dwt. gold per ton being reduced to less than 30 dwt. after one hour's agitation. At the end of 12 hours the undissolved gold in the concentrate is down to 6 dwt. per ton and the cyanide solution has a gold value of 56 dwt. per ton. Washing then proceeds at the rate of ½ ton solution per hour, agitation going on meanwhile. The quantity of solution required for washing the charge of 1800 lb. is six tons, and at the end of this operation the concentrate residue assays 4 dwt. per ton and the solution 0.08 dwt. per ton. This is about the economic limit to which solution and washing can be carried. The cyanide consumption is 2.75 lb. of 120 per cent KCN. Agitation requires 7 cu. ft. free air per minute.

Treatment of Old Slime by Air Agitation.—It is well known that accumulated slime which has become acid by reason of the oxidation of sulphur to sulphuric acid is not readily treated by the cyanide process on account of the heavy consumption of cyanide and the relatively large quantity of gold left in the residues. With a view to determining the best conditions under which such material could be treated, Mr. J. E. R. Adendorff made a number of experiments which are recorded in the July, 1911, issue of the *Journal* of the Chemical, Metallurgical and Mineral Society of South Africa. The material treated was accumulated slime containing 22.7 per cent

moisture, 0.024 per cent acidity as H_2SO_4, 2.31 dwt. gold per ton. The solution used was precipitated slime solution containing 0.017 per cent free KCN, 0.024 per cent total KCN and 0.02 per cent CaO.

Bottle tests were first made and these demonstrated certain facts, viz., the addition of lead acetate was beneficial in raising extraction; the free cyanide was well protected by the lime present; high residues pointed to a lack of oxygen.

Tests were then made on 100-lb. lots in a small Pachuca agitator. The reactions seemed to be most active within the first hour of treatment, during which the alkalinity and percentage of free cyanide dropped rapidly and had to be increased. About 50 per cent of the gold dissolved went into solution in the first hour and the economic limit of treatment seemed to be eight hours. In one test the original slime assayed 2.53 dwt. gold per ton, and at the end of eight hours' agitation the residue contained 20 dwt. undissolved gold per ton. This test seemed to show that success depended on maintaining a high alkalinity during the first three hours of treatment and that lime should be added to the slime before treating with KCN solution. The beneficial effect of lead acetate also was noted in the agitation experiments.

The author makes no mention of preliminary aeration of the slime in alkaline water before the addition of cyanide, which is a condition of success that other experimenters have laid down with reference to the treatment of acid slime for the purpose of oxidizing the reducers present.

Australian Ball Mill Practice.—Pulverizers other than stamps are meeting with success in various parts of the world, and engineers are showing a commendable tendency to investigate the merits of the newer machines. The mills of Kalgoorlie, Australia, have for some time been using ball mills in dry crushing and with considerable success, as is shown by Mr. M. W. von Bernewitz in a contribution to the *Mining Magazine* (London) for August, 1911.

It is doubtful if the ball mill is operated anywhere with greater efficiency and less cost than at Kalgoorlie. The ore ground averages 60 per cent silica and is considered hard to grind. It is dry enough for milling, containing from 0.5 per cent to 1 per cent moisture. Anything above this figure retards the crushing action. The best foundation for these mills has been found to be reinforced concrete blocks, as no other structure has given the required freedom from vibration. Two sizes of Krupp mills are used in Australia having the following dimensions:

	No. 5 Inches	No. 8 Inches
Diameter	80	106
Width	46	54
Outside screen area	46 x 26	54 x 24

Approximately the net screening surface in each mill is 70 sq. ft. and 100 sq. ft. respectively. From 24 r.p.m. to 26 r.p.m. is the correct speed for the No. 5, and from 21 r.p.m. to 24 r.p.m. for the No. 8 mill. They are loaded with from one to two tons of 5-in. steel balls, according to the size and hardness of the ore to be ground. There is no advantage in feeding several sizes of balls. The No. 5 mill takes from 18 hp to 23 hp when in average working condition and the No. 8 consumes 50 hp to 60 hp. Where motor driven the ammeters register 60 amp to 65 amp at 550 volts.

The balls are fed either through the manhole or hopper, and ore is added until the balls are well mixed with it. A mill should not be started without ore and the mass of ore and balls should be evenly distributed with no bare balls showing.

The efficiency of the ball mill over some other forms of grinders, notably stamps, lies in the fact that as soon as the ore is ground fine enough it must pass out of the mill. The ore crushed by the balls passes through the perforated grinding plates and is screened by the inside or coarse screen, the oversize being returned to the mill for further grinding. That part which passes the inside screen is further screened by the outside screen and the oversize returned to the mill.

Mr. von Bernewitz calls attention to the fact that the grinding of a ball mill generally has been supposed to be due to impact, but from observation he is convinced that this is not so and that the grinding is due to rolling and abrasion. The balls are lifted only to about the height of the mill axle and then roll back, rubbing and grinding the ore. This belief is confirmed by a microscopic examination of particles of various sizes taken from a mill. The larger ones, say plus 30-mesh, are shown to be rounded, while the finer grains, say plus 120-mesh, are angular.

The average consumption of balls is about 0.55 lb. per ton, and the grinding plates last about 190 days. A No. 5 ball mill will crush ordinary sulphide ore up to 43 tons daily and the No. 8 up to 100 tons. Dividing this by the horse-power consumed we get 2.1 tons and 1.6 tons per horse-power respectively. In a small plant of 100 tons daily capacity the No. 5 is a useful unit, but in larger plants the No. 8 is preferable. On quartz the capacity of the No. 5 mill is about 30 tons per day. The capacity of the mills is greatest with a feed of 3-in. material. The cost of ball milling is given by the author for several mills in Kalgoorlie. Depending on the ore treated, this varies about 35 cents to 65 cents per ton.

Wet Method for Making Gold Assays in the Field.—In a paper read before the Institution of Mining and Metallurgy, London, May 31, 1911, Mr. G. M. Austin gave his experience with a method of assaying for gold ores by a wet process suitable for use in the field where the usual facilities are not at hand. In brief the method is as follows: 1 A. T. of pulverized ore is treated with 50 c.c. of a solution containing 2 parts of iodine and 4 parts of potassium iodide in 100 parts of water. The mixture is ground in a mortar, and if necessary more iodine is added. The color of the mixture should remain brown. After 10 minutes' grinding and one hour standing the mixture is filtered and the solution agitated in a bottle with 3 g. of pure mercury. The color will disappear and the gold will be reduced and collected in the mercury.

The mercury is then separated from the solution and treated in a dish with 10 c.c. of nitric acid. The mercury will dissolve and leave the gold in a compact form ready to be annealed and weighed, or fused into a button and measured, or compared with standard beads mounted on a card.

Mr. Austin's experience with this method on a variety of ores gave the following results. Oxidized ores containing only a trace of gold showed this trace quite plainly. Low results were obtained in every case with basic ores containing manganese. Iron pyritic ores gave rather low results and copper pyritic ores containing as high as 34 dwt. gave no gold even after roasting. Telluride ores gave fair results, but the consumption of iodine was high. Ores containing blende and mispickle gave only about 75 per cent of their gold content.

Recent Metallurgical and Chemical Patents.

Iron and Steel.

Desulphurizing and Nodulizing Iron Ores.—For the utilization of pyritic cinder and other finely divided iron ores which are not fit for use in making iron, Mr. Tom C. King, of Marion, Ala., has invented a desulphurizing and nodulizing kiln which reduces the sulphur content to substantially nothing and discharges a sintered and agglomerated product suitable for blast furnaces. As shown in Fig. 1, the device consists of a rotating kiln suitably mounted and driven, having a hood 8 mounted on a carriage 9, with a fuel and supply pipe 11 projecting through the hood. The upper end of the kiln projects into a boiler setting, this end also being in connection with a feed chute 14. The heated gases issuing from the kiln are used to generate steam in the boiler 15. The inventor claims that he has found by practice that a temperature of 1200° Fahr. to

1500° Fahr. is sufficient to drive off the sulphur contained in the material and that a further temperature of 2000° Fahr. is required to sinter and nodulize the ore. The presence of moisture facilitates the operation, but no other binders or fluxes are used. (995,542, June 20, 1911.)

Gold and Silver.

Regeneration of Cyanide Solutions.—The recovery of the cyanogen present in solution in combination with the various metals it may have dissolved during the treatment of an ore is the object of a process patented by Mr. Raymond P. Wheelock, of Searchlight, Nev. Generally speaking the results of the process are first the precipitation and removal of such base metals as copper, lead and zinc which may have been dissolved during the treatment of the ore; second, the precipitation of any gold and silver which has escaped regular precipitation by the regular methods; third, the recovery of the cyanogen which was combined with these metals as soluble compounds.

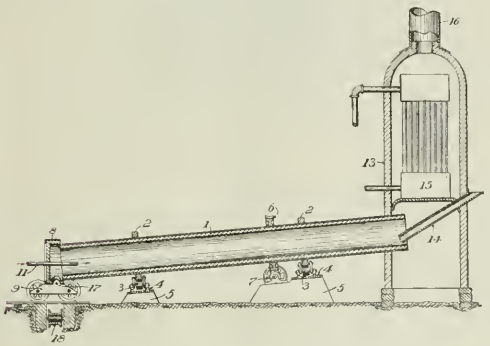


FIG. 1.—DESULPHURIZING AND NODULIZING KILN

The first step is to add sufficient sulphuric acid to the spent solution to precipitate as simple cyanides the metals held in solution as double cyanides of the metals and the alkali or alkaline earths present. This will liberate one-half of the cyanogen combined as double cyanides, which will be held in solution as hydrocyanic acid if care is taken not to agitate the solution. Taking copper as a typical case the following equation will show the reaction: $H_2SO_4 + K_2Cu_2(CN)_4 = K_2SO_4 + 2HCN + 2CuCN$. The precipitate is removed and the solution is rendered alkaline by the addition of a soluble alkali or alkaline earth, such as calcium hydrate, thus regenerating potassium cyanide as follows: $K_2SO_4 + 2HCN + Ca(OH)_2 = CaSO_4 + 2KCN + 2H_2O$. The precipitate resulting from this step is filtered off and the resulting clear solution used again as a solvent for gold and silver.

The precipitate resulting from the addition of sulphuric acid may be further treated with acid and heated to evolve hydrocyanic acid which can be combined with a solution of an alkali for further use. The metals remaining from this treatment can be recovered if their value warrants the operation. (996,179, June 27, 1911.)

Improved Cyanide Process.—Mr. John C. Clancy, of New York, whose inventions relating to cyanidation have been repeatedly noted in these columns, has patented another procedure based on his discovery that when pulverized ore is treated with a solution of a thiocyanate containing a halogen salt, and electrolyzed, the precious metals only will be dissolved to the exclusion of the baser metals, and that the process may be applied successfully to either rebellious or non-rebellious ores without preliminary roasting.

The following example is given: 2000 lb. of pulverized ore are suspended in a solution of 2 lb. of thiocyanate, 2 lb. of caustic soda or lime, 1 lb. of potassium iodide, the mixture being in the proportion of two parts of solution to one of ore. This mixture is continuously circulated for a period of eight hours through an electrolyzer having a current density of about 50 amp per square foot of anode surface. About from 90 amp to 100 amp are used per ton of ore. The function of the current is simply to produce nascent oxygen or liberate iodine from the salt. The same solution can be used repeatedly by keeping up the requisite strength of thiocyanate, but no further addition of iodide is required except to compensate for mechanical losses. The reason for using thiocyanate alone or in conjunction with simple cyanide solution in preference to the ordinary alkaline cyanide solution is that, if cyanide solution be used without containing thiocyanate or the addition thereof, the nascent oxygen produced by the current destroys the cyanide by converting it into a cyanate which is not a gold solvent. The following equations represent the reactions which occur: (1) $KCNS + H_2O + 3O = KCN + H_2SO_4$. (2) $KCNS + KI + 4O = ICN + K_2SO_4$. (908,749, July 25, 1911.)

Refining Gold and Silver by Distillation in Vacuum.—The present methods of refining gold and silver bullion consist in electrolyzing their solutions. Mr. William C. Arsem, of Schenectady, N. Y., proposes to effect a separation and refinement by distillation at proper temperatures in vacuum. He finds that silver can be completely distilled from an alloy by heating it to a temperature of about 1300° C. in a nearly complete vacuum. Subsequently the gold can be separated from any remaining refractory metals such as platinum by raising the temperature to about 1500° C.

The furnace in which Mr. Arsem carries out his distillation is shown in sectional elevation in Fig. 2. The main envelope

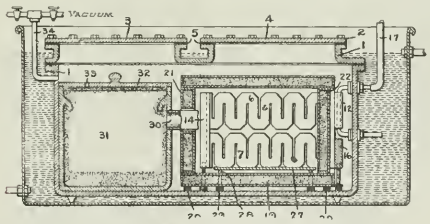


FIG. 2.—SILVER DISTILLATION FURNACE

of the furnace is a rectangular box 1 with a cover 2 bolted to it. The heating element consists of four graphite grids 6, 7, 8, 9, which attain a uniform incandescence when the current passes through them. Current is supplied by way of water-cooled copper tubes passing through the envelope. The heating element is properly insulated by double graphite walls with a

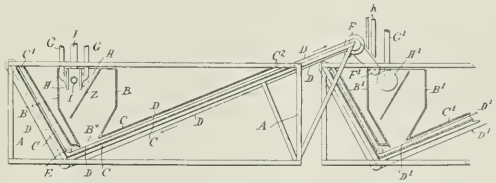


FIG. 3.—APPARATUS FOR SEPARATING CYANIDE SOLUTION FROM ORE PULP

packing of graphite. A crucible of suitable form containing the charge of bullion is placed in the heating space and connected with the condensing chamber 31 by a tubular outlet 30. The condensing chamber is of earthenware or iron of proper size to insure condensation, and properly water-cooled. The vacuum pump is connected with the pipe 34 and is kept in continuous operation throughout the distillation. The distillate can be re-

ceived in a container which can be removed from the chamber when the operation of distillation is completed. (998,665, July 25, 1911.)

Separating Solution from Ore Pulp.—A machine designed to afford a simple and inexpensive means of recovering the valuable cyanide solution from treated ore pulp has been patented by Horace G. Nichols, of Ymir, B. C. One of the difficulties of such separation which the machine is designed to overcome is that of causing the settlement of the finely divided pulp held in suspension in the fluid mass. This is accomplished by steadily and continuously withdrawing the solid matter as it settles from the deepest part of the machine by means of an endless moving belt traveling near the bottom of the tank. As shown in Fig. 3, the device consists of a supporting framework *A* in which is mounted a settling tank *B* converging to a bottom opening *Bx*. An endless traveling belt *D* moves along the bottom of the tank, finally emerging with a layer of settled solids and returning to the tank as shown in the drawing. The angle of inclination of the belt is from 20° to 25° with the horizontal. As pulp is fed into the receiving tank *B*, solids settle to the deepest part and are removed continuously by the belt. Valuable solution is withdrawn by means of suction applied through the pipes *G* to the surfaces *H*, the latter being covered with suitable filtering medium. The filtration may be stopped and the accumulated slime blown from the filters by means of a current of any fluid applied in the reverse direction to that of filtration.

The inventor claims that this procedure will remove the solids with a content of 22.5 per cent moisture, and if desired these solids may be similarly treated in a second tank with washing solution for the removal of the contained value. In practice the inventor finds that three of these tanks in succession are sufficient for complete washing. (996,877, July 4, 1911.)

Sand and Slime Classifier.—An improvement in a former machine of the inventor, John V. N. Dorr, of Denver, Col., has been devised for the purpose of more completely eliminating moisture or solution from the classified sand. The form of the improved machine is not greatly different from that

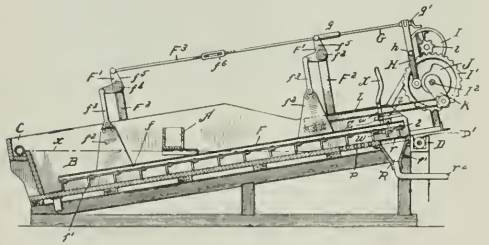
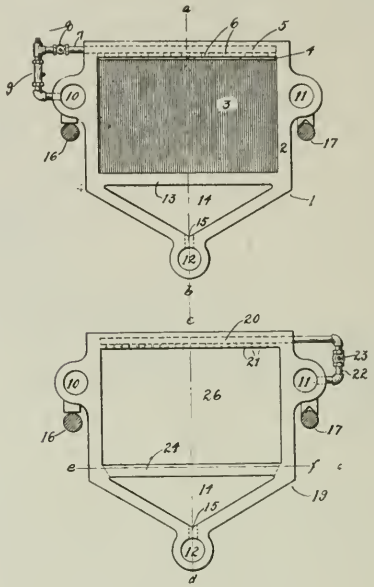


FIG. 4.—SAND AND SLIME CLASSIFIER

of the older, and is shown in Fig. 4. It consists of an inclined settling box B, which is fed with the ore pulp through a launder A. The liquid level in the box is indicated by the line *x*. Moving within the settling trough is a reciprocating rake which moves settled sand up the inclined bottom of the trough on its forward movement and rises above the sand on its return. The sand is thus continuously removed from the slime which overflows at the launder C. Ordinarily in the use of this classifier the sand is removed with about 26 per cent moisture, but it frequently is desirable to deliver drier sand, and it is the purpose of the present improvement to accomplish this end.

Near the point of discharge of the sand, above the level of the liquid in the settling box, will be seen a suction box R which is used for the removal of moisture. The suction box is connected with an exhaust pump by pipe r^2. The arrangement and shape of the rakes is such that an even layer of sand is steadily moved onto the surface of the suction box so that the moisture may be withdrawn uniformly. In this way the sand may be withdrawn with 9 to 10 per cent moisture. If it is desired to wash the sand at the time of discharge this may be accomplished by means of a spray pipe W, carried by the rake and arranged to deliver a spray of water on the sand during the upward stroke of the rake. (996,624, July 4, 1911.)

Filter Press.—In Metallurgical and Chemical Engineering for January, 1910, p. 52, there appeared a description of



FIGS. 5 AND 6.—FILTER PRESS

the filter pressing of slime by means of the Sweetland press. The inventor of the press, Ernest J. Sweetland, of Los Angeles, Cal., has recently patented a form of press differing in some features from that described last year. The principle on which the press works can be understood by reference to Figs. 5 and 6, which show respectively a filter frame or drainage unit and a distance frame unit of the press.

The cycle of operations in the press, viz., filling and forming the cake, washing and discharging the same, can be performed without opening or breaking the press. Each of the frames making up the press has three openings at 10, 11 and 12, which register exactly when the frames are locked in the press and form a continuous channel connecting with pipes at the ends of the press. The drainage frame has a rectangular corrugated part 3, connecting with an upper groove 4 and channel 5. The latter is further connected by suitable pipes and fittings 7, 8 and 9, with the opening 10. The part 9 is a glass tube through which the filtrate can be seen as it comes from the unit. The distance frame corresponds in outline with the drainage frame, but is open in the rectangular portion. The opening 11 is connected by suitable pipes and fittings 22, 23, with a channel 20, which has passageways 21 communicating with the rectangular space 26. In both drainage and distance frames the triangular space 14 is open and connected by a passage 15 with the opening 12.

The frames described are placed alternately in a press, and a filter cloth is placed over each side of the drainage unit. The press is filled through the channel 12 at the bottom of the press. The filtered liquid passes through the cloths on the drainage unit, upward through the corrugations 3 to the groove 4 and thence by 5, 6, 7, 8 and 9 to the conduit 10, which has a pipe connection carrying the fluid to the desired place. When

the cake has formed to a sufficient thickness the excess pulp is withdrawn through the conduit 12 and wash water or solution forced in through the same opening. When the pressure is withdrawn from the cakes they are held onto the filter by vacuum applied through conduit 10. In discharging the cakes from the filters, compressed air or water is forced through conduit 10, whereupon the cakes drop into the triangular space 14 and are flushed from the press by means of water jets at one end. At the same time the filter cloths are washed clean by means of jets of water forced through the conduit 11 and openings 21.

A further claim of the inventor is that the triangular space 14 serves as a storage of filtered pulp. In some cases only thin cakes can be formed on the cloths, and then it is more convenient to drop those cakes off into the space 14 and continue filtering without flushing the cakes out of the press. This he claims is possible because the plastic mass of discharged cake does not disintegrate under the action of the inflowing pulp. (996,537, June 27, 1911.)

Copper.

Smelting and Converting.—Mr. George Mitchell, of Los Angeles, Cal., proposes to construct a furnace in which reverberatory smelting and converting operations shall be carried on in the same furnace structure, and the heat evolved in the converting process utilized on the smelting hearths. His patent papers show various combinations of hearths and converters, the latter being arranged intermediate the former and adapted to receive matte from hearths by gravity. (997,405, July 11.)

Arsenic.

Recovery of Arsenic from Ores.—An improvement in handling arsenical vapors is claimed in a patent recently granted to Ira F. Peck, of Providence, R. I. The arsenic-bearing material is roasted in a revolving drum and the vapors issuing therefrom are conducted through a series of tanks which contain a thick creamy mixture of lime and water. A vacuum

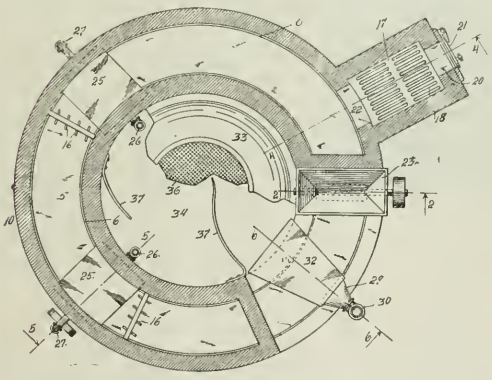


FIG. 7.—MAGNETIC ROASTER

applied at the last tank assists in causing the gas to flow through the lime mixture where the arsenic is absorbed. When the mixture in any tank has taken up as much arsenic as possible it is withdrawn and replaced with a new mixture. The composition of the lime mixture is about 25 lb. lime per 100 gallons of water. The recovered arsenic can be used in the manufacture of insecticides. (996,516, June 27, 1911.)

Magnetic Roasting.

Wilfley Roasting Furnace.—In the issue of this journal for April, 1910, page 171, mention was made of a new furnace designed especially for magnetic roasting of ores. A patent on this furnace has recently been issued to Arthur R. Wilfley, of Denver, Col. A plan view of the furnace is shown in Fig. 7. It consists of a roasting chamber 5 within which is an endless traveling annular hearth. Ore is fed onto the hearth from a hopper 23 having a distributing roll beneath its discharge opening. Heat is supplied to the roasting chamber from a furnace 17 by way of an opening 22. Stationary rabbles 16 are disposed above the hearth at intervals as shown, although if the ore bed is shallow they may not be needed. Receptacles 25 are placed above the hearth as shown, adapted to receive cooling water and thereby regulate the temperature of the hearth at these points. The roasted ore is removed from the hearth by means of a jet of steam delivered through a nozzle 29, this procedure being designed to cool the ore at the time of its removal and thereby prevent further oxidation of the sulphur. The ore thus blown from the hearth passes into the chamber 34, where it is distributed uniformly by the deflectors 37. It is then removed from the chamber in any suitable manner. (996,548, June 27, 1911.)

Aluminum.

The electric reduction of aluminum from its sulphide or a double sulphide of aluminium and another metal is the basis of a patent recently granted to Lucius R. Keogh, of Ottawa, Ontario, Canada. The process consists essentially of causing a reaction between a mass composed of sodium chloride or similar substance and anhydrous aluminium sulphate and carbonaceous matter. Intermediate reactions between these will form aluminium sulphide, thus:

$$6\,NaCl + Al_2(SO_4)_3 = 3\,Na_2SO_4 + 2\,AlCl_3.$$
$$3\,Na_2SO_4 + 6C = 3\,Na_2S + 6\,CO_2.$$
$$3\,Na_2S + 2\,AlCl_3 = 6\,NaCl + Al_2S_3.$$

The mass is fused by the application of heat, as by electricity, and the reduction of aluminium will follow. The process can be varied by withholding the carbonaceous material, in which case only aluminium chloride will be formed, which may be distilled away and collected. For the formation of aluminium sulphide and reduction to aluminium the anhydrous sulphate of aluminium is mixed with carbonaceous matter in equivalent proportions, finely ground, mixed with water to form a paste, dried and again finely pulverized. It is then fed into an electric furnace containing a fused bath of the sodium chloride or other selected reagent. The specific gravity of the bath must not be greater than 2.54 in order that the reduced aluminium can sink below it and be tapped from the furnace. The presence of fluorides in the bath contributes to fluidity and salts of lithium decrease the specific gravity of the bath. By regulating the current voltage so as to overcome the resistance of the circuit and just dissociate the aluminium sulphide, no other metal need be deposited, and the aluminium will be obtained free from any appreciable quantity of other metals. (996,094, June 27, 1911.)

Manufacture of Aluminum Nitride.—Ottokar Serpek of Paris, France, has patented a process for producing aluminium nitride from bauxite which he has assigned to Société Generale des Nitrures, of Paris. The general process is as follows: Ground bauxite is calcined, mixed with carbon and heated in an electrical resistance furnace through which is passed a stream of nitrogen generated for the purpose. The apparatus used is designed to afford a methodical heating of the substances with as complete recovery of the heat as possible. Reference to the complete specifications is necessary for a detailed explanation. (996,032, June 20.)

An improved aluminium alloy has been patented in the United States by Conrad H. H. Claessen, of Berlin, Germany. He makes use of magnesium and copper, the former in quantities not exceeding 2 per cent and the latter in quantities not exceeding 5 per cent. He has obtained particularly good results with an alloy consisting substantially of 99.5 to 99.75 per cent Al, 0.25 to 0.5 per cent Mg and 4 per cent Cu. The alloy is claimed to be light, and at the same time stronger and of better quality than either aluminum or the present alloys of that metal. (995,113, June 13, 1911.)

Toronto Meeting of the American Electrochemical Society

The twentieth general meeting of the American Electrochemical Society was held from September 21 to 23 (Thursday to Saturday), at Toronto, Ontario, Canada.

The number of members and guests registered was 125. The attendance was thus somewhat smaller than at most of the recent meetings of the Society, but this was hardly a drawback, as it gave a distinct homelike character to the affair. It was a very congenial gathering and the hospitality of Toronto was unbounded.

There were three professional sessions for the reading and discussion of papers, two on the morning and afternoon of Thursday in the Chemistry and Mining Building of the University of Toronto, and a third at the clubhouse of the Lambdon Golf and Country Club on Friday morning. In the whole nineteen papers were presented and discussed, and one more was released and distributed on the train on Saturday morning; printed advance copies were on hand of sixteen papers. In some cases a lively and interesting discussion ensued. A full account of the professional sessions with abstracts of the papers and discussions will be found below.

Smoker.

On the evening of Wednesday a very enjoyable smoker was given to the Society in the West Hall of the Main Building of the University. This will long be remembered for its main features as well as for the quaint arrangement of the whole, with the piper, the torchlight parade, and the song of Auld Lang Syne.

The main entertainment features were a remarkable exhibition of glass-blowing by Prof. E. B. Kenrick and an equally remarkable play by Section Q. Prof. Kenrick showed in the most skillful and entertaining manner the development of three embryos (glass bulbs) into a rooster, a cat, and a human being.

W. R. WHITNEY, PRESIDENT AMERICAN ELECTROCHEMICAL SOCIETY

The play by Section Q was entitled "The Electro-Comical Lab," and the playwright actors were Messrs. Lidbury, FitzGerald, Bennie, Saunders, Carveth, Hooker and Dushman. It was an electric-furnace farce, full of bright lines and clever fun.

The rise of the curtain showed an electric furnace room with three assistants getting up things. One has just graduated from a correspondence electric furnace engineering course of three weeks. Instruments are gathered together—an electrode indicator ("going, going, gone") an effishy-ometer (reading up to 250 per cent efficiency), also samples of electrochemical products. "Priceless" ferrotitanium, for instance (the only ferroalloy Mr. Price does not make). Everything is ready, when the doctor enters with a party of visitors to demonstrate the furnace.

To "graphitize our amorphous ideas" a little text of Browning for Section Q is quoted (back to his studies, fresher than at first, fierce as a dragon, he, soul-hydropic with a sacred thirst, sucked at the flagon).

The demonstration of the furnace begins. Not starting at first it is lubricated with jagdag. The furnace works and the visitors and the doctor talk. A phonograph also talks—an auto-starter, guaranteed to start discussion on any subject at any time. It is a pity that all the talk cannot be reproduced.

Dr. Zinc O'Chink, the efficiency engineer, is called in, explains water-cooling, sprays the electrode, the electrode breaks and the efficiency rises to 110 per cent. The furnace works now as an induction furnace. Just when the doctor has explained that the furnace is always working very smoothly, there is an explosion. An officer appears and arrests the furnace man—the "pinch effect," also demonstrating "the conductivity of the copper." Mr. Lidbury sings a clever song on making the punishment fit the crime.

The furnace has made graphite. Ferro-alloys are not attempted, since they are all patented several times each, though they are mentioned in the Old Testament. (Didn't Pharaoh's daughter discover Pharao Moses in the bull rushes?) So a steel mixture is tried. Fractional distillation of impurities begins, driving off nitrogen and oxides. When the furnace is to be tapped, there is trouble and resort is had to the latest remedy in treatment cases of electrothermic constipation, the squirt effect.

Three ingots of steel are tapped Owing to the high efficiency of the furnace, the metal is tapped at room temperature, while by watching the stack the visitors see that the furnace is now giving off solid CO_2. The ingots are taken to the rolling mill. The rail is tested, succumbs to the drop test, and is shipped. The demonstration is over. A song on Dr. Heroult is the finale.

It is impossible to do justice to the play in a brief review. But those who had the privilege of seeing it acted, enjoyed it immensely.

Before the smoker an exhibition was given in the basement of the Chemistry Building of the Evans-Stansfield electric furnace for producing titanium steel direct from ore. A feature of the furnace design is the arrangement of the electrodes, several sets being provided for starting and running.

Excursions and Banquet.

Friday was devoted to a trip to the Lambton Golf and Country Club. The professional session of the morning was held at the club house, while after lunch those who wanted to play golf did so on the beautiful grounds. Ideal weather contributed to the enjoyment.

In the evening of Friday a subscription dinner was held at McConkey's restaurant. Prof. Ellis acted as toastmaster with great success. Three toasts were drunk, to the President, to the King, to the Society.

Speeches were made by the president of the Society, Dr. Whitney, by the secretary, Dr. Richards, by the chief engineer of the Hydro-Electric Power Commission of Ontario, Mr. Sothman, by Dr. Baekeland, and by Mr. Cohoe. The two last named speakers called attention to the eminent importance of next year's Congress of Applied Chemistry—the first to be held in this country.

The third day of the meeting was given over to a highly interesting visit to Hamilton, Ontario, and its vicinity. Over forty members and guests participated. The first plant inspected was the large, and rapidly growing, plant of the Canadian Westinghouse Company. The one hour available in this

plant passed all too quickly, among the multitude of interesting and instructive apparatus and operations.

The party then proceeded to the plant of the Steel Company of Canada. Here were two blast furnaces, and numerous basic open-hearth steel furnaces, mixer, etc. The attraction commanding chief attention, however, was the Gayley refrigerating plant for drying the air for the two blast furnaces. These furnaces each require some 20,000 cubic feet of air per minute, which is cooled to 14 deg. F (—10 deg. C.). The moisture content of 6 to 7 grains per cubic foot is thus reduced to 0.5 grain. The output of the two furnaces has been increased from 200 and 300 tons per day to 225 to 350 tons, respectively, while the coke consumption has been considerably reduced, being now 2200 lb. per 2240 lb. of pig iron. The *net* saving over all in the production of the iron is given as 75 cents per long ton of pig iron. The refrigerating plant contains 48 miles of pipes, and cost $500,000. The party was divided into groups, and conducted through the refrigerators into the coldest chamber—a unique and long-to-be-remembered experience.

The basic open-hearth steel plant for the inspection of which very little time only was available, contains four furnaces, two of 50 tons each and two of 25 tons. It is fired by natural gas.

The transformer station of the Canadian Hydro-Electric Commission was visited in the afternoon after lunch had been taken at the Waldorf hotel in Hamilton. A spick-and-span new plant is this transformer station, receiving power at 110,000 volts from Niagara and reducing the voltage to 13,200 volts for the local Hamilton service. It is also a switching station for all the territory supplied to the north, up to Toronto. It is equipped to handle 70,000 kilowatts, but only 12,000 kw go through it at present. The line loss from Niagara is given as under 5 per cent. Many obscure phenomena have occurred in the handling of this 110,000-volt current; one being a jump of the current *several feet* to another crossing line, during a thunderstorm, when either a lightning flash started the current across, or else a condition of unusual atmospheric electric tension superimposed on the 110,000 volts did the mischief.

The party returned to Hamilton, where it broke up and the members went off by train in different directions.

Professional Sessions.

The first professional session was called to order by the president, Dr. W. R. WHITNEY, on Thursday morning. Since the secretary, Dr. J. W. Richards, had not yet returned from Europe, Mr. Carl Hering was elected active secretary for the first sessions. Dr. Richards arrived on Friday morning.

The first paper of the meeting was presented by Mr. Robertson.

Electric Iron Smelting in Sweden.

A paper by Mr. THOMAS DUNCAN ROBERTSON, of the Canadian Böving, Co., Ltd., of Toronto, Ont., gave an account of "recent progress in electrical iron smelting in Sweden."

The author referred to Dr. E. Haanel's well-known reports on the work of the three Swedish engineers, Assar Grönwall, Axel Lindblad, and Otto Stalhane in the electro-metallurgy of iron and steel, their first 700-hp electric iron ore reduction furnace at Domnarfvet, and their later 2500-hp furnace at Trollhättan, erected by the Jernkontoret (the association of Swedish Iron Masters).

All these developments have been fully covered in this journal (concerning the design of the Trollhättan furnace and the results obtained at Trollhättan see especially the three articles in our issues of July and September and elsewhere in the present issue).

It may therefore be merely stated here that charcoal is used as fuel and that an important feature of the furnace is the provision made for the circulation of the gases produced by the reduction of the iron ore. The gas is drawn from the cool upper part of the furnace by means of a fan, and blown under the hot roof of the crucible, serving the double purpose of prolonging the life of the roof, and at the same time giving up its heat to, and helping in the reduction of, the charge.

During a large portion of the time the charge has been high in iron and the quantity of slag correspondingly small, so that the iron and slag have been tapped out together, and by means of a small dam the slag has been allowed to run off down a sand trench.

Table I given in Mr. Robertson's paper, is interesting, as it gives a very concise and yet full summary of the working results at Trollhättan.

The thermal efficiency of the furnace was carefully worked out, when it had been running for five months, after which time the lining had worn rather thin, so that radiation losses were probably higher than normal. However, 80 percent (the figure obtained) compares favorably with 82 percent for the efficiency of the Swedish charcoal blast furnace, and with the 70 percent of the English blast furnace.

As the volume of the furnace and gas piping is constant an excess of gas is being continually produced, proportional to the amount of ore reduced. This gas is allowed to burn into the air at present at Trollhättan, but in the newer plants arrangements are to be made for using it for calcining purposes. About 550 cubic meters of gas are produced per ton of pig iron, and each cubic meter will develop 2,100 to 2,500 Calories, as the gas averages about 80 per cent of combustibles.

When calculating the various charges the quantity of charcoal is kept constant at 6½ hectolitres and the amounts of ore and limestone are varied to produce the desired grade of iron, a typical charge at the time when the author was present at

TABLE I.

		Period of Operation				
	1910 *Nov. 15.	1910 Nov. 16 to Feb. 11; 1911	1911 Feb. 11 to Feb. 19.	1911 Feb. 19 to March 19.	1911 March 19 to April 9.	Total.
Per cent. of iron in ore	64.92	65.57	65.06	49.50	57.92	61 54
Per cent. of iron in charge	59.80	62.10	62.56	42.42	53.06	57.00
Slag per ton of iron (kg.)	390.	205.	224.	780.	458.	327.
Material charged per hectolitre of charcoal (kg.)	.....	66.49	71.13	90.31	69.88	70.77
Charcoal per ton of iron (hectolitres)	.....	24.22	22.47	26.10	26.97	24.79
Charcoal contents:						
Water (kg.)	.....	69.1	50.8	59.8	40.2	60.9
Gas (kg.)	.....	41.7	36.9	49.3	43.1	42.9
Ash (kg.)	.....	11 8	11.0	13.2	17.2	12.8
Coke (kg.)	.....	293.1	277.6	323.4	325.7	301.4
Total kilogrammes	.....	415.7	376.3	445.7	426.2	418.0
Electric Power.	h. m.	h. m.	h. m.	h. m.	h. m.	h. m.
Time consumed working	7.50	2,009.56	184.32	639.18	506.34	3,348.10
Time consumed in interruptions	.....	105.39	4.58	20.57	22.11	153.45
Total time	7.50	2,115.35	189.30	660.15	528.45	3,501.55
Average load (kw.)	1,121	1,319	1,694	1,017	1,733	1,344
Total kw. hours used	8,780	2,651,029	312,601	650,480	877,706	4,500,596
Kw. hours per ton of iron	3,800	2,296	2,149	2,623	2,643	2,391
Iron per kw. year (tons)	2.31	3.82	4.08	3.34	3.31	3.66
Electrode consumption:						
Gross (kg.)	.....	13,012	1,578	2,281*	2,474	19.345
†Net (kg.)	.....	6,743	763	1,121	1,285	9.912
Per ton of iron:						
Gross (kg.)	.....	11.24	10.84	9.19	7.45	10.28
Net (kg.)	.....	5.83	5.24	4.52	3.87	5.27

*Filling up of furnace. †Net = 51.24 per cent of gross.

the furnace being 266 kilograms of magnetite, 133 kilograms of concentrates, and 11 kilograms mill cinder, the total of these three constituents being 410 kilograms.

To this were added 32 kilograms of limestone and 6½ hectoliters or 100 kilograms of charcoal.

This is dropped into the furnace at one raising of the bell and tappings are made every 18 charges, or about once every six hours.

The conditions governing the grade of iron produced are similar to those in the ordinary blast furnace except that the irregular influence of the air blast is absent. The furnace gives the maximum output when making white iron, as the making of grey iron requires a rather higher temperature and consequently a greater power consumption. By increasing the

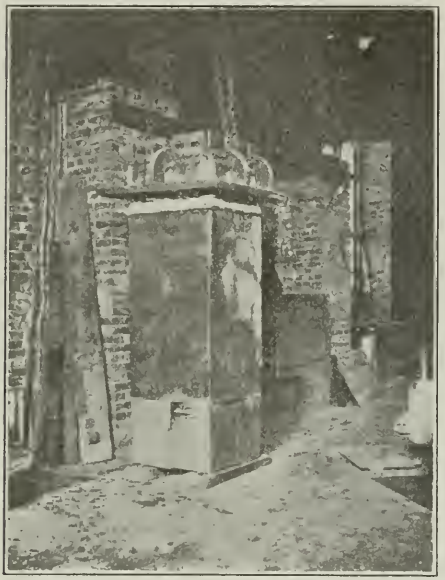


FIG. 1.—NEW ELECTRODE, EQUIPPED WITH IRON SHEATH AND CONTACT FRAME

amount of ore in the charge when the furnace is making white iron, a low-carbon iron with very little silicon is produced, a typical analysis of which is as follows:

	C.	Si.	Mn.	S.	P.
Percentages	2.60	0.10	0.11	0.02	0.01

This iron naturally is full of holes, but instead of these having colored oxidized surfaces, they are silvery white; the absence of oxygen from the furnace atmosphere accounting for the production of this grade of iron free from oxides.

The results of making steel from this iron are given in the second part of the paper.

The tension employed varies greatly and depends mainly on the resistance of the charge. This broadly speaking increases with the amount of slag produced and with the wear of the electrodes, but decreases as the amount of charcoal in the charge becomes greater. The maximum power allowed is 1,900 kw. and to avoid overheating the electrodes the current must not exceed 17,000 amperes, so that the voltage has to be regulated accordingly.

The wearing of the electrodes has given rise to some curious and interesting results. In Fig. 1 is shown a new electrode with its iron sheath and contact frame. When this is placed in the furnace, the bottom end projects into the actual charge and is gradually abraded and consumed. Between the stock line and the roof is a space filled with furnace gases which contain about 15 per cent CO_2, so that the portion of the electrode exposed to them is very gradually oxidized.

The result of this is shown in Fig. 2 where the half-con-

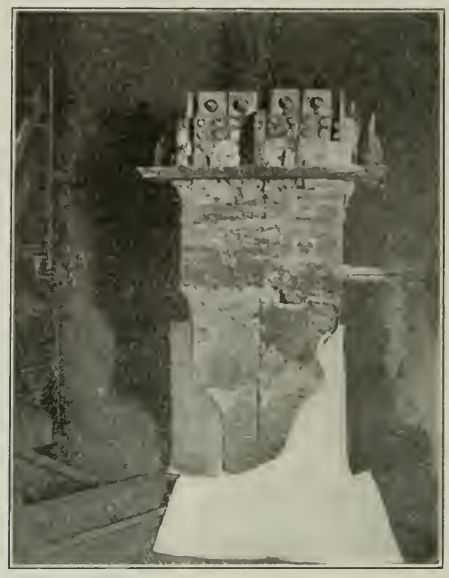


FIG. 2.—HALF-CONSUMED ELECTRODE

sumed electrode was removed for inspection, the line of the

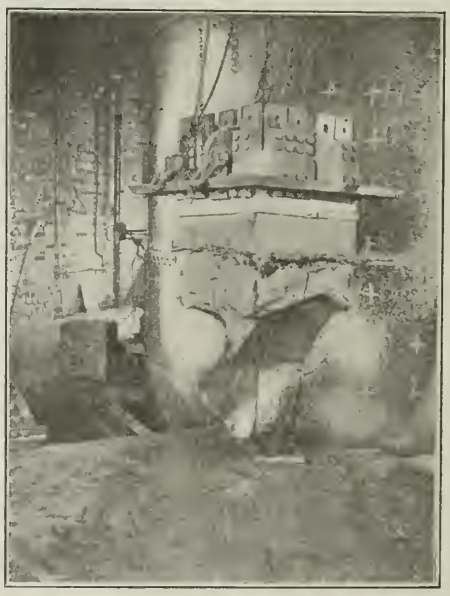


FIG. 3.—STUMP END OF ELECTRODE

surface of the charge being clearly visible in the picture. Fig. 3 shows the stump end of an electrode which had been

lowered as far as possible into the furnace and kept there until unfit for further use. It will be seen that two of the four carbons which compose it have been burnt through to the center.

About three weeks is taken to burn through an electrode above the stock line, so that in practice each electrode is

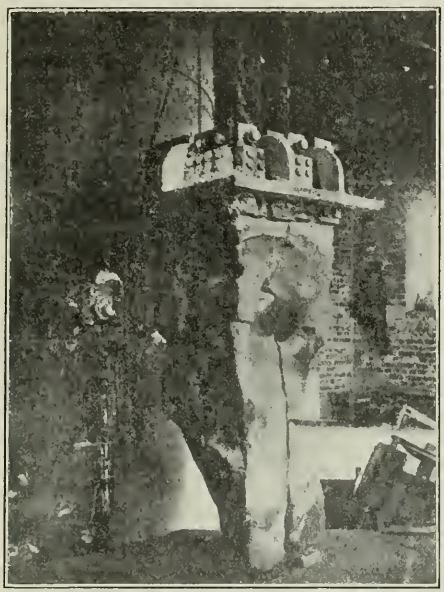


FIG. 4.—EFFECT OF CARBON DIOXIDE ON ELECTRODES

lowered once every two weeks, to ensure regular working without risk of fracture, which would result in a lump of electrode being left in the charge.

With the object of cooling the roof immediately behind the electrodes, gas was blown in beneath the water jackets during a portion of the time. The effect is shown in Fig. 4, the CO_2 in the stream of gas having burnt a hole in the electrode, causing this practice to be discontinued.

A point of interest from a Canadian point of view is the success met with in smelting magnetic concentrates at Trollhättan. The design of shaft in this particular case is not considered suitable for the purpose, being too narrow, but in spite of this, 65 per cent of finely divided concentrates caused no inconvenience in working. The inventors of the furnace are of the opinion that, with a specially designed shaft, charges of all fine concentrates could be smelted.

Canada is rich in deposits of iron sands and lean magnetites, which are easily concentrated, but which are expensive to nodulize or briquette into a form suitable for blast furnace smelting, so that where these are within easy reach of water powers, there seems to be a good field open for electric smelting.

Titaniferous iron ores were found to present no difficulty, although no ores were used with more than 0.8 TiO_2. Nor does high sulphur content prove objectionable, as there can be little doubt that the electric furnace with its reducing atmosphere and basic lining of the hearth, permitting as it does of the use of a slag very rich in lime, offers the best method of producing sulphur-free pig iron from ores containing that unwelcome element.

The pig iron produced by the furnace was sent to various Swedish iron works for conversion into steel in open-hearth furnaces. The characteristic feature of electric pig iron is its freedom from oxides; and in consequence electric pig iron of normal silicon content (say, 1 per cent and over) takes a longer time and more ore to convert into steel than ordinary blast-furnace grey iron.

Low-carbon electric pig iron, however, is found to give surprising results, charges made up of 50 per cent of this iron and 50 per cent of scrap producing hot fluid steel with considerable saving of time over ordinary practice.

As was to be expected, the open-hearth furnace managers looked somewhat askance at this iron at first, as they knew the disastrous effect of using low-carbon iron full of holes from the blast furnace; however, after giving it a trial, the workmen asked for more, as they said that the furnace worked better and more rapidly with the new pig iron.

Fortunately for the electric furnace in Sweden, it is more economical to make the white iron, that the steelmakers prefer, than to make high-silicon grey iron. Thus it may be maintained, on the strength of the experience gained, that for the open-hearth process, high-silicon contents are detrimental rather than advantageous, while in the blast furnace pig iron a certain quantity is necessary to neutralize the defects of a reduction process less perfect and ideal than that employed in the electric furnace.

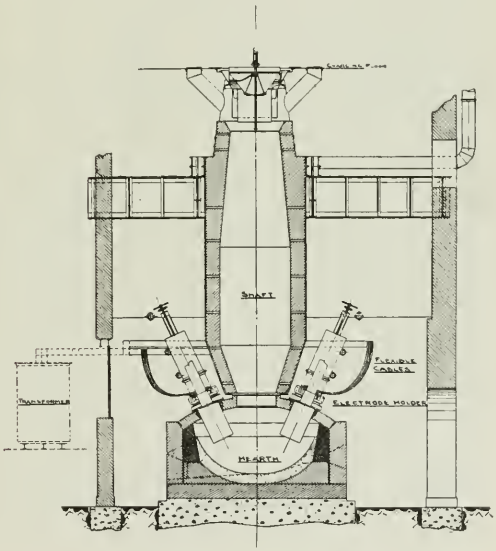


FIG. 5.—DESIGN OF NEW 3000-HP FURNACE

The quality of the steel produced has been thoroughly tested by making it up into various products and comparing it with steel made from Swedish charcoal blast furnace iron. In no case was the new steel inferior, and, from the most recent reports to hand, it was in several cases decidedly superior.

The Trollhättan furnace was shut down at the end of May of this year in order to make certain alterations suggested by the results of its six months' campaign. Two of these deserve mention.

1. Since the furnace was designed, the manufacture of large carbon electrodes, of high conductivity, has made great progress. It is now possible to obtain cylindrical electrodes of 600 mm. diameter, fitted with screw joints. These have recently been installed at Trollhättan, and result in two considerable improvements, the first one being that the loss due to stump ends is done away with, as when an electrode becomes too short, a new one can be screwed on above it. The second

one is that the electrical contact can be made at the point where the electrode enters the roof, in this way saving about 40 kw., which with the old arrangement was lost, due to the resistance of the whole length of the electrode through which the current had to pass. This new arrangement will permit of the furnace taking 3,000 hp. instead of 2,500 hp., and is shown in Fig. 5.

2. The other important alteration is in the gas-circulating arrangement, the new form of which is seen in Fig. 6. The old dust-catcher was not very effective, as only very fine dust is expelled from the furnace. The result was that the dust

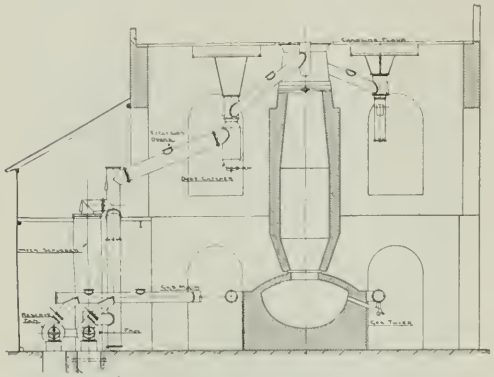


FIG. 6.—NEW GAS-CIRCULATION ARRANGEMENT

collected in the centrifugal fan, and had it not been that this was kept fairly clear by a jet of water, would have choked up the blades in a few hours; as it was the fan had to be stopped and cleaned every week. Now a water-scrubber takes the place of the dust-catcher, and a fan on the ground floor blows the cleaned gas into the furnace, the other fan shown being held in reserve.

Rather more than three metric tons of pig iron are produced by the electric furnace per electrical hp. year, using one ton of charcoal for the reduction of the ore. The blast furnace uses about three times this amount of fuel per ton of iron, so that in Sweden, where, owing to the demand of wood for the pulp industry, the price of charcoal is yearly advancing, there is a great field open for electrical smelting on account of the saving in fuel.

The furnaces in Norway have a somewhat different form of shaft, as coke is the fuel employed for smelting. The volume of the shaft is smaller, but its diameter is greater, than the corresponding shaft of a charcoal furnace. The coke in the charge gives a greater conductivity, so that a lower tension has to be used.

Mr. Robertson summed up as follows:

1. Electric smelting of iron ores is no longer in the experimental stage, but in Sweden is thoroughly established on a commercial scale.

2. The smooth and regular working of the furnace at Trollhättan has been a marked feature, probably less trouble being experienced than would have been the case with a blast furnace doing similar work.

3. The quality of iron produced for steel-making purposes has been shown to be equal and in some cases superior to that of Swedish blast furnace pig iron.

4. The comparative costs of electric and blast furnace smelting depend in general on the cost of suitable fuel and electrical energy, as the electric furnace simply substitutes one electrical hp. year for two metric tons of blast furnace fuel. In Sweden there is do doubt that the electric furnace has come to stay on account of cheaper production.

5. The electric furnace is specially adapted to the smelting of finely divided ores or concentrates.

6. Electric smelting in Sweden has a national value as its adoption will result in the development of water powers which could not in many cases be profitably utilized for any other purpose.

The paper elicited an extended discussion in which Dr. Brady, Dr. Whitney, Mr. Louis Simpson, Mr. Kohn, Mr. Hering, and Mr. Lidbury participated.

Mr. Simpson referred to the troubles due to the dust in the gas and asked whether there was any necessity for using the gas for cooling the roof. This necessity might be questioned since this method is not employed at the Noble electric iron furnace plant in Heroult, California. Another difference from the California practice is that while the electrodes reach deeper into the charge in California, they are free burning in Sweden. Mr. Simpson also questioned whether any reduction of ore was accomplished by the gas in the shaft in view of its relatively low temperature. Finally he stated, that English buyers of electric pig iron had found it irregular; while electric pig iron had always been claimed to be of higher quality than ordinary pig iron, it would be most unfortunate if it should turn out to be irregular.

Mr. Robertson replied that the trouble with the dust in the gases existed, even with the new washing arrangement, yet they were continually making progress and improvements and were sure that the problem would soon be solved. As to the use of the furnace gas for cooling the roof, this is considered by the inventors of the Trollhättan furnace to be one of the essential and most important points of their design. While the temperature of the gases in the stack is relatively low, it is in general still sufficiently high for reduction of magnetite. Finally as to the alleged irregularity of the electric pig iron sold in England, Mr. Robertson stated that most of the pig iron made in Trollhättan in the electric furnace was used by the Swedish iron masters themselves, who had supplied the iron ore. None of the Trollhättan electric pig iron was sold in England. Probably the pig iron sold in England to which reference had been made by Mr. Simpson had been made in small electric furnaces, hence the possibility of irregularity. With the new large electric furnace no such difficulty as to irregularity is experienced.

In reply to another question Mr. Robertson stated that at Trollhättan all kinds of electric pig iron are made, low or high in carbon. There is no more difficulty in making high-carbon pig than low-carbon pig in the electric furnace.

In reply to a question of Mr. Hering, how the 80 per cent efficiency of the furnace had been calculated, Mr. Robertson said that he had not the exact method of calculation at hand, but he knew it had been carefully calculated and the three comparative efficiencies of the Swedish electric furnace, the Swedish charcoal blast furnace, and the English blast furnace had been calculated by the same method. Mr. Lidbury remarked that Table I (given above) was so complete as to exact statements of facts that it enabled anyone to calculate any efficiency desired to suit his idiosyncrasies.

Heat Losses of Electric Furnaces.

A paper by Mr. Francis A. J. FitzGerald, of Niagara Falls, N. Y., is entitled a "note of an unsuccessful furnace experiment." It refers to experiments with various furnaces of the type in which the heat is obtained by downward radiation from electrically heated resistors arranged near the top of the furnace chamber, as described in our vol. VIII, pp. 289 and 317, and vol. IX, p. 259.

Various furnaces of this type have since been built and tested, but the results have been disappointing in one respect. For, while the working of the furnace, so far as its electric properties were concerned, was most satisfactory, the efficiency was very low, owing to excessive heat losses.

The table below gives certain data in reference to five fur-

naces constructed on this principle, the furnace number appearing in the first column. In the second column the nominal electric capacity is given; in the third column the temperature of the hearth at which experiments were made; in the fourth the heat losses in kilowatts, determined by holding the furnace for a long period at a constant temperature and measuring the rate of generation of energy necessary to maintain that temperature constant; the fifth column gives the percentage loss referred to the electric capacity of the furnace. The columns headed "A" and "B" give certain ratios which will be explained later.

Furnace No.	Capacity, Kw.	Temperature Cent.	Losses, Kw.	Losses, Per Cent.	A.	B.
1	40	1300	18	45	0.20	1.1
2	40	1240	15	38	0.20	1.1
3	150	1250	33	22	0.29	1.0
3	150	1450	42	28	0.29	1.0
4	75	1100	38	51	0.20	2.0
4A	75	1200	35	47	0.22	2.0

Furnace 1 had walls 230 mm. (9 in.) thick. The resistor was "open," that is, it radiated directly on the hearth.

Furnace 2 had walls 345 mm. (13.5 in.) thick, the outer part of the wall being built of hollow bricks. The resistor was "enclosed," that is, there was a septum of refractory material interposed between the resistor and the hearth.

Furnace 3 had walls 345 mm. (13.5 in.) thick, and between these walls and the iron casing of the furnace was a space of 30 mm. (1.2 in.) filled with kieselguhr.

Furnace 4 was of the design described in our vol. IX, p. 259, and had double walls, each 230 mm. (9 in.) thick, separated from one another by an air space 25 mm. (1 in.) wide.

Furnace 4A was the same as Furnace 4, except that part of the walls was covered with a layer of heat insulating bricks 55 mm. (2.2 in.) thick.

Now, since Furnaces 4 and 4A were supposed to be the most improved model of this kind of furnace, the very high heat loss was extremely disappointing. In looking for an explanation of the poor results obtained, the first thing naturally suggested was that the septum between the resistor and the hearth might offer such a high resistance to the flow of heat that the efficiency of the furnace would be low. However, as this hypothesis was formed during the experiment, it was an easy matter to test it by the simple expedient of knocking out the septum. As this produced no appreciable improvement in the working of the furnace, the hypothesis ceased to be useful.

There then seemed to be two other explanations of the phenomenon; either the design of the furnace was radically wrong, or the loss of heat by conduction through the furnace walls was excessive.

The heat energy radiated from the resistor is absorbed by the charge on the hearth of the furnace on the one hand, and by the walls and roof on the other. Now, the heat absorbed by the walls and conducted away to the surroundings of the furnace is lost, and, consequently, other things equal, it is desirable to keep the ratio between the hearth area and the total area receiving heat from the resistor as large as possible. This ratio has been calculated for the various furnaces and is given in the column marked A in the table. It will be observed that the ratio is greatest in the case of Furnace 3, and this is at any rate a partial explanation of this furnace's superiority.

Another important consideration is the ratio between the area of the surface, other than the hearth, receiving the heat radiated from the resistor, and the rate of generation of energy in the resistor. This ratio has been calculated for the various furnaces, and is given in the column marked B. Here it is seen that the ratio for furnace 4 is twice as great as in the case of furnace 3; consequently, other things equal, the heat losses would be twice as great.

It does not follow, however, that, because the ratio B is high, the design of the furnace is faulty, for other considerations may make such a ratio desirable or even necessary. The point that requires close attention is the most perfect heat insulation possible. If the heat insulation is good, then, while there may be a great difference in temperature between the charge on the hearth of the furnace and the resistor, there will be very little difference between the rest of the surface receiving radiation and the resistor. Here was found the great weakness in furnace 4, for the walls, instead of being good insulators, were relatively good conductors of heat. Observation showed that when a balance had been reached, so that the total energy radiated from the outside of the furnace was equal to the energy generated in the resistor, there was a marked difference between the temperature of the resistor and the interior surface of the furnace. There were no means by which these temperatures could be measured, but by the eye it could be seen that the difference amounted to at least 200° C.

A very clear demonstration that was approximately quantitative of the high heat conductivity of the walls was made by comparing furnace 4A with furnace 4. In furnace 4A the side walls were partly covered with bricks having high heat-insulating properties, about 25 per cent of the total exterior surface of the furnace being protected in this way. From the table it is seen that, working with even a higher temperature than in the case of furnace 4, the losses were considerably reduced. It is unfortunate that the determinations were not made at the same temperature, but it is safe to assume that, if furnace 4 had been tested at a temperature of 1200 deg. C., the losses would have amounted to at least 43 kw. Hence by the improved heat insulation the loss was reduced about 19 per cent.

In building furnace 4, a good grade of firebrick, so far as its refractory properties are concerned, was used. It was recognized that silica brick are better heat insulators than fire brick, as may be seen from Wologdine's determinations,* which show that the heat conductivity of silica brick is 0.0020, and of fire brick 0.0042. On account of its high co-efficient of expansion by heat, silica brick is sometimes objectionable, and it was thought that, with a wall of fire brick 460 mm. (18 in.) thick and an air gap, the heat insulation would be amply sufficient. The air gap, however, was useless, for a reason that has been well expressed by Ray and Kreisinger.** The surface of the inner wall forming the air gap reached a temperature of at least 600 deg. C., so that the very device on which dependence was placed to prevent heat losses was worse than useless.

In the discussion of Mr. FitzGerald's paper Dr. Whitney emphasized that the description of unsuccessful experiments should be encouraged for the valuable lessons which they teach.

Mr. Lidbury read a communicated discussion by Mr. John Thomson, who had been Mr. FitzGerald's co-worker in the design of the furnace. He pointed out two special differences in the design of the later and earlier furnaces, which might possibly have contributed to the difference in the results; he also expressed the opinion that the point made with respect to the air gap was not yet fully proven.

Titanium in Iron and Steel

A paper by Mr. Charles V. Slocum, of the Titanium Alloy Manufacturing Company, of Pittsburgh, Pa., refers to the work of Mr. Auguste J. Rossi with titanium continued during 30 years or more of his life, until he produced his "titanium alloy" in the electric furnace in 1903. Since that time the manufacture and consumption of titanium in the steel industry has grown with leaps and bounds.

Metallurgists have practically all agreed that the use of a small percentage of an alloy of titanium is of benefit in both iron and steel, but some question has arisen as to the form which this alloy should take. In a recent publication*** the statement has been made that carbon-free metals dissolve more easily than other metals containing carbon, since the latter

*"Electrochem. and Met. Industry," Vol. VII, (1909), page 383.
**"The Flow of Heat Through Furnace Walls," by Walter T. Ray and Henry Kreisinger; Bulletin 8, Bureau of Mines, Washington, 1911.
***H. Goldschmidt in this journal, July issue, p. 348.

contains carbides. The experience of the present author does not bear out this fact, he having found that a carbon-free alloy was not at all adapted to use in the iron and steel industry, and the manufacturers were obliged to make a product containing from 5 per cent to 8 per cent carbon for such use. His early experience with an alloy of iron and titanium free from carbon, but containing 5 per cent to 10 per cent aluminium, resulted in almost complete failure, as the aluminium when present in any appreciable quantity made the steel brittle and its oxide showed great tendency to remain in the steel.

"Titanium alloy" contains 5 per cent to 8 per cent of carbon, mostly in the form of graphite. Analyses of this material made in the laboratory of Dr. C. F. McKenna show the following proportions of the two forms of carbon:

Sample No.	Graphitic Carbon	Combined Carbon
141	9.601	0.147
162	9.179	0.12
291	7.012	0.13
298	6.234	0.118

From these analyses it appears that titanium acts very much like silicon in that it causes a separation of carbon as graphite.

Dr. G. B. Waterhouse recently cited an interesting experiment to bear out this fact: A ladle of iron for making malleable castings was treated with 10 per cent titanium alloy sufficient to equal 0.1 per cent metallic titanium added to the iron. The original iron gave castings showing a white fracture with a slight graphite in the center. The titanium-treated metal showed a white border between ⅜ and ½ in. (1 to 1.25 cm) deep, and a gray center showing the separation of graphite. Contrary, therefore, to the viewpoint of Dr. Goldschmidt, there need be no fear of the non-solution of the electric-furnace titanium alloy, as it does not contain more than the merest fraction of the unreadily soluble carbide.

The density of ferro-titanium has an important bearing on its incorporation in the molten bath. Careful determinations made on the electric-furnace product containing both 10 and 15 per cent titaniums, show a density of 6.20 to 6.40. The alloy produced by the alumino-thermic process shows a density of between 6.20 and 6.30. The difference between the densities of the two alloys is therefore hardly noticeable.

The alloy is best added to the ladle of steel after tapping from the furnace and before the slag begins to run. For open-hearth steel the supply should be placed convenient to the ladle and shoveled in precisely as so much coal. For soft steel an addition equivalent to 0.03 per cent titanium is sufficient to make the steel more ductile. Larger quantities are of increased benefit, but cannot always be added because of the carbon content and also because of the density imparted to the steel, which increases as the proportion of alloy is increased, and therefore is not always desirable. In high-carbon steels this increased density is desired and more alloy is used. The addition of the alloy permits the use of a considerably higher carbon without increasing the brittleness, and several railroads are now using as much as an equivalent of 0.10 per cent titanium or more in securing tough, durable rails.

In the crucible practice the best results have been obtained by adding the titanium alloy to the pots before they are removed from the furnace, giving more time for washing and deoxidizing before teeming.

Leading authorities seem to agree that titanium achieves its remarkable results through its strong deoxidizing powers, together with its effect of giving the slag formed sufficient fluidity to completely separate it from the metal.

"The presence of titanium oxide lowers the melting point of slags occluded in iron and steel and imparts thereto sufficient fluidity to account for their elimination."[2]

"Titanium . . . has a stronger affinity for oxygen than have the well-known deoxidizers; . . . it probably gives the slag such a consistency that it separates more completely from the molten iron."[3]

"The treatment of all steels with ferro-titanium for the purpose of purifying the metal is strongly recommended, the presence of nitrogen to the extent of 0.02 to 0.035 or 0.045 in certain steels being enough to cause the metal to break asunder, destroying all elongation and reduction of area."[4]

"The great affinity of oxygen and titanium is an absolutely sure means of completely deoxidizing the steel, the advantages of which need no further elaboration. It has been proved that an ingot of steel containing a very small quantity of titanium would show practically no segregation."[5]

The cost of treatment with titanium varies from a minimum of 25 cents to a maximum of $2 per ton of metal treated. It is the cheapest deoxidizer above the grade of manganese or silicon, and a far greater purifier than any other alloy. The amount to be used is very small, and should be proportioned according to the carbon content of the steel and to the amount of impurities present. To the electrometallurgist it is a triumph worthy of record that Rossi should take an element like titanium, so long considered useless, and make out of it so wonderful a servant in the iron and steel industry.

Mr. Slocum's paper brought forward a lively discussion.

A communicated discussion by Dr. Richard Moldenke, Secretary of the American Foundrymen's Association, was read by Mr. Lidbury. It referred especially to the use of titanium in foundry practice. Dr. Moldenke attributes the success of titanium treatment to its action as a deoxidizer. It removes the oxidation effects which are due not only to remelting under oxidizing conditions (cupola) but can also be traced directly to the blast furnace. The removal of this oxidation, which is small, is important and this is what titanium does.

Because the oxidation is small Dr. Moldenke points out that only a small amount of titanium is needed and useful. If more is added, it makes apparently no difference. This would mean that titanium acts simply as a deoxidizer, not as an alloying element. A practical conclusion from this is that the titanium treatment of metal should be adjusted to the amount of oxidation present.

Mr. Wm. C. Cuntz, general manager of the Goldschmidt Thermit Co., then read a communicated discussion of Dr. Hans Goldschmidt who first replied to Mr. Slocum's claim that a carbon-free ferrotitanium was not adapted to the use in the iron and steel industry and emphasized that this claim was refuted by the undisputed practical results of a large number of prominent metallurgical works which use continually carbon-free ferrotitanium made by the aluminothermic process.

Dr. Goldschmidt does not claim that ferrotitanium containing carbon is unsuitable or useless. But he claims that the carbon-free ferrotitanium, containing aluminium, is a more effective reagent. His first reason is that ferrotitanium which contains little or no aluminium, alloys with steel only with difficulty. When Dr. Goldschmidt had placed on the market in 1897 a 35 to 40 per cent. ferrotitanium containing very little aluminium, an open-hearth steel plant to which he had furnished an alloy of especially high percentage, inquired whether he could not make the product at very low cost, since they were inclined to use this ferrotitanium as an open-hearth furnace lining; so little was it affected by the molten steel. Dr. Goldschmidt could satisfy his steel customers only after he added aluminium to the alloy so that it contained 1 part of aluminium to 3 or 4 parts of titanium.

Dr. Goldschmidt pointed out that the addition of aluminium to the alloy had a threefold purpose. First, it caused the alloy to be embedded and dissolved more easily and intimately in the steel.

Secondly, titanic acid (the product resulting from the titanium reaction in the steel bath) is very difficult to fuse.

[2] Bradley Stoughton, U. S. Patent Office Proceedings, Ser. No. 463610.
[3] Henry M. Howe, Ibid.
[4] H. le Chatellier, Paper, Congress of Metallurgists, Belgium, 1905.
[5] E. von Maltitz, Stahl und Eisen, No. 41, 1909.

Also alumina (the product resulting from the aluminium reaction in the steel bath) is very difficult to fuse. Hence, if either titanium or aluminium alone are used as deoxidizers, the products of the reaction—titanic acid or alumina, respectively—are liable to remain as suspended particles in the steel, as can be shown by microphotographic methods. But if titanium and aluminium are used together simultaneously, the product of the reaction is titanate of aluminium, which is much more easily fusible than titanic acid or alumina alone. Hence the titanate of aluminium will separate out of the steel bath and pass into the slag much more easily.

Dr. Goldschmidt's third reason for adding aluminium to his ferrotitanium is that when the aluminium serves its purpose of cleaning and deoxidizing it enables the titanium to manifest more strongly its own specific effectiveness.

Dr. Goldschmidt denies that titanium is a stronger deoxidizing agent than aluminium and points out as proof that aluminium reduces titanic acid, but titanium does not reduce alumina. But he agrees that titanium is a good purifier and stands in the first rank as an agent for the production of a dense, non-porous steel. Besides acting as a deoxidizer and for removing nitrogen the titanium itself has some beneficial effect on steel and cast iron.

In the cleansing of the metal bath the great affinity of the added metal for oxygen and its speed of reaction are important, but Dr. Goldschmidt emphasizes that it is undoubtedly the specific properties of the added metal itself which give the result. For example, metallic calcium has a greater affinity for oxygen than aluminium, but in spite of this, though many attempts have been made, calcium is not generally used in steel making. The alloy of calcium and silicon which he has produced by his process as a substitute for aluminium and has put on the market has not shown the results expected and hoped for in spite of the fact that this alloy is decidedly a good deoxidizer. Practical results have not equalled what aluminium alone and what to a far greater extent the aluminium-titanium alloy has accomplished.

Dr. Goldschmidt stated that he had always followed with great interest Mr. Rossi's work in the titanium field. "Mr. Rossi has worked hard to get on the market a high-per cent. ferro-titanium. He had finally to choose a product running 10 to 16% titanium with a high carbon content, in order to make its introduction into steel possible. It is perfectly apparent that a low-percentage alloy, over-ballasted as it is, is not as desirable as a high-percentage alloy. But I do not wish to minimize in any way the services of Mr. Rossi as I appreciate all the difficulties he has overcome in producing the results he has produced."

Dr. Goldschmidt finally referred to the successful use of titanium thermit for the introduction of small quantities of titanium into cast iron and steel, as proof that he has also devoted himself to the metallurgy of titanium for some years and has produced results. Titanium thermit is a thermit which when introduced into iron or steel liberates a fluid alloy of ferrotitanium, while the slag rises to the surface. This has a double effect, an addition of a small amount of titanium to the bath and a violent stirring up with a slight rise of temperature. The quality of cast iron is thereby decidedly improved.

The discussion was continued by Mr. F. A. J. FitzGerald, who thought that the claimed presence of carbides in ferrotitanium containing carbon was not proven and that the reasons for the claim were weak. He asked: If titanium has really such a tendency to form carbides, why should not carbides be formed if carbon-free ferrotitanium is added to molten steel?

Dr. Waterhouse, of the Lackawanna Steel Company, remarked that the claim as to the carbides had apparently been withdrawn by Dr. Goldschmidt as it was not mentioned in his reply. (To this Mr. Cuntz replied afterwards that Dr. Goldschmidt had further investigations on titanium in hand and would give his results on a later occasion.)

While Mr. Slocum recommends that the alloy is best added to the ladle of steel *after* tapping from the furnace and before the slag begins to run, Dr. Waterhouse thinks it is best added *during* tapping.

An important point, according to Dr. Waterhouse, is that time must be allowed for the reactions to go on and for the products of the reactions to separate out.

In his final reply, Mr. Slocum referred to early experiments of Mr. Rossi in the reduction of titaniferous iron ores. He thought that if aluminium is beneficial in the titanium raaction it could be added separately. He finally gave figures on the rapid increase in the production of ferrotitanium at the Niagara Works and its consumption in the steel industry.

Niagara Transforming Stations.

A paper by Mr. A. J. Jones, of Niagara Falls, gives a review of the "transforming stations of Niagara electrochemical and electrometallurgical industries."

Grouped around Niagara Falls are seven hydro-electric generating stations, four of which are on the Canadian side. From these seven stations energy is being used at the rate of 128,000 hp for electrochemical purposes, 56,000 hp for railway service, 36,000 hp for lighting, and 55,000 hp for various industrial services, the total being 275,000 hp or about 5.5 per cent of the available power of the cataract. Of this total amount 146,000 hp is employed locally in industries that have been attracted to Niagara Falls by reason of the generating stations located there, electrochemical processes forming 87 per cent of this amount and 46 per cent of the total amount utilized.

The paper is essentially a catalog of the equipment of the transforming stations of the following companies, with an account of the more salient features of the equipment: International Acheson Graphite Company, Aluminium Company of America, Carborundum Company, Niagara Lead Company, Castner Electrolytic Alkali Company, Norton Company, Union Carbide Company, Niagara Alkali Company, Hooker Electrochemical Company, Niagara Electro-Chemical Company, United States Light and Heating Company, and Electro Metals.

In the discussion which followed, Mr. Fitzgerald pointed out that electrochemical newcomers to Niagara are often astonished at the amount of apparatus they have to put in. The plants at Niagara often have a cramped appearance. Usually not enough space is allowed. This is an important matter.

Dr. Northrup referred to a mercury ammeter, based on the pinch effect and being in use at the Aluminium Company's plant at Massena; it measures 14,000 amp regularly; the capacity being 16,500 amp. The instrument is in continuous operation, no failure having ever occurred.

Production of Molybdenum Steel in the Electric Furnace.

A paper by Messrs. E. T. Dittus and R. G. Bowman gave an account of an investigation carried out in the Colorado School of Mines.

The use of a sulphide ore of so active a metal as molybdenum in connection with the manufacture of steel presents a number of problems. Foremost among these are the complete reduction of the molybdenum without serious loss, the diffusion of the molybdenum through the steel to form a homogenous product and the elimination of the sulphur from the steel. The process employed was one based on a reaction described by Mr. F. M. Becket in U. S. Patent 855157, and is believed to be new.

In the production of steel direct from ore in the electric furnace the removal of certain impurities, particularly sulphur, offers difficulties. For the reduction of molybdenite in the presence of molten iron it is necessary to have present some substance which has a greater affinity for sulphur than either molybdenum or iron. The compound formed by the desulphurizing agent and the sulphur must either pass into the

slag or be volatilized as soon as formed. The two metals which seem best adapted for use as desulphurizers, under these conditions, are manganese and silicon.

Manganese in the form of metal, or as the ferroalloy, reacts with sulphides at high temperatures according to the following reaction:

$$2\,Mn + RS = R + 2MnS.$$

The manganese sulphide forms a slag resembling iron sulphide. If the above reaction takes place in a bath of molten steel or iron the resulting metal is apt to contain small included masses of manganese sulphide. Sulphur in this form has little effect on the properties of steel. This reaction might be applied to the production of alloy steels, such as molybdenum

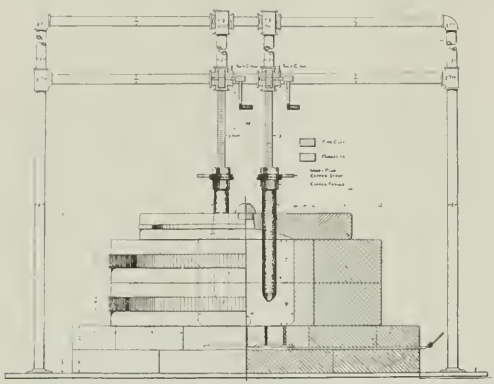


FIG. 1.—LABORATORY FURNACE FOR PRODUCTION OF MOLYBDENUM STEEL

steel, by adding a mixture of molybdenite and ferromanganese to the bath of molten steel just before tapping. This would result in the formation of ferromolybdenum and manganese sulphide; the former alloying with the steel and the latter passing into the slag.

Silicon reacts with sulphides to form silicon sulphide SiS_2 and liberates the metal of the original sulphide. This is the reaction described by Becket.[1] The reaction has been investigated by Fielding[2] who produced a yellow powder which sublimed at 1500° C and which decomposed water with the formation of H_2S and silicic acid. This compound did not correspond to the formula SiS_2. Sabatier[3] describes a somewhat similar compound and suggests the formula Si_2S_6. The heat of formation of SiS_2 is given by Sabatier[3] as + 40.4.

This reaction might be applied to the production of molybdenum steel in the same manner as the manganese reaction described above. The silicon, in the form of ferrosilicon, and the molybdenite should be reduced to powder, intimately mixed in the proper proportion to produce the reaction and added to the steel in the furnace before tapping. The mixture might be added in a soft iron tube or in small briquets made up with a binder of sodium silicate. Addition to the ladle during tapping would probably result in raising the sulphur content of the metal on account of the absence of slag, which in the furnace would remove small amounts of sulphur which might tend to pass into the steel.

The experimental work was done in an electric furnace which could be operated either as a Heroult furnace (two arcs in series on the top) or as a Girod furnace (two arcs in parallel on the top and a metallic electrode in the hearth).

The crucible is elliptical in cross section and measures 6 in. x 9 in. x 9 in. (15 x 23 x 20 cm.) deep. The total volume of the crucible is about 360 cu. in. (5.5 litres), the volume of the smelting zone is about 90 cu. in. (1.4 litres). The walls have a slight bosh to effect a concentration of heat at the base of the crucible. The crucible lining is of burned magnesite obtained by crushing magnesite brick to pass 10 mesh (2.5 mm.) The magnesite was mixed with tar to form a paste and rammed in hot around a central wooden form. The tar was burned out in the heating up of the furnace.

The walls are built up of eight fire-brick sections of special design, luted together with fire-clay and encircled by steel bands. These rest on a foundation made up of two courses of standard fire-brick, with the joints filled with fire-clay.

The cover for the crucible is a solid elliptical fire-brick section 17 in. x 20 in. and 2½ in. (43 x 50 x 53 cm.) thick, encircled by a steel band. The electrodes enter through the slot in the center of the cover. This slot admits of considerable lateral or angular movement of the electrodes. The under side of the cover is slightly concave. An asbestos gasket is placed under the cover where it rests on the top of the furnace, and small asbestos washers encircle the electrodes where they pass through the cover. The tap-hole is 1¼ in. (3 cm.) in diameter, and is well rounded at its point of entrance to the crucible.

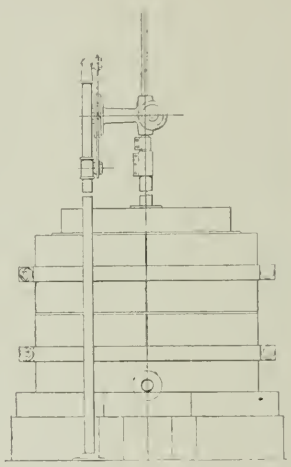


FIG. 2.—MODIFICATION OF FURNACE DESIGN, SHOWING ELECTRODE SUPPORT

The hearth electrode consists of six ½-in. (1.3 cm.) rods of Swedish iron screwed into a ⅜-in. (0.9 cm.) plate of the same material. The plate is embedded in the lining of the bottom of the crucible in such a position that the upper ends of the rods are flush with the inner surface. A copper strap 3-16 in. (0.5 cm.) thick and 1.5 in. (4 cm.) wide, bolted to the plate, extends outside the furnace for connection to the source of current.

The two movable electrodes are supported on a frame above the furnace. The supporting frame is entirely independent of the furnace proper and is therefore unaffected by expansion or contraction of the walls. The electrodes are cylindrical graphite rods 1.25 in. (3 cm.) in diameter and 11 in. (28 cm.) long. Each electrode is attached to a brass rod by means of a collar clamp and cotter pin. The rod carries a rack which meshes with a pinion mounted on the electrode support. The electrode supports are mounted to slide from right to left on the supporting frame. This facilitates adjustment of the electrodes while the surface is in operation and also allows the electrodes to be pushed back out of the way when relining the crucible. The support is so constructed that the electrode may be swung about it as a center and inclined at any angle. Ordinary pipe fittings were used as much as possible in the construction of the frame and supports since these give a light, strong construction, and require no special castings or forgings. An electrode support built entirely of pipe and fittings is illustrated in Fig. 2.

The furnace illustrated in Figs. 2 and 3 differs from that described above in having a crucible of double the depth de-

[1] Elec. Chem. & Met. Industry, Vol. V., p. 237 (1907.)
[2] Bul. de Soc. Chem., Paris 2, 38, 152.
Comptes Rendus, 90, 819.

scribed, i.e., 16 in. (40 cm.), and a different arrangement of the rods in the bottom of the crucible. This furnace was designed to operate continuously with a column of cold charge resting on the molten material, as in a shaft furnace.

This type of furnace may be operated as a Girod steel furnace by connecting the two movable electrodes in parallel with one side of the circuit and the hearth electrode with the opposite side. The movable electrodes may then be placed parallel vertically, thus forming two arcs, or set to converge to a single central point forming but one arc. To operate the furnace as a Heroult steel furnace, the points of the rods in the hearth electrode are covered with a layer of magnesite, and the two movable electrodes are connected in series with the circuit. The magnesite covering for the hearth electrode need not be of any great thickness, since there is but a slight tendency for the current to cross through the iron plate after the charge becomes molten.

The furnace is charged at the beginning of a run by removing the cover and distributing the charge with a trough or funnel. Additional material may be added through the opening in the cover while the furnace is running.

The small amount of coke in the charge used was found

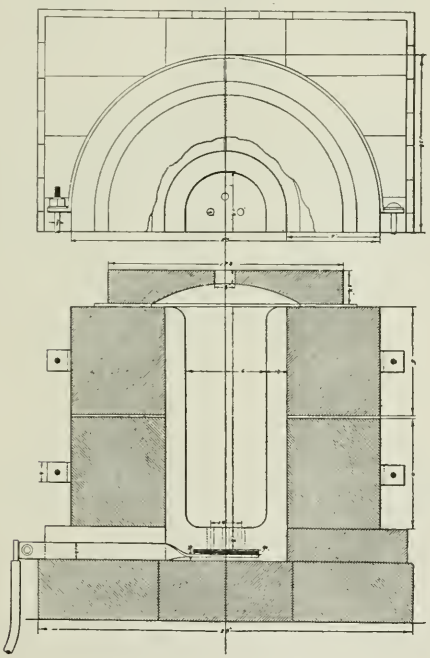


FIG. 3.—MODIFIED FURNACE DESIGN

to be sufficient to make the cold charge slightly conducting, the resistance rapidly decreases with rise of temperature, and the furnace may thus be readily started on a cold storage charge, providing the walls of the crucible are hot. The electrodes are forced down through the charge to within a short distance of the bottom of the crucible at starting. As the charge becomes hot, the electrodes are gradually raised until the charge is reduced and the arcs play between the ends of the electrodes and the surface of the slag. With the current available it was found impossible to maintain arcs below both electrodes simultaneously for any length of time. The best results were obtained by raising and lowering the electrodes alternately at intervals of two or three minutes, which caused the arcs to alternate from one electrode to the other electrode.

The capacity of the furnace as described is about 1,000 gm of metal per charge, or approximately 2,000 gm. of raw charge.

Single-phase alternating current at 60 cycles, 25 volts, was used, with an average amperage of 250.

The iron ore employed was a good grade of hematite from the mines of the Colorado Fuel and Iron Co. at Sunrise, Wyoming, the analysis being 66.06 per cent Fe., 1.52 Al_2O_3, 4.12 SiO_2, 0.04 P and 0.04 S. The ore was crushed to pass eight mesh (3mm.) and the fines were retained. A quantity of molybdenite from Yorkes Peninsula, Australia, was used, being practically pure MoS_2 with only a trace of impurity in the form of copper sulphide.

Ferrosilicon was used with an analysis of 49.7 per cent Fe, 50 Si, 0.005 S, 0.20 C.

The ferrosilicon and molybdenite were ground to pass thirty mesh (0.85 mm.) and intimately mixed in the proper proportions to bring about the reaction:

$$MoS_2 + Si = Mo + SiS_2.$$

The proper amount of this mixture to give the desired molybdenum content in the steel was enclosed in a small paper tube and added to the bath of metal just before reduction was complete.

Analysis of the coke and lime (ordinary builders' lime) are also given. Both were crushed to pass eight mesh (3 mm.) and the fines retained.

Six furnace runs are given in the paper. There was considerable trouble in tapping, as generally the metal would not flow. Only one of the runs is here given.

The charge consisted of 3025 grams iron ore, 660 coke, and 412 lime, the calculated iron content being 2000 grams.

The length of the run was 1 hour, 30 minutes, the mean amperes 200, mean volts 25, power 5 kw.

1,700 grams of metal were obtained. The calculated molybdenum content was 2.5 per cent. The run is described as follows:

Iron turnings were spread over the bottom of the crucible and covered by a layer of coke, lime and silica to form slag and protect the iron. The metal melted in 25 minutes but was kept molten in the furnace for 15 minutes after fusion in order to burn out the tar from new portions of the lining. This metal was tapped cleanly and a charge sufficient for 500 gm. of metal was added. This reduced in 20 minutes. Molybdenite-ferrosilicon mixture sufficient to give a content of 5 per cent. Mo in the metal was added in a paper tube just before reduction was complete. The tap hole was opened but the metal was level with the tap and only slag flowed out. The tap hole was again plugged and a charge sufficient to give 1,000 gm. of metal was charged.

Reduction was rapid and the charge was entirely reduced in 30 minutes. Molybdenite mixture sufficient to give a molybdenum content of 2.5 per cent. in the metal was added just before final reduction of the charge. Since the bottom of the crucible appeared to be sinking constantly, another charge equivalent to 500 gm. of metal was added and reduced in 15 minutes. This gave a total of 2,000 gm. of metal (calculated) in the furnace, with sufficient molybdenum to yield 2.5 per cent. in the total.

The tap hole was opened and slag flowed out freely but no metal was obtained. The current was shut off and the furnace allowed to cool. A solid mass of metal level with the tap hole was found in the bottom of the crucible.

The furnace was taken down and the entire lining and bottom electrode removed. A solid mass of metal weighing 1,700 gm. was obtained. The metal was malleable and tough, and was broken with great difficulty. When broken across the center the fracture was very fine grained and dense at the center of the mass and coarsely crystalline and full of small blow holes at its outer edges.

A section of the central portion of the mass, polished and etched with picric acid showed a network of a bright white

constituent, probably a double carbide of molybdenum and iron. A section in the coarsely crystalline portion of the mass showed a coarsely granular structure.

The analysis of the metal showed 0.62 per cent. C, 0.91 Si, 1.15 Mo, 0.08 P, 0.37 S.

In one of the later runs molybdenite concentrates from Woods flotation process were used. Most of the tests were qualitative rather than quantitative, so that further experiments would be necessary in order to determine the loss and distribution of molybdenum.

Since no analyses were made upon the slag, it is impossible to determine what proportions of the sulphur were removed by the ferrosilicon and the slag.

The following final conclusions are drawn:

(1) Molybdenum steel can be made in the electric furnace by the direct reduction of iron ore and the addition of molybdenum in the form of molybdenite, MoS_2.

(2) Molybdenum steel of low sulphur content can be produced from molybdenite by the use of ferrosilicon as a desulphurizer.

(3) Molybdenum steel of low sulphur content can be produced from molybdenite in the form of low grade concentrates by the use of ferrosilicon as a desulphurizer.

With regard to the design of a furnace for small scale operations, the experiments seemed to indicate that:

(1) The Girod principle as used was superior to the Heroult.

(2) A tilting furnace would be more effective than a stationary furnace.

(3) The tap hole on a stationary furnace should be made short, with a steep inclination. Ample provision for heating the tap hole should be made.

(4) Tar does not make a satisfactory binding material for crushed magnesite.

In discussing this paper Mr. FitzGerald pointed out there was danger in drawing conclusions from such small-scale experiments. He objected to the use of the names of Heroult and Girod for designating the two respective types of electric furnaces.

Mr. Hansen thought that the use of the two names was useful as a brief name for the two furnace designs, but Mr. FitzGerald objected to it as "electrochemical mythology."

Dr. Whitney liked the paper as coming from two young enthusiastic students. Such work should be encouraged.

Design of a 30-Ton Electric Induction Furnace

A paper by Mr. Albert Hiorth, of Kristiania, Norway, describes the design of a 30-ton induction furnace based upon the data and results obtained with the author's original 5-ton induction furnace in Jossingford, Norway.

The two most important things to be determined in the design were the power factor and the energy required to keep the charge at the working temperature.

The power factor of the furnace depends essentially on the ratio of the resistance to the reactance (inductance). The effective resistance consists of three parts, as is indicated by the fact that the power consumption in the furnace consists of three items, two representing the losses, namely, the loss due to the resistance of the primary coils and the hysteresis and eddy-current loss in the iron core; while the third item is the production of useful heat from electrical energy in the (secondary) bath; the secondary resistance may therefore be termed the useful resistance. The useful resistance depends essentially on the conductivity of the iron at the working temperature of the furnace and on the dimensions of the bath.

The author takes from his records figures for the volts, amperes and watts of his 5-ton furnace and calculates the total resistance. From this he deducts the resistance of the primary coils and the resistance corresponding to the losses in the iron core. He thus obtains the equivalent useful resistance of the bath. Figures are given for this useful resistance for different charges in the bath (that is, for different cross-sections of the molten metal). The resistance should be proportional to the cross-section.

This is indeed approximately fulfilled and from his figures he obtains a specific resistance of the molten iron equal to 1.4 ohms per sq. mm. cross-section and m. length. In handbooks the value given is somewhat higher, and even as high as 1.7. The discrepancy may be due to inaccuracies, which are quite unavoidable, considering the difficult nature of the measurements, or may be due to the fact that in the present case the section of the bath is not absolutely uniform and that, for instance, the central portion of the bath, where the two rings meet, represents a somewhat lower equivalent resistance than the rest. The value 1.40, it must be remembered, includes all the irregularities in the shape of the bath, and also the skin effect, as far as this influences the resistance. It can only be used for figuring the resistance of furnaces of similar shape and disposition of the bath.

The reactance (or inductance) of a given furnace and for a given frequency is independent of the weight of the charge and remains constant all the time.

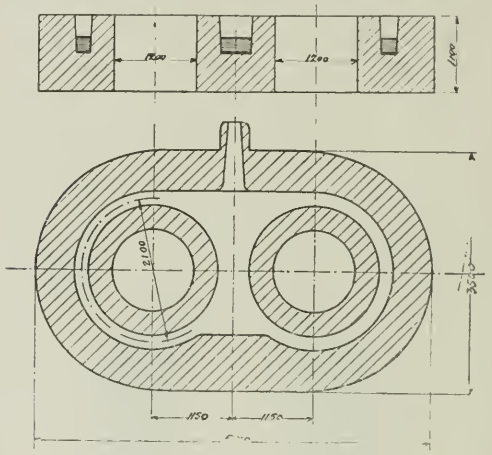


FIG. 1.—DESIGN OF 5-TON FURNACE.

Using the figures thus found for the resistance and inductance of his 5-ton furnace, he figures out theoretically the power-factor of the 5-ton furnace with different charges, and obtains the following table:

With	3 tons	Cos. ϕ	$=0.81$.
"	4 "	"	$=0.73$.
"	5 "	"	$=0.65$.
"	6 "	"	$=0.60$.

These figures correspond very well with the actual figures.

With regards to the design of a 30-ton furnace the first condition, of course, is that the space for the charge must be large enough to hold 30 tons, but the actual dimensions and shape of the bath must be such as to give the best possible power-factor.

In order to increase the capacity of the furnace, it is necessary to increase the sectional area of the bath more than the length, as otherwise the diameter of the bath, and consquently the dimensions of the furnace, would become too large. The resistance of the bath will thus be considerably decreased and at the same time the inductance will be increased, owing to the large diameter, which increases the area of the space between the primary and the secondary windings (hence the magnetic leakage).

The power-factor will thus in any case be reduced with the increased capacity, and the author has figured that a 30-ton furnace built on the same line as the 5-ton one, for single-

phase current, with a two-leg magnet and a current supply with 25 alternations, would show a power-factor equal to 0.25, and that the power-factor even with 15 alternations per second would still be only 0.38.

It was therefore decided by the author that the 30-ton furnace should be built on the three-phase principle instead of the single-phase. In this case the weight of the charge per ring is only 10 tons instead of 15 tons with a single-phase furnace and the reduction of the power-factor is not nearly so great.

In order to facilitate the tilting of the furnace, the three rings were disposed in one row.

After having finally chosen the dimensions for his new 30-ton furnace, he calculates the resistance and inductance and finds from these the power-factor which comes out as 0.34, which is still too low.

In order to increase the power-factor, it is necessary to further decrease the inductance by using a lower periodicity. What can be obtained in this way is shown in the following table:

Periodicity	Power-factor
25 alts.	0.34
20 "	0.41
16 "	0.50
10 "	0.67

From this table he selects a periodicity of 16 alternations per second. To decrease the periodicity further still will hardly

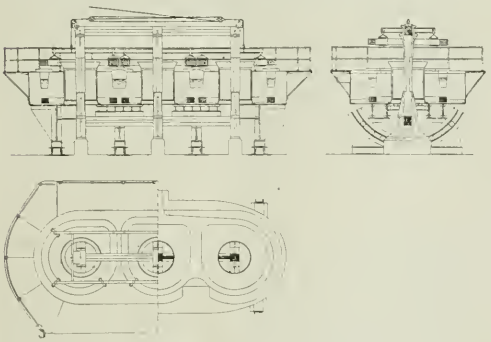


FIG. 2.—DESIGN OF 30-TON FURNACE

pay, seeing that the cost of the generator will thus be further increased, even if the output is reduced on account of the power-factor being improved.

It remains to figure out the necessary energy required for the furnace, the dimensions of the iron-core and the number of primary turns, together with the sectional area of these.

As the furnace will be used with a liquid charge, and no melting-down will take place, the amount of energy required is only the amount necessary to keep the bath at the required temperature. Apart from the small losses in the primary coil and the iron core, all the energy is converted into heat in the charge itself, and, as soon as a constant temperature is reached finally conducted to the surface of the furnace and given out from this by radiation, conduction and through the cooling arrangements. If water cooling is arranged in the same way and the same amount of water increased correspondingly, it may therefore be assumed that the loss of heat, and consequently the energy required to keep the charge hot, will be in proportion to the surface of the furnace.

According to the dimensions of the two furnaces the surface of the new furnace is between two and two and a half times as large as the old one. The amount of energy required in the old one to keep the charge in a molten condition during the night was approximately 180 kw., and during the refining the energy varied between 250 kw. and 350 kw. The minimum energy required for the new furnace should consequently be $2 \times 180 = 360$, and the maximum, $2.5 \times 350 = 875$. He assumes 700 kw., but at the same time the generator should be constructed in such a way that, by merely altering the voltage it can give anything between 400 and 1,000 kw. With 700 kw the current per ring becomes approximately 46,000 amperes, and the current density in the bath will be about 0.34 ampere per sq. mm.

The further figures are then easily calculated.

The table below shows the principal data for the old furnace and the new one:

	Old Furnace.	New Furnace
Total capacity	5 tons	30 tons
Capacity per leg	2.5tons	10 tons
Diameter of bath	2.1 m.	3m.
Width of bath	20 cm.	30 cm.
Depth of bath	27 cm.	45 cm.
Total surface of masonry	55 sq. m.	110 sq. m.
Total length of platform	8¼ m.	13 m.
Total width of platform	5¼ m.	6.5 m.
Sectional area of core per leg	1800 sq. cm.	1200 sq. cm.
Weight of core	15 tons	23 tons
Number of primary turns per leg	15	13
Sect'nal area of primary windings	1000 sq. mm.	4000 sq. mm.
Weight of copper per leg	0.875 tons	*ca.* 4.5 tons
Total weight of copper	1.75 tons	*ca.* 13.5 tons
Energy used	250 kw.	700 kw.
Copper losses	12.3 kw.	40 kw.
Iron losses	18 kw.	18 kw.
Power-factor	0.65	0.50
Voltage	250	230
Periodicity (½ alternations per second)	12½	8
Amperes per phase	1400	3540
Kind of current	1 phase	3 phase.

The number of primary turns of the new furnace is given above as 13, but in order to be able to run the furnace at full charge, even if the inductance and resistance should prove to be somewhat different from the values given, two turns should be added per leg, making the number of turns in the upper coils 8. The last three turns in each of these coils should be provided with terminals, enabling one to use at will 12, 13, 14 or 15 turns per leg.

The generator should, according to the foregoing, be designed for a normal output of 700 kw., 8 periods (corresponding to 16 alterations) per second, 230 volts, 3,540 amperes, and a power-factor of 0.5. At the same time it must be able to give 400 kw. at 165 volts, and 1,000 kw. at 275 volts, with the same power-factor, and 290 and 125 kw., respectively, with a power-factor of 0.362. The maximum current will thus be 4,300 amperes.

The engine or turbine driving this generator should be for a maximum output of 1,550 effective h. p., and run at a practically constant speed at all loads between 600 h. p. and 1,550 h. p.

This plant should be quite sufficient to supply energy to the 30-ton furnace, even if the constants of this should vary considerably from the figures given above. In order to study this, he has figured out what would happen, and the maximum and minimum energy which could be supplied to the furnace, in case either the resistance or the inductance or both should vary 25 per cent from the assumed value, either up or down. The results are given in this paper in form of a table.

Fig. 1 gives the designs of the old 5-ton furnace, Fig. 2 the design of the new furnaces.

In a communicated discussion Mr. Unger, of the General Electric Company, emphasized that the power installation as proposed in Mr. Hiorth's paper would be very costly on account of the use of so low a frequency as 8. He thought it would be cheaper to combine a number of smaller units in one furnace. In such a way it would be possible to build up an induction furnace of any size for commercial frequencies.

Dr. Waterhouse asked whether there would be any advantage of such a 30-ton pure induction furnace over a Roechling-Rodenhauser furnace or over an arc furnace of the same size.

Mr. FitzGerald replied to this question that the Roechling-Rodenhauser furnace for such sizes would be found greatly superior to other forms of induction furnaces, but made no comparison with any arc furnace.

Mr. Carl Hering referred to some calculations which he had made concerning the possibility of reducing heat losses on the basis of Mr. Hiorth's figures. The final conclusions will be published in an article in this journal. This was the last paper of the morning session.

An enjoyable lunch was then served in one of the halls of the University, and shortly afterward the second session began, being held as in the morning in the lecture room of the Chemistry and Mining Building.

Mineral Resources of Ontario.

The first paper was presented by Mr. T. B. Gibson, Director of Bureau of Mines of Ontario, on the mineral resources of Ontario. With the aid of a map the author sketched the situation and geology of the different portions of Ontario and gave an account of the mineral resources which have been found or may be expected to be found in these different portions. In Eastern Ontario there are a great variety of ores, but northern Ontario is the real home of the metallic deposits.

Mr. Gibson discussed at some length the production of silver, nickel, and gold in Ontario. There have been two eras of silver production in Ontario; the present one began with the opening of the Cobalt mines in 1904.

The nickel industry depends on the nickel-copper ores in the Sudbury district.

The gold industry is at present widely scattered and while it is not yet very successful it may be expected to become of great importance in the future.

Mr. Gibson concluded by suggesting two problems of importance which might be solved by electrochemical means. The first has to do with corundum which is practically pure alumina and is used at present mainly as an abrasive. But it is a richer ore of aluminium than bauxite and a process of making aluminium from corundum would be very profitable.

The second problem is the possible recovery of potash. There are deposits of feldspar in Ontario containing 13 to 14 per cent of potash. If the potash could be gotten out of it we would be able to get independent of Germany which has at present the monopoly in potash salts.

In the discusion Dr. Whitney remarked that another problem which might be worked out profitably is a process for the complex cobalt silver ores.

Addition Agents in Electrolytic Copper Deposition from Solutions Containing Arsenic.

A paper by Dr. Ching Yu Wen and Dr. Edward F. Kern gives an extensive account of an investigation carried out at the School of Mines of Columbia University on "the effect of organic and inorganic addition agents upon the electrodeposition of copper from electrolytes containing arsenic."

Since the object of this investigation and the results obtained were already discussed at some length in our June issue, page 318, it will suffice here to give the chief final conclusion. The combined "addition-agents" of gelatine and sodium chloride, and of glue and sodium chloride, when present in small amounts, are the most suitable and most satisfactory "addition-agents" for copper sulphate electrolytes, especially such as contain a high proportion of arsenic, as with these the deposited copper possesses the greatest ductility and the highest purity. But the combined "addition-agent" of gelatine and sodium chloride (say 0.01 to 0.02 per cent. gelatine and 0.01 per cent. Cl as NaCl) seems to be a little better than sodium chloride and glue, for it gave smoother deposits. In the case of glue and inorganic compounds, the use of cuprous chloride, cupric chloride or hydrochloric acid is just as effective as sodium chloride; but for economic reasons sodium chloride is to be preferred.

In a communicated discussion by Mr. W. L. Spaulding, of the Buffalo Smelting Works, reference was made to experiments on the addition of NaCl to copper refining solutions. In one case the addition of the salt had been found to have deleterious effects, causing a slimy layer on the anode.

Conductivity of Mixtures of Copper Sulphate and Sulphuric Acid.

A paper by Prof. H. K. Richardson and Mr. F. D. Taylor of the Pennsylvania State College gave an account of a series of measurements of the conductivity of mixtures of copper sulphate and sulphuric acid.

In the absence of the authors it was presented in abstract by Mr. Lidbury.

To obtain figures for calculating the voltage drop due to electrolytic resistance in a copper refining tank knowledge is necessary of the conductivity of sulphuric acid and copper sulphate mixture. This was measured by the authors in different proportions at two different temperatures, namely, 25 deg. C. and 45 deg. C.

CONDUCTIVITY IN RECIPROCAL OHMS PER CM.

Grams H_2SO_4 per 100 c.c.	$CuSO_4.5H_2O$	25° C.				45° C.			
		0	5	10	15	20	0	5	10	15	20
Grams per 100 c.c	0	....	0.208	0.410	0.565	0.683	....	0.246	0.492	0.683	0.839
	5	0.953	0.204	0.388	0.531	0.646	0.0205	0.242	0.461	0.643	0.791
	10	0.0221	0.195	0.350	0.500	0.600	0.0294	0.222	0.422	0.606	0.738
	15	0.0343	0.189	0.338	0.458	0.558	0.0468	0.217	0.381	0.545	0.690
	20	0.0423	0.182	0.319	0.433	....	0.0574	0.212	0.378	0.521	0.643

In practical work most all strengths of solution are given in terms of percentages. To convert the values in the above table to these units the following table may be used.

Percentage-			Grams per 100 c.c.			Gram Equivalent per 1000 c.c
Copper	$CuSO_4$	$CuSO_4.5H_2O$	Copper	$CuSO_4$	$CuSO_4.5H_2O$	
1.00	2.5	3.91	1.04	2.56	4.0	0.321
2.00	5.0	7.82	2.09	5.25	8.2	0.658
3.99	10.0	15.64	4.41	11.05	17.4	1.387
5.99	15.0	23.46	6.97	17.50	27.4	2.194
6.98	17.5	27.37	8.37	22.00	32.9	2.631

% H_2SO_4	Grams per 100 c.c.	Grams Equivalent per 1000 c.c.
5	5.15	1.05
10	10.7	2.18
15	16.5	3.38
20	22.8	4.66

Measurements were also made of the temperature coefficient of the conductivity.

Of the general conclusions of the authors these are the most important:

Addition of copper sulphate to solutions of sulphuric acid increases the conductivity of the mixture if the sulphuric acid is less than 3 grams per 100 cc. On the other hand, it decreases the conductivity if the sulphuric acid is over 3 grams per 100 cc. If the sulphuric acid is just 3 grams per 100 cc. the addition of a little copper sulphate to the solution of sulphuric acid has no effect on the conductivity of the mixture.

The temperature coefficient of mixtures of equal parts of copper and sulphuric acid appears to be that of the sulphuric acid alone.

Transformation of Other Forms of Carbon into Graphite.

A paper by Mr. W. C. Arsem, of the Research Laboratory of the General Electric Company, Schenectady, N. Y., discusses the conditions under which "amorphous carbon" is transformed into graphite. These conditions have been the subject of much discussion, and many statements have found their way into literature which are not supported by experimental evidence.

The following theories are commonly held:

1. A high temperature alone will convert "amorphous carbon" into graphite (Moissan).

2. Pure carbon is not converted to graphite by heat alone (Berthelot).

3. Graphite is the result of intermediate formation and decomposition of carbides, due to the presence of mineral matter as an original constituent, or purposely added (Acheson). For instance, in one of Dr. Acheson's patents (617,979, Jan. 17, 1899)

it is mentioned that a mixture of 97 parts coke or charcoal and 3 per cent iron oxide can be changed to a greater or less extent into graphite by varying the time and temperature of heating. As the iron is insufficient to convert all the carbon to carbide, it is assumed to have a catalytic effect, first forming a carbide which decomposes, yielding graphite and setting free iron, which again forms carbide, and the cycle is repeated.

The present author has collected all available references relating to the problem. These are summed up in the paper.

He also emphasizes the ambiguity in the use of the term "graphite." The Brodie test is thought not to be decisive. The most characteristic property seems to be specific gravity. Mr. Arsem therefore proposes the following definition:

Graphite is that allotropic form of carbon having a specific gravity of 2.25 to 2.26.

Those varieties of carbon which have some of the physical properties of graphite, such as color, softness and streak, but a lower specific gravity, may perhaps be regarded as impure graphites; that is to say, mixtures of graphite with other forms of carbon.

Mr. Arsem then gives an account of an experimental investigation the object of which was to clear up these two points at issue:

1. Can a pure form of carbon be transformed into graphite by simply heating it to a high temperature?

2. If this is not the case, is it possible to cause this transformation to occur by heating the carbon, well mixed with a quantity of mineral matter, insufficient to form carbides with all the carbon present?

There are two ways of attacking the problem.

(a) To determine the effect of heating various forms of pure carbon alone and with small amounts of added mineral matter.

(b) To remove the mineral matter from certain varieties of impure carbon which ordinarily change to graphite when heated and see if the change will still occur.

Both methods were tried and the following results were obtained:

All the pure forms of carbon which have been tested, when fired above 3000°, reach a limiting density which is not appreciably raised by the addition of small amounts of mineral matter. The end product is graphite in some cases and not in others. Pure petroleum coke, heated without addition of mineral matter, is converted into graphite of excellent quality, while lampblack, although it increases in density, does not reach the value corresponding to graphite, nor acquire any of its other physical properties, even when heated with various oxides.

The impure carbons show a similar behavior. The properties of these carbons after firing are characteristic for each variety of carbon and independent of the amount of ash present.

Anthracite coal is only imperfectly graphitized by heating. The specific gravity of the fired material was approximately the same for three samples having a range of ash content. Moreover, coal from which most of the ash has been previously removed graphitizes better than the crude material.

Bituminous coal coke, which graphitizes fairly well, yields an even better product if a part of its ash has been removed before firing.

It must, therefore, be concluded that a small amount of mineral matter exercises no beneficial effect in the manufacture of graphite by the heating of carbon, and that the quality of the product cannot be improved in this way. As to the effect of mineral matter on the rate of conversion, no data are yet available, although some experimental work along this line has been started. At 3000° the maximum density is reached in less than 15 minutes for any variety of carbon.

Some theory of the nature of graphite and of amorphous carbon is needed which will permit a rational explanation of the changes which occur on heating. Mr. Arsem offers the following discussion as a tentative step in this direction:

Graphite, in the most restricted sense of the term, is an allotropic form of carbon having a definite and not very complex molecular configuration. The molecule might, for example, be regarded as two benzene rings side by side and joined at all six angles, the extra bonds being satisfied as in the usual centric formula. Such a formula might be used to explain the formation of mellitic acid and graphitic acid by oxidation of graphite.

Amorphous Carbon. When an organic compound is decomposed, there results a mixture of substances constantly increasing in complexity until finally carbon is obtained. This carbon need not be regarded as a simple substance, but may be considered to be a mixture of many varieties of carbon, each with a different number and arrangement of atoms in the molecule. According to this view, it seems better not to regard "amorphous carbon" as the name of a distinct allotropic form, but rather as a general term covering all varieties of carbon except graphite and diamond.

In speaking of amorphous carbon, therefore, the source of each sample should be stated.

The Mode of Transformation. In a given sample of amorphous carbon, some of the molecules will be capable of easily undergoing rearrangement under the influence of heat to form graphite molecules, while others will not, and the proportion of molecules capable of such change will determine the character of the final product.

Petroleum coke would therefore consist almost wholly of graphitizable molecules, while anthracite coal contains a smaller proportion.

Mr. Arsem's paper which had been read in the absence of the author by Mr. Hansen was briefly discussed by Mr. Lidbury and Mr. Hansen.

Electric Laboratory Furnace with Resistor of Ductile Tungsten or Molybdenum.

A paper by Messrs R. Winne and C. Dantsizen of the Research Laboratory of the General Electric Company, of Schenectady, N. Y., describes two simple electric furnaces for laboratory work with resistors of ductile tungsten or molybdenum wire or ribbon. On account of their very high melting points and relatively low cost these metals are well adapted for that purpose.

The first type is a **crucible furnace** and consists of a helix of ductile tungsten or molybdenum, supported by an alundum tube and protected from oxidation by a hydrogen atmosphere.

Fig. 1 shows the furnace in vertical section. The heating wire, of ductile tungsten or molybdenum, is wound on *G*, an alundum cylinder which is molded plain inside and with a helical groove on its outer surface. *A* is a Battersea crucible; *B* is an inverted Battersea of which the bottom has been cut off; *C* is the crucible bottom, used as a cover. The bottom of *A*, as well as the space between *A* and *B*, is filled with powdered silica. Hydrogen is constantly supplied to the furnace through the pipe *D*, and burns as it escapes from the top of *B*. This gives a protecting atmosphere for both the tungsten or molybdenum winding and the object to be heated. Current is supplied to the winding through two large copper connectors *E*.

An object to be merely heated, as for annealing, may be inserted directly into the alundum cylinder. Material to be melted is placed in a small crucible, preferably made of pure alumina, and this is let down into the alundum cylinder.

In the apparatus shown in Fig. 1, the Battersea crucibles *A* and *B* are sizes *O* and *J* respectively. The alundum body *G* (made by the Norton Company, of Worcester, Mass.) consists of two cylinders placed end to end. Each is 7.6 cm. high and 5.1 cm. inside diameter, while the distance from one convolution of the helix to the next is .95 cm. The winding consists of 260 cm. of square molybdenum wire, 1.27 mm. on a side. This wire is first wound on a mandrel 3.8 cm. in diameter, and wire and mandrel are then heated up to about 800 deg. C. Upon then releasing the coil, it expands to the diameter of the alundum body, on which it can then be screwed. The copper connectors

E are 0.8 cm. in diameter, and the ends of the coil are simply clamped in with set-screws. The copper connectors and the hydrogen inlet tube are held in place in the crucible wall by a mixture of powdered silica and water glass, which at the same time prevents loss of hydrogen.

This furnace can be safely run up to 1700 deg. C. At this temperature it calls for 25 volts and 45 amp.

Pure iron can be melted and ferro alloys and other alloys can be produced in this furnace.

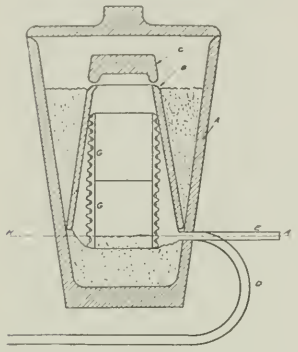


FIG. 1.—CRUCIBLE FURNACE WITH TUNGSTEN WIRE RESISTOR.

The second type is a **tube furnace** and consists of a porcelain or alundum tube, wound with tungsten or molybdenum foil. Around the tube is a tight metal casing filled with powdered silica, and, to prevent oxidation of the winding, a hydrogen atmosphere is maintained in the casing. Fig. 2 shows the furnace in vertical length section. *A* is an alundum tube, 2.3 cm. inside diameter and 46 cm. long. The casing *D* is made of thin sheet iron and is oxy-acetylene welded. The winding *B* is a molybdenum ribbon, 0.184 mm. thick, 2.54 mm. wide and 445 cm. long. *I*, *I* are heavy copper leads, fastened to the ends of the coil by twisting the latter about them. *C* is the powdered silica packing, *F*, *F* is some asbestos wool, used to prevent the escape of hydrogen at the ends of the casing.

H and *G* are the hydrogen inlet and outlet tubes respectively, *E* is a rubber stopper.

With this furnace a temperature of 1600 deg. C is readily attained. At this temperature it calls for 80 volts and 14.3 amp. The two furnaces described are adapted to heating the charge in an atmosphere of hydrogen. In case it were desirable to heat

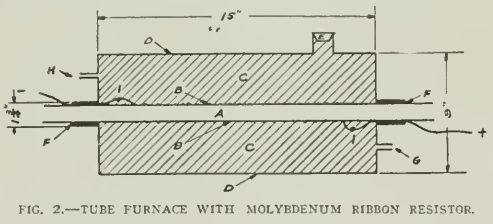


FIG. 2.—TUBE FURNACE WITH MOLYBDENUM RIBBON RESISTOR.

in an oxidizing atmosphere, it would be necessary to replace the very porous alundum by porcelain, as otherwise the furnace winding would become oxidized.

In the discussion of the paper of Messrs. Winne and Dantsizan in which various speakers participated, Mr. Saunders suggested that the porosity of alundum tubes could be overcome and that their field of application could thereby be extended by providing a glazed inner surface. The melting point is high enough.

Electric Properties of Sodium, Potassium and Their Alloy

A paper by Dr. Edwin F. Northrup, of Princeton University, gives new data on some electrical properties of sodium, potassium, and their alloy. The author intends to determine the electric resistivities and temperature coefficients of as many metals as possible at higher temperatures. To carry out the investigation in a systematic manner, he will take up the elements in the order of Mendelseff's table. The investigation began with soduim and potassium, which fall in the first group. In view of their low melting points it seemed sufficient to limit the investigation to the temperature range between 0 deg. and 150 deg. C.

A glass tube, supplied with platinum potential and current terminals, was first filled with pure mercury, and the resistance of this, throughout the range of temperature employed, was determined between potential points. The same tube was then filled with the metal, in a molten state, which was to be measured. Its resistance, between the same potential points, was then measured at various temperatures as the metal cooled, from the highest to the lowest temperature. From the data so obtained the specific resistances and temperature coefficients were determined.

The methods of measurements are described in some detail.

For the resistance of the mercury in the interval from 20 to 110 deg. C the following relation giving the relation between resistance and temperature was found.

$$Rt = 0.02889\ (1 + 0.0010036t)$$

where t is degrees C above zero.

Substance	Spec. Res. at 20° C. $10^{-9}\times$	Spec. Res. at 100° $10^{-9}\times$	Temp. Coef. Low Range	Temp Coef. High Range	Density at 0°
Mercury	95,810	103350	0.001003 (20° to 100°)		13.595
Sodium	4,875	9,705	0.0053 + (20° to 80°)	0.00494 + (100° to 140°)	0.978
Potassium	7,118	15,275	0.0058 (20° to 50°)	0.00507 + (66° to 120°)	0.87
Alloy, K + Na	33,800	37,795		0.00147 [46° to 125°)	0.92
Copper	1,721	2,266	0.00387 +		8.89

Spec. res. of Na at 20° C. = 0.05088 × Spec. res. of Hg at 20° C.
" " " Na " 100° C. = 0.09390 × " " " " " 100° C.
" " " K " 20° C. = 0.07429 × " " " " " 20° C.
" " " K " 100° C. = 0.1478 × " " " " " 100° C.
" " " Na + K " 20° C. = 0.3528 × " " " " " 20° C.
" " " Na + K " 100° C. = 0.3657 × " " " " " 100° C.

Some figures in this table are interesting. First the very high coefficient of potassium in the range 20 deg to 50 deg. of 0.0058 and the slightly smaller coefficient of sodium in the range 20 deg to 80 deg. of 0.0053. Potassium is thus probably unique in possessing the highest coefficient of any pure metal in this range that has been observed. It should be noted that the coefficient of the alloy of potassium and sodium, instead of being approximately the mean of the coefficients of sodium and potassium when liquid, is only between a third and a fourth of the mean value of the coefficients of its constituents in the same temperature range.

The mean specific resistance at 100 deg. of sodium and potassium is $12{,}490 \times 10^{-9}$, while the specific resistance of the alloy, in equal proportions by volume, is $37{,}795 \times 10^{-9}$, or slightly over three times as great as the mean specific resistance of its constituents. This is an interesting physical fact which may help to throw light upon the theory of conduction in metals.

In this connection we may note that alloying metals generally, if not always, increases their specific resistance and lowers their melting point.

The abrupt increase in resistance of the metals experimented with, expressed in percentage, is as follows:

Sodium, in passing from the solid state at 93.5 deg. to the liquid state at 96.3 deg., increased 44.8 per cent.

Potassium, in passing from the solid state at 60.4 deg. to the liquid state at 63.2 deg., increased 52.2 per cent.

The alloy of sodium and potassium, in passing from 2.8 deg., at which temperature it may probably be considered as completely solid, to 20 deg. C., at which temperature it may probably be considered as completely liquid, increased in resistance 48.5 per cent.

The ratio of the resistivity at 20 deg. C. of sodium to copper is 3.058, but the density of copper is almost exactly nine times that of sodium, which, in round numbers, makes sodium three times as good a conductor per unit of weight as copper. Hence it could compete with copper as a conveyor of electric power if it cost anything less than 3 times as much as copper per pound, provided, of course, current-carrying capacity were the only consideration. It has been suggested before now that sodium might be used in iron pipes in a practical way for carrying electric power. Dr. Northrup suggests that inasmuch as electrochemical processes often necessitate the carrying over short distances of enormous currents, the consideration of the proposal might be with advantage taken up afresh.

The paper was discussed by Messrs. Morrison, Whitney, Hering, and Lloyd. Reference was made to the considerable change in electric conductivity of steel at the melting point.

Electric Resistivity of Iron Alloys.

A paper by Prof. Chas. F. Burgess and Mr. James Aston, of the University of Wisconsin, discusses the electric resistivity of various iron alloys. In the design of electrical machinery, the electrical resistivity of the iron used for electromagnets is of an importance second only to the magnetic permeability, particularly for alternating-current working. A high resistance is desired, since it will reduce to a minimum the eddy currents sets up by the reversals of the magnetic flux. It is

TABLE I.—HIGH ELECTRICAL RESISTANCE. RELATIVE ABOVE 5.

Composition	Microhms per cm.*	Relative Resistivity	Condition
10 Cr 5 Si	103.0	8.50	Forged
10 Cr 12 Ni	99.6	8.23	"
10 Cr 20 Ni 2 Si	93.0	7.68	"
10 Cr 5 W 2 Si	92.2	7.61	"
15 Cr 2 Si	92.6	7.55	"
7.23 Cr 9 W 3 Si	89.5	7.39	"
10 Cr 3 Si	87.5	7.23	"
10 Cr 2 Si 2 Ti	86.0	7.11	Quenched
13 Cr 1.8 Si	85.6	7.08	"
10.3 Cr 1 V 2 Si 0.3 C	84.9	7.01	"
10 Cr 2 W 2 Mo 2 Si 0.3 V	83.8	6.93	"
21.4 Cr 3 Mn 1 Si 1 V	82.0	6.77	"
28.42 Ni	82.0	6.77	"
35.09 Ni	81.1	6.70	"
20 Ni 10 Mo 1 C	79.6	6.57	Forged
10 Cr 10 Ni 2 Si	78.8	6.51	"
2 Si 10 V	78.9	6.51	"
5 Cr 6 Mo 10 Mn 10 Ni 1 C	78.0	6.45	Quenched
10 Cr 2 Si 0.3 V 0.6 C	77.4	6.40	"
6.2 Si	77.2	6.37	Forged
20 Ni 10 Mo	76.9	6.36	Quenched
17 Cr 1½ Si	76.0	6.28	"
20 Cr 10 Ni 1 Si 0.3 V	76.0	6.28	Forged
10 Cr 2 Si	75.0	6.20	"
10 Cr 1 V	74.8	6.18	Quenched
10 Cr 10 Ni 1 Si	74.2	6.13	Forged
10 Cr 10 Ni 1 C	73.8	6.10	"
10 Cr 2 W 2 Si	72.8	6.02	Quenched
10 Ni 2 Si 1 C	70.3	5.81	Forged
2 Ni 12 Mn 6 Mo 0.8 Si	68.5	5.66	"
7 Cr 10 Ni	67.3	5.65	Quenched
10 Cr 8 Ni	68.4	5.65	Forged
20 Cr 3 Mn 1 Si	67.1	5.55	Quenched
10 Cr 10 Ni	66.9	5.44	Forged
26.40 Ni	65.5	5.41	Quenched
6 Cr 1.68 Si	64.4	5.32	"
25.20 Ni	64.2	5.30	Forged
25.20 Ni	63.2	5.22	Quenched
4.65 Si	62.2	5.13	"
10 Cr 2 W 2 Mo 0.6 C	61.6	5.09	Forged
8 Cr 1 Si	61.2	5.06	Quenched
10 Cr 10 W	60.7	5.02	Forged
10 Cr 2 W 0.3 V 0.6 C	60.4	5.00	Quenched

also of interest to know the conductivity of the various iron alloys, regardless of their magnetic properties, because of their possible utility in rheostats, resistance heating wires and the like.

In connection with the series of tests which the authors carried out on the various properties of electrolytic iron and its alloys, they determined the electrical resistance of the numerous bars used in the magnetic tests. They started with electrolytic iron as base material (the resistivity of which is taken as unity) and made up the various alloys by addition of the other elements.

It appears as a result of these tests that all three of the elements of arsenic, silicon, and tin, which are beneficial to the magnetic properties of iron, have the desirable accompaniment of high electrical resistivity.

Among the materials adapted for high resistances, various combinations of iron with nickel, chromium, and silicon are most suitable.

The change of resistance due to heat treatment in general follows the changes of physical hardness due to the treatment. Annealing usually results in a decrease of resistance. Quenching is uncertain in its effect; usually there is an increase of resistance with physical hardening of the bar.

There are grouped in Table I, in the order of their relative resistances, all those iron alloys tested which showed a relative value as compared with the electrolytic iron of *five or upwards*. (The electric resistivity of pure electrolytic iron being taken as unity.) The highest resistance recorded is 8.50, reached by a bar with 10 per cent Cr, 5 per cent Si. Examination of the table shows 43 bars listed, all of which contain chromium, nickel or silicon, either alone or in various combinations.

The paper was briefly discussed by Messrs. Hansen, Lloyd, Northrup, and Watts. It was stated that the determinations of resistivities had been made at ordinary temperatures.

Before the conclusion of the Thursday afternoon session Mr. F. A. J. FitzGerald read the report of a special committee which had been appointed by the president to consider the possibility of setting a higher standard for the papers published in the Transactions of the Society. The report had been very carefully prepared and contained a series of valuable suggestions which had been approved by the meeting of the Board of Directors the night before. A committee was appointed to carry the suggestions of the report into effect.

Mr. Edward R. Taylor read a set of recommendations on principles of water conservation which he suggested to present at the coming National Conservation Congress in behalf of the Society. These recommendations were approved and endorsed.

Electrochemical and Electrometallurgical Developments in Canada.

The first paper of the Friday session was presented by Dr. S. Dushman, of the University of Toronto.

He first gave a review of the electrochemical plants at present in operation throughout the Dominion of Canada. Their number is at present not very large.

In the extreme west in the Province of British Columbia, there is in Trail the electrolytic lead refining plant (using the Betts process) of the Consolidated Mining and Smelting Company of Canada.

The oldest electrochemical industry in Ontario is the manufacture of calcium carbide at the plant of the Willson Carbide Company at Merriton. Carbide is also made by the Ottawa Carbide Company on Victoria Island, in that section of Ottawa known as the Chaudière.

In the old Heroult electric furnace plant at Sault Ste. Marie, in which Dr. Haanel made his well known tests for the Government on the reduction of iron ore in the electric furnace, and which was later taken over by the Lake Superior Power Company for the manufacture of nickel pig from pyrrhotite, the company is now making ferrosilicon for its own consumption.

Ferrosilicon is also made by the Electro Metals Company, at Welland.

On the Canadian side of Niagara Falls there are plants for making cyanamide, graphite, and carborundum. The American Cyanamide Company commenced operations in January, 1910, with a 10,000-ton plant and is at present producing to its full capacity. The whole output is shipped to the United States.

The Norton Company has a plant at Chippewa for the manufacture of crystolon—another trade name for carborundum or silicon carbide. The raw materials used are two different

grades of coke (a metallurgical coke with 92 per cent or more fixed carbon and 5 per cent ash, and a petroleum coke, containing about 91 per cent fixed carbon with less than 1 per cent ash), a very pure silica sand, and sawdust. Two grades of product are manufactured, green and steel gray. At present 2000 hp are employed. The whole output of the plant, which is 5 tons per day, is shipped to the company's main works at Worcester, Mass., where it is used for the manufacture of different abrasive articles.

The International Acheson Graphite Company has also established a plant near Niagara Falls, Canada. Its output in 1909 was 513,430 pounds of artificial graphite.

No electrolytic alkali plant is at present in operation in Canada. The old Rhodian mercury-cell plant at Sault Ste Marie was discontinued years ago and has not been opened again.

At the present time the Canadian Salt Company, of Windsor, Ont., is erecting a plant at Sandwich for the manufacture of caustic soda electrolytically, the Gibbs cell being used. It is expected that this plant will begin operations very shortly.

There is an electrolytic gold and silver parting plant at the Royal Mint in Ottawa. The gold bullion (after being assayed) is melted with silver in such proportion as to form an alloy containing 40 per cent gold, 56 per cent silver, and 4 per cent base metals. The alloy is cast into anode plates and used in a dilute solution of silver nitrate and a little nitric acid, and the silver deposited out. When nearly all the silver is removed a spongy mass of gold of the original shape of the anode plate is left. This is washed and re-melted and treated as anodes by the Wohlwill gold refining process.

Most of the electrochemical industries of Quebec are situated around Shawinigan Falls. The Northern Aluminum Company, a branch of the Aluminum Company of America, produces aluminium by the Hall process. The total capacity of the works is 25 tons of aluminium per day, about 500 men being employed when the plant is running at full capacity. The output in 1909 was over 3100 tons.

Carbide is made by the Shawinigan Carbide Company, by a continuous process.

The Electro Reduction Company, at Buckingham, manufactures phosphorus and ferro-phosphorus from apatite.

In the second part of his paper Dr. Dushman gave an account of the possibilities of future electrochemical and electrometallurgical developments in Canada, discussing the available raw materials, water powers, and transportation facilities.

As to mineral resources the greater portion of Canada is as yet unprospected, about one-third lies within the prospected area and two-thirds without, and the mineral resources have been developed (leaving out the Yukon) practically only in the territory lying fairly near the southern boundary. Even the portions of the country represented as being within the prospected territory must not be considered more than partially explored for minerals, as new discoveries are continuously being made within the supposed prospected areas.

At the present time the mining industry of Canada is second only to agriculture. In 1886 the total mineral production was slightly over $10,000,000; in 1909 it had grown to $92,000,000 and in 1910 to over $105,000,000. In 1909 the total mineral production of Canada had a value of $105,000,000. Of this $40,000,000 was the value of the metallic products including $17,000,000 for silver, $11,000,000 for nickel, $10,000,000 for gold and $7,000,000 for copper. Among the non-metallic products coal leads with $30,000,000.

The gold resources of Canada have only been skimmed so far. About 90 per cent of the silver produced in Canada in 1910 was derived from the rich silver deposits of Cobalt.

The ores of the Cobalt district are extremely complex, and the problem of treating these ores economically is still awaiting a solution. Under present conditions one of the pressing needs for the further promoting of mining and metallurgical activity in Northern Ontario is the devising of suitable methods for the treatment of the Cobalt silver ores. Only about 30 per cent of the ore shipped from Cobalt in 1909 was treated in metallurgical works in Canada, most of this being high-grade silver ores, and the rest was shipped to a number of large companies operating in the United States. With respect to the content of nickel, cobalt and arsenic in the ores, the mining companies are paid only for a small portion of the cobalt content, and nothing for the nickel and arsenic, in fact, in certain cases the latter two are penalized.

Another interesting problem in connection with the cobalt silver ores, is what to do with the cobalt.

In the solution of these problems there is undoubtedly a field for the electrochemist.

With respect to iron much attention has been paid in Canada to the possibility of using electric furnace processes since Canada has immense iron ore deposits and water powers. What has held back this development in the past is the very large capital investment which would be necessary.

Electric furnace experiments have also been made on a larger scale with respect to zinc in Canada, but have not led to commercial operation.

As to water powers in Canada, the total amount of power already developed and in process of development in British Columbia is 75,000 hp. The numerous powers off the coast would aggregate another 50,000 hp.

In Ontario the sources of water power are almost innumerable.

The Ottawa River alone would be capable of supplying over 300,000 hp, while the numerous falls on the Gatineau can supply over 200,000 hp.

In the province of Quebec the numerous tributaries of the St. Lawrence present remarkable power possibilities.

In New Brunswick only the St. John river has been exploited.

The Grand Forks Power Company is building a plant at that place to develop 80,000 hp.

With respect to transportation facilities, Canada in 1867, had 2278 miles of railroad, while it has at present close to 25,000 miles. The Canadian Pacific Railway was only completed in 1886, while there are now two other transcontinental railways in Canada.

The author concluded with the remark that in the great evolution which Canada is to undergo during this century the electrochemical engineer will be called upon to solve some of the most important problems.

Metallic Cerium.

A long paper by Dr. Alcan Hirsch on "The Preparation and Properties of Metallic Cerium" gives an account of a very extended and elaborate investigation which required over three years' work at the University of Wisconsin.

The investigation was undertaken with a two-fold purpose. First, starting from as pure cerium salts as it was possible to obtain, it was desired to prepare the metal in quantities sufficiently large that its physical and chemical properties could be studied and a large number of alloys could be made. Second, by preparing metal from the unpurified rare-earth residues from monazite sand, obtained as a by-product in the incandescent gas mantle industry, it was hoped that alloys of commercial value could be obtained. In other words, whereas the application of these residues as oxalates or oxides is very limited to-day, their value might possibly be increased materially if a useful metallic or alloyed product be prepared from them.

A historical review is given of former work done on metallic cerium and then the methods developed for the preparation of the starting material are discussed in some detail. The general scheme followed was to use the mixed rare-earth oxides for experimenting on the exact conditions required for the preparation of the desired salt, and after the proper con-

ditions of the process had been determined, to prepare the pure cerium salt. Oxalates of the rare-earths, a by-product of the commercial extraction of thorium and cerium from monazite sand, were employed for these experiments. The oxides prepared by calcining the oxalates at a temperature of 750-800 deg. C. in a gas muffle furnace, analyzed 97.8 per cent. rare-earth oxides, of which approximately 49 per cent. was cerium dioxide.

The first salts required were the **anhydrous chlorides** (composition RCl_3). There are several hydrated chlorides of cerium known. The chief difficulty is in their dehydration since cerium chloride is similar to magnesium chloride in that when its solution is evaporated to dryness and calcined, HCl is evolved with partial decomposition of the chloride to oxychloride and oxide.

The author discusses a good many methods for the dehydration of chlorides of cerium. One method of obviating the difficulty would be to use a non-aqueous solvent. The carbide RC_2 may be prepared by heating the mixed oxides with powdered graphite in an arc furnace. Alcoholic HCl, made by passing dry HCl gas into cooled absolute alcohol, acts readily on the carbides. But the solution of the chlorides prepared by this method always contains more or less graphite, and is very difficult to filter because of its viscosity. The method is in general unsuitable for large-scale operation.

Resort was, therefore, again made to dehydration methods. It was found that by treating the mixed oxides with concentrated HCl, evaporating to dryness, adding more HCl and repeating the process a number of times, a product was obtained corresponding to one of the lower hydrates of the chlorides. But the difficulty is to get rid of the last molecule of water. In this process certain chlorides of the metalloids, especially those of phosphorus and sulphur, can be used as catalysers in the dehydration of hydrated chlorides. Experiments in this direction are described and finally a method of dehydration which may be carried out by HCl gas alone (without phosphorus and sulphur chlorides) was evolved which was used in the research. This is as follows.

The double cerium-ammonium nitrate from the fractionating vats, where it has been purified and separated from the other rare-earths by fractional crystallization, was used. The absorption spectrum of a concentrated solution of the nitrates showed only the faintest traces of the neodymium and praseodymium bands. The contents of a large bottle containing five kilograms of the double nitrates was divided into four portions and placed in fourteen-inch (35 cm.) porcelain evaporating dishes. The dishes were filled with concentrated HCl and placed on steam-heated sand-baths. Each dish was suitably protected from drippings from the hood by wooden covers raised a few inches above the tops of the dishes and supported on silicia bricks.

Fresh acid was added daily as required and the evaporation continued for about three weeks, at the end of which time the nitrates had been completely converted into chlorides and all the ammonium salts had been expelled. The white solid was broken into lumps and heated on an asbestos-covered hot plate to convert it to the monohydrated chloride.

The final dehydration was carried out as follows (see Fig. 1). About 400-500 grams were placed in a ten-inch porcelain evaporating dish. A short-stem funnel about nine inches in diameter was inverted and placed over the salt. A layer of asbestos fiber was packed tightly around the outside edge of the funnel and plaster of paris set over this. A glass tube passing through the steam of the funnel and reaching to just above the layer of chloride conducted in the dry HCl gas, which was passed at a rapid rate over the heated chloride. The salt was stirred around from time to time by means of a long, thick glass rod. The heating was done gradually, and the progress of the dehydration could be watched through the glass funnel and its completion checked nicely.

The anhydrous chloride was placed, while still warm, in dry glass vessels, and the corks paraffined in to exclude moisture from the extremely hygroscopic product.

The chloride prepared by this method melts to a clear, limpid liquid, and is suitable for electrolysis.

It was observed that the anhydrous rare-earth chlorides may be partially converted into oxides or oxychlorides if they were carelessly heated in the presence of air. The best method of melting the anhydrous chlorides without decomposition was found to be by heating a small portion in a covered crucible until a clear melt was obtained, and then introducing successive small portions of the chlorides to the melted portion. In this manner circulation of air is avoided, as the crucible is kept covered. A good way is to first melt sodium or potassium chlorides, or a mixture of these, and then to add small amounts of the rare-earth chlorides, keeping the crucible covered as much as possible. A small amount of oxide or oxychloride will "gum up" the entire melt, and for electrolysis this condition must be carefully guarded against.

Another salt required in large amounts was the **anhydrous mixed fluorides**, RF_3. Lead lined apparatus are needed in the preparation.

After discussing several methods and their preparation the following method is given as the most satisfactory: About a kilogram of the mixed oxides was placed in a large porcelain evaporating dish and digested in a steam closet with concentrated HCl until a clear concentrated solution of the chlorides was obtained. The hot solution, containing only a slight excess of hydrochloric acid, was transferred to a five-gallon lead jar and concentrated HF added to complete precipitation. The fluorides were allowed to settle somewhat, as much of liquid syphoned off as was possible, and washed twice with hot water and several times with 95 per cent. alcohol. Only a few minutes' time was allowed after each washing for the precipitate to settle.

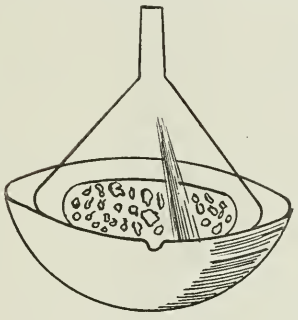


FIG. 1—APPARATUS FOR FINAL DEHYDRATION.

The fluorides were then placed in a lead-lined cage-centrifuge, and most of the remaining alcohol removed in this manner. Absolute alcohol was added and the solution evaporated to dryness without decanting. The absolute alcohol dehydrates the centrifuged fluorides very well. They were then dried at 100°, and the temperature raised to 200° at the end. About six kilograms of the fluorides were prepared in this manner.

Preparation of Metallic Cerium.—Two methods were tried: first thermal reductions and second electrolysis.

Thermal Reductions.—Magnesium and calcium shavings and aluminum powder were used as reducing agents on the mixed oxides. These experiments were conducted in electrically-baked magnesia linings placed in graphite containers. Carbon and silicon were also used to reduce the oxides, but external heat was applied to give the high temperature required. No metal was formed in these last instances but the carbides and silicides respectively. The carbide RC_2 is pyrophoric, but the silicide RSi_2 is not. Reductions using magnesium and calcium in excess gave alloys, many of which were pyrophoric.

In the case of these thermal reductions, either an alloy, a compound or a lower oxide of the rare-earths was obtained. Owing to the high heat of formation of these oxides, the failure of alumino-thermics in the production of metals was only to be expected. The author believes that a study of the

equilibrium conditions of these reactions would form an interesting and valuable research and might result in the discovery of lower rare-earth oxides.

Preparation of Cerium of Electrolysis.—Electrolysis of the anhydrous chlorides was first tried but was found to involve a great number of difficulties which are recorded in some detail.

Electrolysis of anhydrous fluorides was then taken up. The anhydrous mixed fluorides melt at a temperature of a little over 900 deg. C, and dissolve the mixed oxides with avidity. The electrical conductivity of the molten fluorides is low, but increases rapidly with addition of oxides. The solution containing about 20 per cent. of oxides conducts very well.

Several runs of electrolysis are described. The principal difficulties encountered in the electrolysis of the oxides dissolved in the fused fluorides were:

1. Oxides must be added to the fluoride to make a sufficiently conductive solution, and the melting-point of the bath is very high.

2. At the high temperature of the bath, rare-earth metal and graphite combine to form carbide.

One of the principal difficulties in all previous electrolyses was the formation of carbides, which made the bath viscous and unfit for electrolysis. It was thought that if a non-carbon cell was used, the formation of carbide would be less likely to occur, as the only carbon in the cell would be the anode. On account of their corrosive action and high melting point the fluorides could not well be used in other than carbon vessels. It was, therefore, decided to try the electrolysis in a wrought-iron vessel.

Several experiments are described which show (1) that rare-earth metal could be produced by electrolysis of the chlorides in iron vessels; (2) that the use of an insulated cathode of graphite gave a lower current efficiency and resulted in some of the metal being converted to carbide; (3) that it was necessary to regulate the temperature of the bath so as to have the electrolyte well fused and yet well below the alloying temperature of iron and rare-earth metal; and (4) that control of the electrolysis could be maintained by judicious addition of small amounts of potassium fluoride, sodium chloride, and barium chloride.

The experiments in which the anhydrous mixed chlorides were used, were so successful that the method of electrolysis was applied to pure anhydrous cerous chloride. An account is given of several runs made in this way. A difficulty encountered was the removal of the material and electrolyte at the end of the run. This difficulty was overcome by using a vessel of the following construction. (Fig. 2.)

It consisted of a 3-in. (8 cm.) iron pipe screwed tightly into a 5-in. (13 cm.) iron flange, to which a bottom plate was attached by means of four large bolts and nuts. The electrolyte was cerous chloride containing a small percentage of sodium chloride, to which very small amounts of potassium fluoride and potassium fluoride-barium chloride mixture were added when required (about every half hour or so). The function of these additional salts has already been explained in detail.

The voltage was kept at its proper value, which should not be greater than 15 volts. The average current used was 200 amperes, for three hours. The electrolysis required attention about every twenty minutes; that is, the anode needed slight adjusting or the temperature had to be corrected. An ingot weighing 380 grams was obtained. The number of ampere hours was 600, and the ampere efficiency 36.5 per cent.

The electrolyte remained liquid during the entire course of the electrolysis. About 1½ kilograms of cerous chloride was used. The cell held perfectly, and at the end of the run the electrolyte and metal were easily removed. This type of electrolytic vessel is highly recommended for similar work. Care should be taken that the flange and bottom plate are well faced and that the lap is wide enough (at least 1½ in. (4 cm.))

Two methods of analysis of cerium are then described, one being gravimetric, the other volumetric.

The chief impurities of cerium (amounting in the aggregate to about two per cent.) obtained by the method previously described consisted of iron, cerium oxide and cerium carbide, of which over one per cent. was iron.

The best method of **purification of cerium** consists in preparing cerium amalgams. The amalgams are prepared by boiling mercury with cerium in a long iron pipe arranged with a condensing tube at the top. Solid amalgams are easily obtained by this method. The iron and impurities float to the top and may be skimmed off. The amalgams carefully prepared give only a very slight test for iron. The cerium and mercury may be separated by distilling the latter, but this must be done in a high vacuum to prevent oxidation of the cerium. A high temperature is required to drive off all the mercury. The amalgam should be placed in a magnesia vessel, and this in turn heated in an evacuated quartz vessel. The high temperature required would cause the collapse of an evacuated glass vessel.

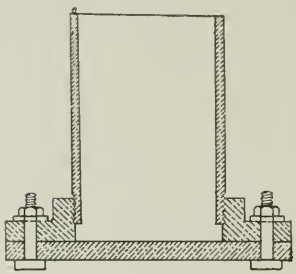


FIG. 2.—ELECTROLYTIC CELL.

Physical Properties of Metallic Cerium.—In the determinations of the physical properties, metal of approximately 98 per cent. of cerium was used, unless otherwise stated.

Atomic weight 140.25.

Specific gravity 6.92 at 25 deg. C.

Hardness. For the determination of hardness, the scleroscope was used. The hardness of cerium varies, depending upon whether the surface of the metal is rolled, freshly cut, or old. The average value of the hardness for rolled surface was 25.9. The average for freshly cut surface was 9.5. The reading 100 is the hardness of the standard steel test plate.

Tenacity. Cerium is malleable and highly ductile. A strip of cerium was rolled to a thin sheet of a thickness of 0.015 mm. Cerium metal can easily be cut with a knife or scissors, and can be machined fairly well, although there is some tendency for the metal to buckle, as does lead.

The specific electric resistance of a sample containing 93.6 per cent. cerium and 4.5 iron was 71.6 microhms per cm. cube.

Magnetic properties. Pure cerium is weakly but definitely paramagnetic.

Melting point 635 deg. C.

The ultimate strength was determined in an Olsen testing machine. The average diameter of the test bars was 0.212 in. (0.55 cm.) There was no elongation with constant load at 350 pounds (159 kg.) The test bar broke with a snap (like cast iron) at 495 pounds (225 kg.). The ultimate strength is equal to 12,900 pounds per square in. (9 kg. per sq. mm.).

The specific heat of cerium (20 to 100° C.) was found to be 0.0524. The specific heating of copper is 0.0925.

The heat conductivity seems to be fairly high and, if so, cerium is probably one of the exceptional metals which has a low electrical conductivity and high thermal conductivity.

The latent heat of fusion seems to be fairly high.

Heat of oxidation. The average value of the heat of oxidation was 1,740 calories per gram, or 60,900 calories per gram equivalent ($CeO_2 = 243.600$).

Single potential. As the single potential of cerium could not be measured successfully, it was calculated and the single potential of cerium in normal solution of its chloride measured against the normal calomel electrode was thus found to be − 3.16 volts.

Thermo-electromotive force. With a hot junction at 100° C and a cold junction at 30 deg., the thermo-emf was 0.000250 volt. With the hot junction at 200.5 deg. and the cold junction at 30 deg. C. the thermo-emf was 0.000502 volt. The current flowed at the hot junction from copper to cerium.

Optical properties of magnesium cerium alloys were also determined.

Chemical properties of metallic cerium.—Cerium is very slightly attacked by cold water, but in boiling water a slow evolution of hydrogen gas occurs and the metal is tarnished black. At room temperature the following solvents have no action on cerium: Ethyl alcohol, amyl alcohol, chloroform, carbon tetrachloride, concentrated sulphuric acid, concentrated ammonium hydroxide, concentrated sodium hydroxide. Ethyl ether has a very slight action on the metal, as have also 3 per cent and 30 per cent hydrogen peroxide at room temperature. The action of dilute sulphuric acid, concentrated and dilute hydrochloric acid, concentrated and dilute nitric acid, in the cold, is moderately vigorous. Ammonium chloride and potassium chloride, at room temperature, have moderate action on the metal. This may be explained by the fact that in water a small amount of cerium hydroxide forms which is soluble in potassium chloride and ammonium chloride, thus causing fresh surface of the metal to be exposed and resulting in moderate attack of the metal by the solvent.

At boiling temperature the following solvents did not attack the metal. Chloroform, carbon tetrachloride, concentrated sulphuric acid, concentrated ammonium hydroxide. The metal was only slightly attacked by the following solvents at boiling temperature: Ethyl alcohol, amyl alcohol, ethyl ether and concentrated sodium hydroxide. Dilute nitric acid, ammonium chloride, potassium chloride and 3 per cent. hydrogen peroxide at boiling temperatures gave moderate action. Dilute sulphuric acid, concentrated and dilute hydrochloric acid, concentrated nitric acid and 30 per cent. hydrogen peroxide gave vigorous evolution of gas at boiling temperature.

The action of various gases on cerium was studied. A small amount of cerium filings was placed in a porcelain boat in a hard glass tube. The air was displaced by the gas under test, and the tube heated gently by means of gas burners. The cerium filings heated in chlorine emitted a very bright light at a temperature of 210° to 215° C. The salt formed was tested and found to be anhydrous cerous chloride.

In bromine, cerium burned at a temperature of 215° to 220° C. with emission of less light than in the case of chlorine. The salt formed in solution in water, and was identified as the bromide.

In iodine vapors no light was emitted, although the tube was heated to a temperature of 300° C. On examination of the contents of the tube it was found that some iodide had formed.

Cerium burned with luminescence when heated in air to a temperature of 160° C. If a lump of cerium is sealed in a glass bottle and kept warm for some time, a black powder is seen to form upon the surface of the cerium, and when the bottle is opened this powder ignites at room temperature. This is probably due to the formation of a highly pyrophoric suboxide of cerium.

When cerium filings are heated to 345° in hydrogen gas, the hydride forms without emission of light. The hydride has been described and studied by Muthmann. (*Liebig Ann.*, 325, 281.)

Cerium filings were heated in nitrogen to a temperature of 1000° C. No luminescence was observed, but some nitride was formed. This nitride is a black powder, and has been studied and described by Matignon. (*Comptes Rendus*, 131, (21), 837).

When the nitride is heated in potassium hydroxide solution, evolution of ammonia gas occurs. When exposed to air the nitride gradually changes over to a brown oxide.

Cerium filings were heated in the vapors of sulphur. When the black substance which is formed is treated with dilute sulphuric acid, sulphuretted hydrogen is evolved. Cerium sulphide has also been formerly studied by various investigators.

Cerium filings were heated to a temperature of 500° C. in an atmosphere of carbon monoxide. There was no visible reaction, but part of the filings appeared oxidized.

Alloys.—A number of different alloys of cerium were studied. In most cases the alloys were high in cerium, usually containing about 70 per cent. cerium. Alloys were made with silver, gold, platinum, copper, tin, antimony, arsenic, carbon, silicon, sulphur, selenium, tellurium, lead, calcium, sodium, aluminum, zinc, cadmium, chromium, manganese, iron, nickel, tungsten, mercury and magnesium.

Cerium and silicon form the silicide $CeSi_2$. This compound may be formed by the reduction of the oxide of cerium. Large amounts of the mixed silicides have been prepared by the author. The following mixture is recommended for their preparation:

M_3O_5	1,000	grams
Powdered graphite	200	"
Powdered silicon	450	"

The silicide of cerium is somewhat brittle, and may easily be pulverized to a fine powder. It is a splendid reducing agent. When the silicide is added to cerium so that the silicon content is about 15 per cent., a good pyrophoric alloy is obtained.

The iron-cerium alloys are very interesting, as they were the first pyrophoric alloys known, and were discovered by Dr. Auer von Welsbach. The alloys of about 70 per cent. cerium content are fairly hard and somewhat brittle. The microstructure of one of these alloys is shown in Fig. 11. It is seen that the structure is heterogeneous, probably consisting in part of a cerium-iron compound. These alloys have been the subject of some discussion in the literature,[53] as have the reasons for the pyrophoric properties.[54] The writer believes that the question of the pyrophority depends upon the following factors: Cerium alone is soft and malleable, and when scratched with a file, small particles are not broken off. However, when the alloy is hard and brittle, small particles are easily detached from the mass, and the friction is sufficient to raise the temperature of these small particles to the incandescent point of cerium (160°). As metallic compounds (such as Cu_3Sn) are as a rule hard and brittle, the addition of such compounds to cerium usually assures a pyrophoric alloy. The alloy should be high in cerium, so that ignition occurs at a low temperature, and should contain excess of cerium above that required for the formation of a compound (such as $CeSi_2$) to act as a binder and so prevent the disintegration of the alloy, as in the case of the cerium-aluminum alloy containing about 60 per cent. cerium.

The whole series of magnesium-cerium alloys were prepared and investigated for their optional properties. The alloy containing about 83 per cent cerium is highly pyrophoric. Most of these alloys are brittle and can be easily pulverized. Excellent flashlight powders can be prepared from the ones of higher cerium content. These alloys also form splendid reducing agents, as the combination of cerium and magnesium is an endothermic reaction, and when the alloy is oxidized, more heat is emitted than from an equivalent mechanical mixture of the two constituents. The fact that the alloys of from 60 to 75 per cent. cerium content may be easily pulverized in a mortar to a fineness of 200 should render these alloys valuable for thermal reductions. A small amount of metallic vanadium was prepared by reducing V_2O_5 by this method. The alloys of from 75 to 85 per cent. cerium content can be pulverized, but they are so highly pyrophoric that it is difficult to prevent igniting them.

Dr. Hirsch had quite an interesting series of samples which were passed around. In the discussion of the paper he was highly complimented on his success by Dr. Whitney, Dr. Parsons, and Dr. Northrup.

Dr. Richards spoke briefly on the cause of the sparking of the alloys. The pyrophoric alloys are brittle. When scratched with a knife fine particles are given off which are

heated to incandescence, but with pure cerium which is soft this does not take place. Dr. Hirsch said that to get a pyrophoric alloy there must be enough cerium in it, generally above 70 per cent.

Mr. Hansen thought the scleroscope was a good instrument for testing the hardness of different steels among themselves, but not for comparing the hardness of different metals or alloys with each other.

He thought the scratch test would give better results for comparison.

Thermal Conduction and Convection in Gases at High Temperatures.

A paper by Dr. Irving Langmuir of the Research Laboratory of the General Electric Company, deals with "thermal conduction and convection in gases at extremely high temperatures."

The author first describes a series of experiments on the formation of nitric oxide from air in contact with a glowing Nernst filament. Some theoretically curious results were obtained. They seem to suggest that some new and previously unsuspected phenomenon was taking place. A dissociation of the gas molecules into atoms or an absorption of energy by the emission of ions from the hot surface suggest themselves as possible causes, but this could only be decided by further experimental investigation.

However, the Nernst glower is not suitable for such experiments, because the range of temperature is limited. The author, therefore, continued his experiments with ductile tungsten wire in a hydrogen atmosphere. Some of the results obtained are as follows:

The energy required to heat the wire to a given temperature is very nearly independent of the size of the vessel containing the hydrogen.

The relation between energy consumption and temperature was determined, and the loss of energy was found to increase very rapidly with increasing temperature.

Up to about 1300 deg. K. the energy increases proportional to a power of the temperature between the first and the second, but above that it gradually increases more rapidly until at 3400 deg. K. it is actually increasing with more than the tenth power of the temperature. In a vacuum the energy loss is proportional to the 4.8 power of the temperature, and this exponent is constant over a very wide range. So, for a wire of about 0.045 mm. diameter, heated in hydrogen, the ratio of radiated to total energy reaches a maximum at about 2600 deg. K. At this temperature, however, only about 9 per cent of the total energy is lost by radiation, while 91 per cent is carried away by conduction through the hydrogen. At 1000 deg. K. the proportion of radiated energy is about 0.7 per cent, and at 3400 deg. it is 3.4 per cent of the total energy.

These results with hydrogen strongly confirm the conclusions drawn from the experiments with the Nernst glower. The most reasonable explanation of the very large heat conductivity of these gases at these extremely high temperatures seems to be that the molecules dissociate into atoms close to the filament, thereby absorbing a large amount of heat. The atoms then diffuse away from the wire, and there, because of the lower temperature, recombine and give up the heat that was absorbed from the wire.

Experiments are now in progress to determine the energy loss of a tungsten wire heated in monatomic gases. (From remarks made by Dr. Langmuir during the discussion it would appear that recent investigations of this kind by the author have made the correctness of the above explanation at least questionable, although the results are not yet conclusive.)

In connection with these experiments it should be noted that the candle-power measurments indicate that the melting point of tungsten is at least 3450 deg. K. Pirani had found 3570 deg. by a very unsatisfactory method of extrapolation, but the best value up to that time, and the one that even now seems to be generally accepted, is that of von Wartenberg, who gave 3170 deg. K.

In the course of the discussion in which Dr. Northrup and Dr. Lloyd participated, Dr. Langmuir discussed the discrepancy of a tungsten filament from a thermodynamically black body and the advantages of determining the melting point of tungsten in hydrogen.

Measurement of Small Gas Pressures.

A paper by Dr. C. F. Hale, of the Research Laboratory of the General Electric Company, deals with the "measurement of very small gas pressures." The extension of studies of the phenomena in rarified gases to the region of very low pressures has made increasingly important the development of manometers whose indications of pressure are not dependent upon the use of mercury.

The manometer of Pirani (*Ber.* der Deutsch. Phys. Ges., 1906, p. 686), depends for its ability to measure pressures upon the fact that at low pressures the heat conductivity of gases is a function of the pressure. It is only necessary, therefore, to observe the effects of varying heat dissipation, with pressure changes, upon some source of heat within the vessel undergoing evacuation.

Pirani has accomplished this by connecting to the system an ordinary tantalum lamp. The wire in this lamp was heated by a constant-voltage current. As the pressure in the system was decreased more and more, the heat lost by conduction through the gas was continually decreased; consequently the temperature of the wire mounted steadily, and, with the temperature, the resistance of the wire increased, thereby cutting down the current.

One had then only to calibrate ammeter readings against the indications of some form of manometer.

A better instrument was obtained when the tantalum wire was replaced by platinum, and the wire was clamped tightly to the anchor wires in order to keep constant the heat loss through the supports.

In simplicity both of construction and of operation, involving only current and resistance measurements within the range of instruments readily accessible, the Pirani manometer recommends itself at once.

However, in point of sensitiveness to pressure changes it is somewhat disappointing.

The present author has modified the Pirani manometer so as to increase its sensitiveness quite materially. The construction of the improved manometer is described in great detail and it is found to give reliable measurements of pressure down to 0.00001 mm. of mercury.

A method of calibrating the manometer is also described. Since the heat conductivity varies with the nature of the gas, it is necessary to calibrate the manometer with the same gas with which it is to be employed subsequently. Calibration curves are given for pure dry air and for hydrogen.

The Electrochemical Industries of Norway.

Prof. Jos. W. Richards, of Lehigh University, who had just returned from abroad, gave a description of a number of electrochemical plants in Norway, which he had visited this summer. There are in the whole thirteen electrochemical plants in Norway and he visited nine of them.

First an aluminium plant at Kristiansand was noticed which operates with a capacity of 12,000 hp by the Heroult process, producing 800 to 1000 tons of aluminium per year.

The next plant dealt with was a nickel refinery depending for its ore supply on a copper-nickel ore mine. The ore is smelted by semipyritic smelting near the mine and the nickel-copper matte shipped to the refinery where the nickel and copper are electrolytically separated by the process of Hybinette (our vol. IV, pp. 33 and 34, 1906).

The next plant described was the experimental plant of the Badische Company for the manufacture of sodium nitrite, 2000

hp being used for this purpose. The well known Schoenherr process for burning the atmospheric nitrogen by electric discharges is employed. But the company finds it more profitable to work up the nitrogen oxides into nitrite instead of nitrate. The nitrite is shipped to Germany. The production is 300 to 400 tons per year.

The next plant visited by Dr. Richards was the aluminium plant erected by the British Aluminium Company in 1906, using 2000 kw and producing about 600 tons a year.

Close by is a sodium plant which was started by Mr. Ashcroft. When Dr. Richards visited it it had been closed down, due to lack of water, but when the power supply does not fail, it is in operation. The Castner process for making metallic sodium is used with a few modifications.

At Odda there are two large and interesting plants, one being a carbide plant and the other a cyanamide plant.

The carbide plant contains 12 furnaces of 1500 hp each and one large one of 2000 hp. The factory works very regularly.

An interesting device for protecting the workmen against the heat from the furnaces is used there. It is a double screen of wire, two screens being 1 in. from each other placed before the furnace. The view is not shut off thereby, since one can look through the screen as through a mosquito screen, but it cuts off a great deal of heat.

Twenty thousand hp are utilized at these works, producing 30,000 tons of carbide per year. Of these 10,000 tons (largely fine stuff) are supplied to the cyanamide works. The carbide plant is to be doubled next year and the whole extra production is to be sent to the cyanamide plant.

This indicates that the cyanamide industry is in a good condition, but it is noteworthy that they prefer not to sell cyanamide itself as a fertilizer but to convert it into ammonium sulphate. The cyanamide is first converted into ammonia and then into ammonium sulphate. The sulphuric acid necessary for this process will be supplied from a near-by plant, to be mentioned below.

Four miles north of Odda there is the Tysse water power, where 125,000 hp will be available.

One of the interesting developments there is electric zinc smelting. Mixed Broken Hill concentrates will be used. They will be roasted and the sulphuric acid will be shipped to the cyanamide plant mentioned above. The zinc oxides will be reduced in electric furnaces. The furnace to be used is stated to be a modified De Laval furnace, but is probably a resistance furnace rather than an arc furnace.

Smelting of pig iron in the electric furnace has not yet been started in Norway, but will be started next year at Hardange. The design of the two furnaces, each of 4000 hp, will be based on the experience obtained at Trollhättan. Also, steel furnaces of the Ludvika type are installed there.

At Arendal there is another company which will produce pig iron in two 4000-hp electric furnaces. This is one of the undertakings of Mr. Eyde.

With respect to electric steel the Norwegians are looking now toward the use of steel scrap. At Stavange there is a company which breaks up old ships. The scrap is to be used by the Stavange Electric Steel Works. They estimate they will have a supply of 3000 to 6000 tons of scrap per year. The water-power plant is being built and the steel plant is to be completed by December.

The steel is melted down in a Siemens open-hearth furnace and roughly refined and is then still further refined in a Roechling-Rodenhauser electric furnace. There is also a rolling mill and a foundry.

At Mr. Hiorth's plant the induction furnace is used for melting down high-quality metal and making a high-quality steel, but he also uses an electrode furnace for working up the scrap.

Dr. Richards attended the jubilee of the University of Kristiania, and found that all Norwegians were extremely interested in the electrochemical industry, as they expected that electrochemistry would play an important part in the future development of Norway. Since strong restrictions against foreign capital have been put in force in Norway, Norwegians now are putting their own money into such developments. The King of Norway is much interested in the use of the electric furnace for iron and steel. Similar interest is taken in all these developments in Sweden.

Among the plants which were not visited by Dr. Richards is the large Nottoden plant for making nitrates from air by the Birkeland-Eyde process. It has been found that for large units the Birkeland-Eyde furnace, being compact, is more economical than the Schoenherr furnace, which must be built high up in order to increase its size.

As to the cost of power in Norway, Dr. Richards stated that power could be bought there from a power company for $6.50 per hp-year. This is at a place right at deep water, so that shipping facilities are also provided. The cost of producing power was given as between $4 and $5 per hp-year.

Galvanizing Wire in Zinc Dust.

A paper by Mr. Alfred Sang, of Paris, France, emphasizes that a perfectly clear conception of the forces at work is essential for successfully operating his process of galvanizing steel wire by heating it electrically in zinc oxide or zinc dust.* Once the nature of zinc dust was well understood and the action which different temperatures may have on it was clearly demonstrated, the application of the process was found to be, after all, not nearly as complicated as might have been feared, in fact it was less complicated than the other two processes in general use.

It may be recalled that zinc dust does not melt when heated to the temperature of fusion of zinc, but at higher temperatures reducing agents will "precipitate" the metal, as it were, and it can also be squeezed out under mechanical pressure. This property of resisting fusion is supposed and is probably due to the surface of each particle being formed of zinc oxide or zinc carbonate.

If zinc dust is heated evenly, the temperature is found to rise steadily until around 390° C., when the rise becomes much more rapid, attaining its maximum rate at about 420°, then falling gradually until in the neighborhood of 470° it again becomes normal. When the dust is cooled down an opposite effect is noted between 450° and 260°. If the operations of alternately heating and cooling the same dust are repeated, the exothermic and endothermic effects are manifested each time between the same temperatures, with their apices at about 420° on the rise and about 350° on the fall.

It is noteworthy that the exothermic effect on the rise occurs at the melting point of zinc metal, but when zinc is melted the effect is the very reverse, heat being absorbed at the moment of fusion to the extent of 28 calories per kilogram.

This represents, therefore, an anomalous critical point, and the author considers separately the operations of galvanizing in zinc dust below and above that critical point.

If a clean piece of steel wire is immersed in zinc dust which is heated to a temperature below 420° C., or at what is known as "sherardizing temperatures," a slow cementation takes place and zinc is absorbed from the very moment that the wire reaches the temperature of the dust; there is at the same time a slow sublimation of zinc at the surface increasing with time but sufficient for all practical purposes after a few hours of treatment; in the presence of reducing gases the time of treatment can be considerably lessened, but the penetration of zinc is, of course, less. This process is sherardizing, unsurpassed for many purposes, unsuitable for many others. There are two varieties of sherardized coatings; the gray, porous and brittle variety, mostly composed of zinc carbonate which darkens with age; the other variety has a high metallic luster

*This journal, Vol. VII., p. 532. Concerning sherardizing see this journal, Vol. V., p. 187; Vol. VI., p. 189, and Vol. VII., p. 485.

and is compact and of a finely crystalline structure. If this were the class of sherardizing produced by all existing plants, there would not be much hot-galvanized work produced outside of wire and sheets.

When we work with the dust at a temperature above the critical 420°, results are entirely different; galvanizing is instantaneous, taking place the very instant that the wire reaches the temperature of the dust. There is very little penetration of zinc in the iron; there may be vaporization and condensation of zinc as in sherardizing, although this is doubtful. A wire heated to 800° or 900° in a non-oxidizing flame and quickly thrust into cold zinc dust will condense around itself a considerable thickness of zinc. The best working temperature ranges between 500° C. and 600° C. The galvanizing is bright and more pleasing in appearance than that obtained by any other means.

The author does not go into details of the furnace used, which merely consists of a zinc-dust trough between two sand troughs provided with a set of cast-iron cover-plates over the zinc trough and power-driven rolls. As high a speed as 850 ft. per minute can be used for securing good merchant galvanizing. It is found preferable to run at 35 ft. only, running a large number of strands and attaching the coils end to end.

The process is economical as regards current consumption which outside of zinc and labor, with a very small amount of power for driving the rolls, represents the bulk of direct cost.

The dust very rapidly assumes a fairly constant temperature. When this is the case the consumption of electrical energy in watt-hours is 7/6 times $w\,s\,t$, where w the weight of wire treated (say 1000 kg), s the specific heat of steel=0.18 at 550° C. and t the temperature to be maintained, say 550° C. The calories needed are $w\,s\,t$, hence by conversion into electrical units we get 7/6 $w\,s\,t$ watt-hours.

Figuring 75 per cent efficiency for the dynamo or transformer supplying the current, the above formula gives 154 kw-hours per long ton of wire of any diameter.

While the wire is softened by the process, a full annealing can only be secured by using a higher temperature, the consumption of electrical energy being then increased from 40 per cent to 50 per cent and the sand trough through which the wire passes after leaving the zinc dust must be appreciably lengthened. If a slight scale is to be removed, the high temperature must be used in connection with a reducing agent. Sand must be used with the zinc dust to prevent caking and cavitation in the path of the wire. Aluminium contact rolls and special mercury contacts are the only ones suitable for this kind of plant.

The wire can be brought out into the air immediately after coating, but it is preferable to let it cool by passing it through sand which brightens it at the same time.

The instantaneous galvanizing of steel in zinc dust above 420° is rapidly explained by completing the theory which the author developed as far back as 1907; it is now stated in its complete form as follows:

"Zinc dust is produced by the sudden cooling of the vapor of zinc and the transition from the gaseous to the solid state is so rapid that, for all practical purposes, the liquid state is skipped. The setting of the zinc, brought about by sudden chilling, prevents regular arrangement of the molecules along determined stereochemical lines; it forbids the formation of crystals for which time is required. The heat of solution of matter in an amorphous state is always higher than when in a crystalline state; the same is true of a strained condition which likewise presents an amorphous appearance. At the time of formation considerable heat is retained which would have been radiated by slow cooling. The condition is one of physical instability resulting in readier and, oftentimes, explosive decomposition.

"The physical instability of zinc dust may be explained thus: In the prolongs of zinc furnaces the uncondensed vapor is suddenly chilled and collects in minute drops which are instantly congealed at the surface and oxidized, forming a rigid spherical skin or crust. Within this crust the liquid zinc cools and contracts more slowly but the rigidity of the crust does not permit the drawing in of the surface to form the facets of a crystal and compensate this internal contraction. Thus voids must exist within the crust and we may presume a structure which might be described as 'piped radially' or as negatively crystalline.

"Below the melting point of zinc this condition persists, but affinities and thermal inducements find a readier response from zinc dust than from any other form of zinc, both by reason of the enormous amount of free surface and because of this unstable condition.

"At about 415°, which is the melting point of zinc, the strain within the zinc dust particle is totally or almost totally relieved as indicated by the loss of heat, although some of the latent heat must be retained for fusion.

"At high temperatures zinc dust is pasty, because formed of innumerable little bags of melted zinc, but these will not coalesce on account of their small size and closely packed condition, unless considerable pressure is used. These minute bladders might be compared to soft-shell turtle's eggs; they form a plastic and cellular mass.

"If, however, a reducing agent, say a drop of coal tar, is introduced, reduction of the skin takes place and we get a small mass of more or less spongy metallic zinc. If a metal which is electronegative to zinc, such as iron, is put into contact with these bags of zinc, reduction and an exchange of metals take place, the bag is punctured at the point of contact and the zinc spreads itself on the iron.

"Hence it is that wire can be galvanized in this manner by means of a plant closely resembling a hot galvanizing plant, for the process is, in fact, one of hot galvanizing."

Electric Nickel Smelting.

A paper by Mr. Walter Leonard Morrison, entitled "Electric Furnace Treatment of Nickel Ore and the Development of a Commercial Process" (being a thesis presented to the Case School of Applied Science), gave an account of the work done by the Consolidated Nickel Company, at Webster, N. C., about which an article was already published in this journal, Vol. VIII, p. 277 (May, 1910). This paper was distributed on the train on Saturday.

The nickel deposits are located about one-half mile from Webster, N. C. The analysis of ores from the same cross cut vary greatly in chemical composition and color. The elements which vary most are iron, magnesia, and alumina.

The reddish brown variety of ore carries the greatest amount of iron and alumina, while the green ores carry the greatest amount of magnesia.

A typical analysis of the hard green ore is 1.90 per cent nickel oxide, 34 silica, 48 magnesia, 3.4 alumina, 1 iron oxide, 1.7 chromic oxide, 10 water of crystallization.

A typical analysis of the soft green variety (genthite) is 42 per cent silica, 8.1 nickel oxide, 36.2 magnesia, 2.1 chromic oxide, 1.2 alumina, 1 ferric oxide, 10.8 water of crystallization.

While the quantity of ore running higher than 10 per cent nickel oxide is very small, the available quantity of ore averaging 1.75 per cent nickel oxide is enormous.

The deposit has been known since the latter part of the sixties, but being low-grade ore it has been impossible to work these properties for two reasons: first, because the former owners did not have money enough to develop a low-grade proposition of this magnitude, and second, until recently there was no process that could treat the ore without hand picking.

The nature and extent of the deposit make the mining a very simple matter, since the ore for the most part can be handled with a steam shovel and can be put into cars at a cost not exceeding 11 cents per ton.

The latest attempt to smelt this ore has been by means of the electric furnace, and as most of the nickel is used in the

steel industry, it was decided to make in one step from the ore a nickel silicide which could be sold directly to nickel steel makers.

Various samples of the nickel silicide thus produced analyzed about 10 to 25 per cent nickel, 55 to 50 iron, 30 to 20 silicon, aluminum, magnesium, chromium, etc., 5 per cent. The presence of the silicon aluminum, and magnesium is in no way objectionable in view of the purpose for which the alloy is intended, since they act as purifiers and gas removers in steel refining.

1. An experiment was first made in which it was tried to reduce the ore directly by hot reducing gases (CO) at a high

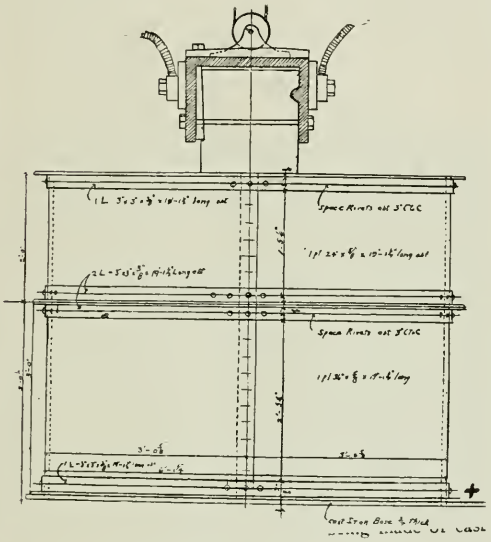


FIG. 1.—SIDE VIEW OF FURNACE.

temperature. The ore mixed with charcoal was placed in a vertical flue and the hot reducing gases passed upward through it. But although the temperature of the ore mass was raised to 1100 deg. C. and held there for an hour, there was absolutely no reduction to metallic nickel. The nickel appears to be held very strongly combined as the double silicate of magnesia and not to be reduced or affected in any way by the hot carbonous oxide gas.

2. About 60 to 75 per cent of the nickel ore runs very high in magnesia (25 to 38 per cent), but low in iron. The fact that the ore is so high in magnesia, makes it a difficult proposition to form a fluid slag, even in the electric furnace, which produces the highest temperature of any present commercial furnace. The idea naturally suggested itself, that the ore could be subjected to a high temperature, in a reducing atmosphere, and intimately mixed with charcoal, and then removed as soon as reduction fusion had taken place. The semi-fused mass could then be crushed and the nickel separated from the slag by a magnetic separator, or on the Wilfley concentrator.

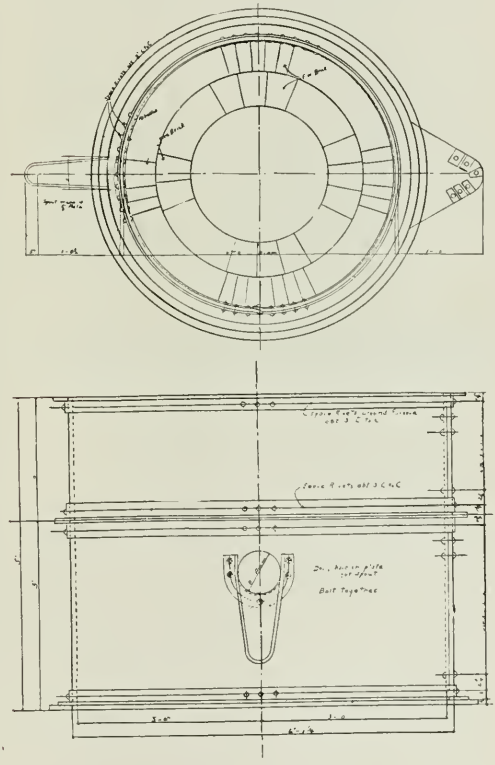
The experiment was carried out in a heating chamber heated by an electric arc, with an inclined bottom, so that the fused mass would flow off in a continuous stream. It was crushed and concentrated on the Wilfley table. The amount of ore treated per kw.-day was very low (14 pounds) compared to the results of a fusion or slag-making electric furnace, which will smelt 23 to 27 pounds of ore per kw.-day. The concentrates from the Wilfley table were high in nickel, but the total extraction of nickel from the ore was about 60 per cent, which is very low.

Further, a furnace of this kind would, after being run for a few hours become so strongly heated that the lining would be in serious danger of falling in from fusion. The experiments along this line were discontinued for they did not appear very promising.

3. Wet processes were then tried. Experiments were made with sulphuric acid, ammonium chloride, sodium chloride, sodium sulphate, and chlorine gas, but neither gave satisfactory results.

4. **Electric Smelting.**—Laboratory experiments were made with a 100-ampere 50-volt Borchers furnace, obtained from Eimer & Amend, with a fixed lower electrode and a movable upper electrode, and also with a larger furnace of the same general type.

The furnace was tapped about every 15 to 20 minutes, giving about 10 lb. (4.5 kg.) of slag and metal, about a quart (liter) in all. The author thinks that with this furnace it is not neces-



FIGS. 2 AND 3.—PLAN AND FRONT ELEVATION OF FURNACE.

sary to tap so often, first because the efficiency of the furnace is increased if the slag is removed as soon as it has become inactive in the reactions within the furnace; second, if the "source of heat" which is near the dividing line of the slag and ore be raised too much above the bottom of the furnace, the metal on the bottom will be cooled to its solidifying point, and the furnace will start to freeze. Further, if the furnace is let run too long between tappings, carbides are formed in the slag, which renders it very infusible and sticky, so that it will not flow.

The capacity of the furnace was limited to the capacity of the gas engine, which gave a maximum of 15 kw. at the switch-

board. The furnace would treat about 100 lb. (45 kg.) of ore in six hours.

Several charges of the green and brown ores were heated with charcoal in graphite crucibles in a gas blast furnace.

The object of this special series of experiments was to get an idea of the self-fluxing qualities of the green and brown ores.

For this purpose the relative weight of brown ore in the mixture was varied from 90 to 10, the corresponding weight of green ore from 10 to 90, a weight 5 of charcoal being added in each case. A mixture of 40 weight of brown ore, 60 of green ore, and 5 of charcoal appeared to give the best slag. Since the greater part of the ore is high in magnesia (the

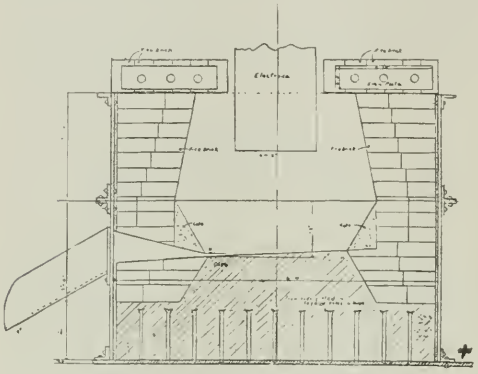


FIG. 4.—SECTION OF FURNACE WITH LINING.

green ore), it is an advantage to be able to use the brown ore as flux instead of barren matter.

Under good working conditions the slags varied about as follows:

Silica	40.00	to 45.00	per cent
Alumina	30.00	to 20.00	"
Ferric oxide	2.00	to 0.50	"
Calcium oxide	3.00	to 10.00	"
Magnesia	20.00	to 15.00	"
Nickel oxide	0.40	to 0.80	"
Alkalies	4.60	to 8.70	"

All the above slags gave little or no trouble in the furnace, flowed freely and allowed the metal to separate and collect in the bottom of the slag pot.

In the smelting of this ore in an open furnace there was considerable loss of nickel by volatilization. The alkalies and magnesia also volatilized to a considerable extent. The loss by volatilization is from 15 to 20 per cent of the weight of ore.

Lime in conjunction with carbon up to certain limits plays a very important part in the amount of silicon reduced. The author thinks that the formation of calcium carbide acts as a very powerful reducing agent and breaks up the silicates, liberating silicon and forming more complex silicates.

The iron aids in the collecting of the metallic nickel, very little iron goes to the slag. At first considerable trouble was experienced with the coke, it would accumulate in the furnace and finally cause it to freeze up. This trouble was practically eliminated by using finer coke, instead of ½-in. (1.2 cm.) coke 10-mesh (0.25 cm.) coke was used which gave better results. In order to get complete reduction and a good extraction it is necessary to add a little excess of coke, and being in the fine condition it is easily carried from the surface with the slag.

The formation of carbides in the slag is a signal that the furnace will soon give trouble, unless the charge is changed or the carbon lessened. The carbides are easily detected by the odor of acetylene about the cool slag. A slag which decomposes as it cools is another indication that something is wrong with the fluxing. The furnace must be tapped at comparatively short intervals of not more than 30 to 40 minutes.

Granulation of the slag was tried and was quite successful; the slag is then easily removed from the metal by simply washing. The metal remains in fairly large pieces, and in the form of shot. On a large scale, granulation would be a very cheap method of separating the metal, and the slag could be easily flushed to the dump.

The Wilfley Concentrator with material through 30-mesh (0.8 cm.) gave excellent results.

The metal reduced from the ore and separated from the slag is in all cases an alloy of nickel, iron and silicon. In no run were they able to obtain a low-silicon alloy, and keep their extraction about 80 per cent. This alloy must from necessity carry considerable iron. The amount of metal separated from a ton of ore depends upon the ore treated. The brown ore will give about 440 lb. (200 kg.) of metal to the ton, while the

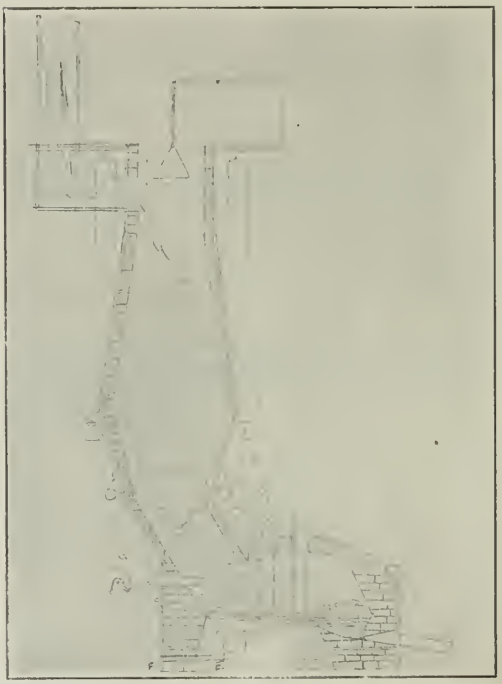


FIG. 5.—ELECTRIC SHAFT FURNACE.

green ore will not give more than 200 lb. (91 kg.) to the ton. Using a mixture of the ores the amount of metal derived, varies from 300 to 350 lb. (136 to 150 kg.) to the ton.

The following are analyses of metal obtained from the brown ores:

Nickel	10.90	11.00	11.50	10.20	
Iron	58.30	56.80	57.40	56.90	
Silicon	28.20	30.10	26.30	27.40	

The following are analyses obtained from green ore:

Nickel	23.95	21.80	22.60	19.30	18.96
Iron	50.70	51.20	48.90	49.10	54.00
Silicon	21.20	18.80	23.24	26.40	22.80

The following are analyses of metal obtained from the mixtures of the ores:

Nickel	15.04	13.70	13.10	13.30	14.80
Iron	54.80	53.90	56.00	57.80	54.70
Silicon	26.60	27.40	25.70	24.00	26.20

In looking over the above analyses, it will be noted that the green ore gives a metal much higher in nickel, all the runs from which this metal was derived were made on picked ore averaging about 6 per cent nickel. It is not advisable to pick ore too much, as it will leave a worthless ore which it would not pay to treat; then too the amount of rich ore to the lean ore is about 1 to 10 or 20 and means a large expense to sort out.

The ferronickel silicide also contains aluminum, calcium, magnesium in traces, besides 1.5 to 2.5 per cent chromium.

Notes are given in the paper on the methods of analysis of the ferronickel silicide and the ore.

Commercial Electric Smelter.—The electric smelter and equipment is located on the nickel deposit at its middle point.

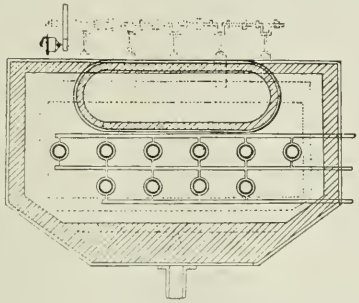


FIG. 6.—DETAILS OF FURNACE CONSTRUCTION.

In the boiler room there are four horizontal-fired tube boilers of 200 boiler hp. each, bituminous coal being fired. The steam engine is of the Hamilton Corliss type. The line shaft drives two 170-kw. generators, giving 3500 amperes at 500 volts direct current.

The furnace room equipment consists of the furnace and electrode control, switchboard, 4 slag pots, 500 lb. (225 kg.) capacity, and weighing scales for ore and flux.

The furnace used in Webster is similar to the one which was used at Sault Ste. Marie by Dr. P. L. T. Heroult, with a few modifications. The furnace was originally designed in two sections, it being intended that the top section could be removed in order to patch up the crucible, but later this was found to be unnecessary.

The bottom plate of the furnace is of cast iron 1.5 in. (3.8 cm.) in thickness, set on porcelain blocks for insulation. Tapered holes were bored in the cast-iron plate, and iron rods 10 in. (25 cm.) long by 0.5 (13 mm.) diameter were inserted, in order to give a large area of contact. A large copper plate was fastened to the cast iron at the rear, to which the lugs carrying the cables were secured.

Fig. 1 gives a side view of the furnace, the holder is shown, and method of holding the carbon electrode, also position of cables. The furnace shell is made up of ¾-in. (16 mm.) iron plate, and 3 in. by 3 in. by ⅜ in. (7.5 x 7.5 x 0.95 cm.) angle irons.

Fig. 2 gives a plan of the furnace, showing the thickness of the fire-brick lining.

Fig. 3 is the front elevation of the furnace, showing slope of hearth and position of spout, one spout being used for both the slag and the metal.

Fig. 4 is a section of the furnace after the lining is in; also showing the covering used.

The shell of the furnace was insulated from the cast iron bottom with heavy asbestos and fibre bushings around the bolts. The interior of the shell was first lined with ¼-in. (0.6 cm.) asbestos. A mixture of hot graphite and glucose was tamped into bottom of furnace around the iron pegs. Above the pegs the amount of graphite was lessened, and coke dust substituted, and near the top of the bed coke dust with a binder was used. The coke lining was carried up the side for a distance of ten inches; this coke lining appears to resist the corrosive action of the slags better than any other material we used. The cover of the furnace consisted of fire brick clamped together by tie rods.

The lining of the furnace required about four days, one man and helper. The graphite and glucose was heated in an iron soap kettle. Materials used in the lining and cover of the furnace were as follows: One barrel glucose, ¾ ton coke, 2,000 fire bricks, 1 barrel sand, 1 ton graphite, 1,000 circular fire bricks, 2 barrels fire clay, 20 sheets asbestos.

After the lining was all in, a wood fire was built in the furnace and kept burning three days. This dried the furnace very well. After this the electric current was run in the furnace for several hours, which finished the drying out of the bottom of the furnace.

Operation of Furnace.—The furnace was started with a little coke in the bottom, but after a few trials this was found to be an unnecessary precaution. The electrode was controlled by a hand-wheel located on the floor, connected to a worm and gear on a carriage, by a chain.

To the gear on the carriage was secured a notched wheel which carried a chain connected to the electrode holder. The holder is very heavy, being made of cast iron and weighing about 1,000 lb. (450 kg). The carbon electrode which was first used weighed about 1,400 lb. (635 kg.) when new. During the operation of the furnace if the current was shut off for

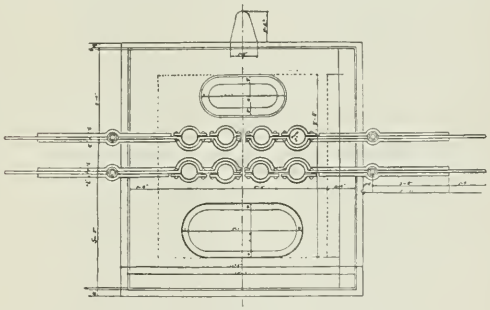


FIG. 7.—CRUCIBLE FURNACE

more than an hour it would usually cause the furnace to freeze up. At first the coke caused trouble in the furnace by accumulating, but this was eliminated by using finer coke and a little less of it. They also noted a decrease in the electrode consumption when finer coke was used.

The electrodes which we first used were of carbon, built up from smaller electrode sections. The large electrode was 18 in. (45 cm.) square by 7 ft. (2.2 m.) long, made from 3 by 6 in. (0.75 x 15 cm.) carbon slabs. The first time the electrode was heated by the arc it cracked and pieces broke off in several places. The pieces were picked out of the furnace, and ore with flux fed in, trusting that after a slag bath had been formed the cracking would stop, however this was not the case; after having to stop frequently to pick out solid pieces of the electrode weighing 25 to 50 lb. (11 to 23 kg.), the furnace was shut down, this electrode after so much damage being too short for further use. It was removed and the other carbon electrode placed in the holder.

Before striking an arc with the second carbon electrode a wood fire was placed beneath it in order to heat it up evenly. After two hours of this treatment the electrode was placed in the furnace, contact made, and ore started in. The electrode, however, went to pieces.

A copper bushing was then ordered for the holder, so that they could use two Acheson graphite eletrodes together. The Acheson electrodes were spliced so as to feed continually with-

out waste. These electrodes held up finely and gave very good results. The dynamos gave trouble, however, and they could only draw half load, consequently for nearly all the runs only one 8-in. (20 cm.) Acheson graphite electrode was used, which carried as high as 5,000 amperes at times.

Trouble was also experienced with the ore if not dried in advance.

The "efficiency" of the furnace under the present condition is said not to be very high. The author says: "The efficiency of this furnace as compared with one utilizing the waste gases to preheat and dry the ore, I would estimate at 60 to 70 per cent. The fact that we used wet ore instead of dry ore makes a difference of 6 to 8 per cent in efficiency."

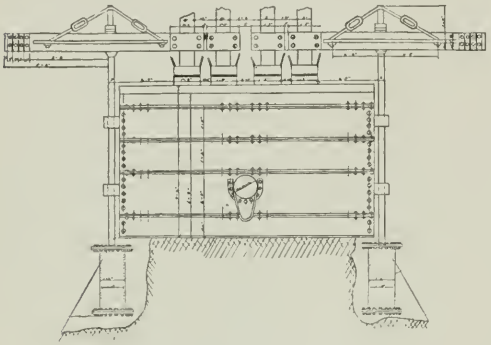


FIG. 8.—CRUCIBLE FURNACE

The following is the record of a furnace run made on December 9, 1909:

CHARGES.

No. 1.	No. 2.
4 tons brown ore.	3 tons brown ore.
1.02 tons limestone.	0.6 tons limestone.
0.48 tons coke.	0.48 tons coke

Moisture in ore	42.50 per cent
Total dry ore treated	8,050 pounds (3,660 kg.)
Net time of smelting	45.75 hours
Limestone used	3,600 pounds (1,635 kg.)
Coke used	1,940 pounds (882 kg.)
Power used, average kilowatts	170
Kilowatt hours used	7,755
Kilowatt-years used	0.896
Ore treated per kw.-year	9,000 pounds (4,090 kg.)
Ore treated per kw.-day	24.7 pounds (11.2 kg.)
Nickel silicide obtained	1,242 pounds (565 kg.)
Per cent of nickel in alloy	11.90
Silicide obtained per kw-year	1,380 pounds (627 kg.)
Silicide obtained per kw.-day	3.8 pounds (1.7 kg.)

In other runs, from 8,250 lb. (3,750 kg.) to 10,160 lb. (4,620 kg.) of ore was treated, and from 975 lb. (443 kg.) to 2,230 lb. (1,014 kg.) of nickel silicide obtained per kilowatt-year of electric energy consumed. The power consumption is nearly constant per unit weight of ore treated, independent of its metal contents, showing that the chief worth of the furnace is the drying of the ore and the smelting of the slag. The silicide produced contained 10.50 to 22.60 per cent of nickel.

The cost of production varies with the nickel content of the ore and of the silicide formed. The cost of smelting by hydro-electric power is only about 14 per cent of what steam power cost us. The above figures are quite conservative in the case of the hydroelectric power.

Cost of Operation.—The cost of operation is based on the approximate cost as the plant was operated from October, 1909, to April, 1910, and goes to show the expense of operating or trying to operate in a primitive country. Two shifts of 11 and 13 hours respectively were run. The furnace was run about 4 to 5 days out of every week, very little repair work being necessary on the furnace. The engine gave us considerable trouble and during every run several stops were necessary for adjustment of the crank pin or the valves. The generators giving the trouble they did, and forcing the plant to operate at 50 per cent of full load, caused a great decrease in the efficiency of the power plant. The cost of smelting cannot therefore be reasonably figured from the cost of operation which the author gives as follows:

Cost of operation per 24 hours:

1 Superintendent	$10.00
1 Assistant	2.50
2 Oilers @ $1.80	3.60
2 Firemen @ $2.40	4.80
2 Assistant firemen @ $1.80	3.60
2 Head furnacemen @ $2.50	5.00
4 Assistant furnacemen @ $1.80	7.20
2 Miners @ $1.50	3.00
1 Blacksmith	2.00
1 Timekeeper	2.00
Coal, 18 tons, @ $4.50	81.00
Machine oil	4.00
Electrodes	9.10
Flux and coke	3.00
	$148.80

The average power at the switchboard was 200 kw. and on

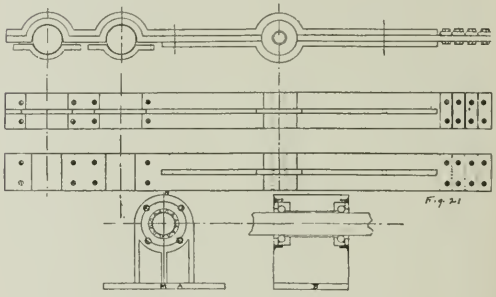


FIG. 9.—ELECTRODE CONTROL

an average 7,800 lb. (3,500 kg.) of ore were smelted per 24 hours.

Cost of power: $0.74 per kw.-day, $270.00 per kw.-year.

Cost of power under full load would be: $0.562 per kw.-day, $206.00 per kw.-year.

In the above figures no account is taken of depreciation, interest, and repairs, which would undoubtedly raise the above figures 10 per cent.

Output of Plant.—The output of the plant was limited to the available output of the generators, and varied from 800 to 1,200 lb. (365 to 545 kg.) silicide per day. This output on the present plant could be raised to 2,500 lb. (1,135 kg.) of nickel silicide, by operating the generators at full load, and using a dryer and preheater with the furnace. However, the furnace has served its purpose for a commercial demonstration, and has given the operators a chance for observations which are only possible with a furnace of this kind.

In the concluding part of this paper the author gives a discussion of various furnace designs and a proposed improved electric smelter design for treatment of the ore under discussion with estimated cost sheets.

The Shaft Furnace.—The shaft furnace for electric smelting is the ideal furnace. However, since the flow of gases is slow compared to the air blast furnace, it does not need to be so very high. For ordinary operations on the mixed nickel ores a shaft furnace should not be more than 10 to 12 feet high, *i. e.*,

above the fusion zone. There is considerable fines in the ore, which would cause trouble with a higher shaft. For this ore a dryer would be necessary, utilizing the waste gases from the furnace. Fig. 5 gives a general idea of the shaft furnace the author has in mind.

The ore with the flux is introduced at the top of the furnace from the charging floor. The furnace has a fore hearth for the collecting of slag and metal. The furnace is designed to carry ten 8-in (20 cm.) graphite electrodes. The ore enters the crucible back of the electrodes. To aid in the distribution of the sticky ore mass, a set of plungers (G) are arranged at back of furnace.

The furnace is designed for direct or alternating current (single phase). The bed of the furnace acts as one conductor, the dotted portion shows this bed. Tubular conductors carry

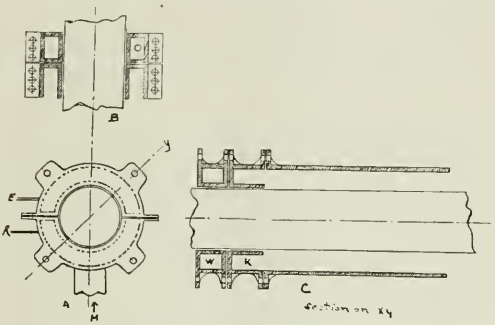


FIG. 10.—DETAILS OF JACKET CONSTRUCTION.

the current to iron distributers in the bed of furnace, which is heat insulated.

Coke is introduced at (c) independent of the charge; by so doing the electrode consumption is lessened; air enters for combustion of gases in the shaft at (A). The author has found that, by introducing the coke or reducer around or near the electrode, the consumption of electrode can be cut down one-half. Each electrode is operated automatically and independent of the others.

Although the shaft furnace is very good in theory, he does not think it practical at this time for the ore in question. The ore is extremely refractory and shut-downs are frequent. The cost of shutting down a shaft furnace and repairing would be very high. If it were desired to run the extremely fine ore alone at any time, then the crucible furnace would be the one to use. It would be impossible to run the extremely fine ore in a shaft furnace successfully. In the treatment of iron ores and other ores, the shaft furnace no doubt would be all right.

The crucible furnace.—By crucible furnace the author has reference to a furnace which is comparatively shallow and has no shaft for preheating and reducing the ores.

The crucible furnace illustrated in Figs. 7 and 8 is of a special design, which the author sketched, after careful observations on the present furnace. The furnace as shown is designed for direct current, which at the present time is installed.

The furnace is designed to carry eight 8-in. (20 cm.) Acheson graphite electrodes, operating under a maximum load of 2,000 electric h. p. The electrodes are worked in pairs and controlled by automatic hydraulic regulating device. The electrodes are carried in cast steel holders, to which are fastened eight flexible cables.

The holders are supported on a three-inch steel rod, which works in a hydraulic ram Fig. 8. To give the rod a free movement a special guide was designed, shown in Fig. 9. The ball bearings will not allow the rod to hang if properly lubricated and kept covered.

Fig. 7 gives a front view of the furnace, showing position of spout and electrodes. Fig. 8 is a side view of the furnace and gives an idea as to method of making the lower contact. The conductors are heavy copper tubing which enter the lower bed of the furnace; to these are secured wrought iron distributers.

The carbon bottom of furnace is not brought to the outside wall of furnace, but to the brick lining, which cuts down the radiation losses. There are openings in the top of furnace for electrodes, and a flue for escaping gases and an opening for ore.

The gases are taken off from the front part of the furnace and burned in the preheated. The ore as it leaves the preheater is dumped into the furnace at the rear. The two flues are brick lined. The interior of the furnace is lined very much in the way the experimental furnace was lined.

The electrodes are water jacketed; below the water jacket is a jacket constructed for the introduction of coke around the electrode. In Fig. 10 is a plan of jacket, B and C are sections. W shows section of water jacket. The water enters at E and flows out at R. K represents section of coke jacket. The coke enters at M and is fed by a screw device.

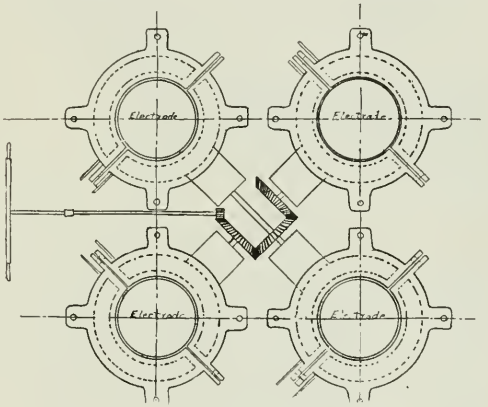


FIG. 11.—INTRODUCING COKE TO ELECTRODES.

The general plan of introducing coke to electrodes is shown in Fig. 11, feeding coke to four electrodes, the other four being taken care of in the same way. The two jackets are suspended on tie rods, then built in top of furnace. They are built in two sections so they may be easily renewed. For the first operations it is intended to feed the coke intermittently by hand feed.

By introducing the coke at this point, an extremely high reducing atmosphere is constantly kept around the electrode, consequently the electrode consumption is cut down. As to the reducing power, or action on the ore by this method, it is about the same.

The fusion zone is limited to a few inches immediately around the tip of the electrode, extending perhaps 8 or 10 inches (20 to 25 cm.). Under normal conditions there is a violent bubbling and boiling, which causes the fine coke 10 to 20 mesh (2.5 to 5 mm.) to intimately mix with the fused mass and bring about reduction. On a small scale the author has found that this method of introducing the coke works out very nicely. With frequent tappings the excess of coke is carried away in slag.

Electric smelter design.—Fig. 12 shows a proposed electric smelter design for the treatment of this ore. It consists of the generator room, furnace room, and ore bins.

The generator room contains the motor-generators for the

changing of the current from three phase to single phase. The room contains two 3,000 kw. motor-generators, the exciter, the high-tension switchboards, transformers and recording-meters. Current is to be furnished by outside power stations.

In the furnace room there are the furnaces, dryers, switchboards, slag and metal conveyors, hydraulic accumulator, and pumps. It is intended to keep three furnaces in operation, and have one for emergencies, which can be used at any time.

All electric cables are carried in conduits, except the cables to the electric holders. The hydraulic control is so arranged with each furnace that it may be regulated by hand or automatically.

All metal and slag is tapped into an iron trough of water, and is carried to the concentrating room by endless conveyor, where slag is flushed to dump.

The dryer is of such length that the ore is dried and preheated before entering the furnace. A check draft and explosion door is arranged at end of dryer. Each dryer has a stack to give better regulation.

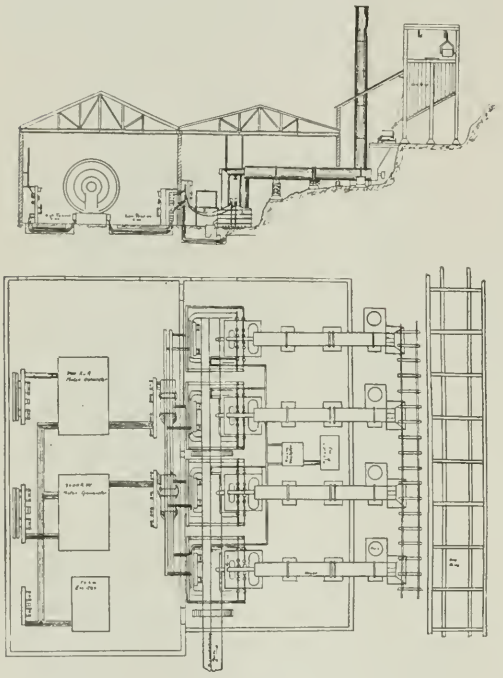


FIG. 12.—ELECTRIC SMELTER DESIGN.

The limestone and ore are crushed before being placed in the bins. From the bins they are drawn into weigh cars, then dumped into the feeding hopper of the dryer. A screw feed regulates the amount of ore being fed into the dryer.

Estimated cost of 100-ton plant.—Approximate cost of a 100-ton electric smelter, including aerial tramway, rock house, crusher, cars, buckets, etc.:

Ore bins	$ 3,000
Buildings	12,240
Tramways, cars and equipment	25,000
Weigh car	1,000
Crusher	900
Pulverizer for coke	800
Motors	1,600
Furnaces (5)	44,000
Motor-generators	45,000
Transformers	20,000
Structural work	22,000
Dryers and motors	15,000
Switchboard and equipment	36,000
Erection, labor, materials, etc	60,000
Total	$280,540

Although these figures appear quite high, from past experience the author thinks them conservative.

Cost of operation.—Estimate as to cost of operation of a 100-ton smelter, figuring operating 300 days per year, and cost of power $25.00 per kw.-yr.

Expense per day:

Interest and depreciation (14%)	$130.00
Mining @ 15 cents per ton	15.00
Power	450.00
Labor and supervising	75.00
Electrodes	130.00
Repairs	10.00
Coke and limestone	75.00
Total cost	$885.00

From 100 tons of ore on an average 15 tons of ferro-nickel-silicide would be extracted.

$885.00 ÷ 15 = $59.00 per ton, average cost of metal alloy.

The author concludes that the present plant for commercial operations is out of the question; that the wet process now existing is also out of the question, and that the only process available for this ore at present is the electric furnace process in a first-class efficient plant.

Entertainment of Ladies.

The program of the entertainment of ladies included a reception on Thursday afternoon by Lieut.-Col. and Mrs. J. B. Miller, at their residence. This was greatly enjoyed.

On Thursday evening there was a theater party at the Royal Alexandra ("Rebellion" with Miss Gertrude Elliott in the leading part.)

On Friday morning a motor drive was made through the city to the Lambdon Golf Club for lunch.

This report should not be concluded without emphasizing again the splendid hospitality extended to the Society by Toronto. The thanks of the visitors are due to all Toronto members and local committees, but special credit should be given to the chairman and secretary of the general committee, Dr. W. Lash Miller and Mr. W. P. Cohoe.

In the following we give an alphabetical list of all members and guests who registered at the meeting:

T. B. Allen, Niagara Falls, N. Y.; J. E. Babb, Pittsburgh, Pa.; L. H. Baekeland, Yonkers, N. Y.; T. F. Bailey, Alliance, Ohio; J. W. Bain, Toronto; W. D. Bancroft, Ithaca, N. Y.; E. A. Beck, New York; P. McN. Bennie, Niagara Falls, N. Y.; S. L. Bigelow, Ann Arbor, Mich.; Wm. Brady, Chicago, Ill.; Wm. C. Bray, Boston, Mass.; J. W. Brown, Cleveland, Ohio; C. L. Bryden, Scranton, Pa.; R. H. Bulley, Syracuse, N. Y.; A. Burton, Toronto; J. F. Callahan, Niagara Falls, N. Y.; H. R. Carveth, Niagara Falls, N. Y.; A. P. Cohoe, Toronto; E. R. Cole, Niagara Falls, N. Y.; W. M. Corse, Buffalo, N. Y.; J. L. Crider, Cleveland, Ohio; R. A. Crider, Cleveland, Ohio; Wm. C. Cuntz, New York; C. Dantsogen, Scotia, N. Y.; Tos. Davies, Toronto; S. Dushman, Toronto; J. W. Evans, Belleville; Mr. and Mrs. F. A. J. FitzGerald, Niagara Falls, N. Y.; M. W. Franklin, Schenectady, N. Y.; K. H. Gaines, New York; A. E. Gibbs, Wyandotte, Mich.; Miss M. E. Gibbs, Wyandotte, Mich.; Thos. W. Gibson, Toronto; W. L. Goodwin, Kingston, Ont.; A. M. Hamblet, Rumford, Me.; C. A. Hansen, Schenectady, N. Y.; H. E. T. Haultain, Toronto; Carl Hering, Philadelphia, Pa.; P. Heroult, New York; A. Hirsch, Corpus Christi, Texas; B. H. Hite, Morgantown, W. Va.; A. H. Hooker, Niagara Falls, N. Y.; L. E. Howard, Lockport, N. Y.; W. B. A. Howk, Elyria, Ohio; T. C. Irving, Jr., Toronto; A. von Isakovics, Monticello, N. Y.; O. S. James, Toronto, Ont.; A. J. Jones, Niagara Falls, N. Y.; W. P. Kaufmann, Toronto; J. Kelleher, Toronto; Mrs. and Miss Kelleher, Copenhagen; P. H. Kemble, Toronto; P. T. Kerivan, Toronto, Ont.; E. C. Klipstin, New York; Wm. Koehler, Cleveland, Ohio; F. L. Koether, Niagara Falls, N. Y.; M. M. Kohn, New York; A. T. Laing, Toronto; J. Langmuir, Schenectady, N. Y.; W. S. Lecky, Toronto; F. A. Lidbury, Niagara Falls, N. Y.; C. F. Lindsay, Bridgeport, Conn.; M. G. Lloyd, Chicago, Ill.; Thos R. Loudon, Toronto, Ont.; H. Lundgren, Chicago, Ill.; K. S. Maclaellan, Toronto; J. H. Mason, Toronto; W. Lash Miller, Toronto; J. W. Moffat, Toronto; G. W. Morden, Agric College, Winnipeg; W. F. Morrison, Lockport, N. Y.; J. M. Muir, New York; Dr. and Mrs. E. F. Northrup, Princeton, N. J.; A. Nueghorn, Toronto; C. H. Ohlwiler, Altoona, Pa.; A. C. Parsons, Buffalo, N. Y.; Chas. L. Parsons, Durham, N. H.; N. K. B. Patch, Toronto; M. Petinot, Niagara Falls, N. Y.; Jos Race, Toronto, Ont; M. Randall, Poplar Bluffs, Mo.; C. H. Rich, Clanton, Pa.; Jos. W. Richards, So. Bethlehem, Pa.; T. D. Robertson, Toronto, Ont; Mr and Mrs. A. G. Rodgers, Niagara Falls, N. Y.; E. F. Roeber, New York; S. S. Sadtler, Philadelphia, Pa.;

L. E. Saunders, Niagara Falls, N. Y.; C. N. Schluederberg, Pittsburgh, Pa.; W. B. Schulte, Madison, Wis.; J. A. Seede, Schenectady, N. Y.; Louis Simpson, Ottawa, Ont.; C. V. Slocum, Pittsburgh, Pa.; A. Smith, Niagara Falls, N. Y.; W. Snow, Penn, Ill.; R. C. Snowden, Niagara Falls, N. Y.; J. L. K. Snyder, Cleveland, Ohio; T. Somery, Lyons (France); E. C. Sprague, Niagara Falls, N. Y.; T. A. Stamps, Niagara Falls, N. Y.; W. C. Stone, Watertown, Wis.; Mr. and Mrs. E. R. Taylor, Penn Yan, N. Y.; R. W. Thornson, Toronto; R. Turnbull, Welland; Mr. and Mrs. L. D. Vorce, Detroit, Mich.; G. B. Waterhouse, Buffalo, N. Y.; Dr. and Mrs. O. P. Watts, Madison, Wis.; R. J. Weitlaner, Buffalo, N. Y.; W. R. Whitney, Schenectady, N. Y.; Roy Winne, Schenectady, N. Y.

Screenless Diametric Sizing of Ore.

A prerequisite for successful dry concentration of ore is close sizing. This holds true for separation based on the principle of difference of specific gravity and for electromagnetic or electrostatic separation of raw or roasted ores. The necessity for close and accurate sizing increases with the complexity of an ore; that is, when several minerals of narrow differences in specific gravity are found in the same ore.

Quite naturally screens of various forms and methods of operation have been used for diametric sizing and such devices are widely used for separating coarser grades. But the difficulties of diametric sizing increase with the fineness of the pulp and consequently hydraulic classification is more commonly used on the finer sizes. Nevertheless it would be preferable in some cases to continue sizing into the finer grades, obtaining sized products within narrow limits without respect to the specific gravity of the constituent particles.

A screenless sizer is not easily conceived of, but such is the nature of the machine invented by Messrs. C. L. McKesson and B. F. Rice, of Colorado Springs, Col. Their dry concentrator was described in our issue of September, 1910, page 549, and it was for the purpose of obtaining accurately sized products for this machine that the screenless sizer was evolved.

The work of the sizer bears no relation to that of a concentrating table, but for the purpose of getting a mental picture of it a comparison can be made. On first sight the sizer gives the impression of a table deck set at an angle of almost 45°, vibrating somewhat after the manner of a concentrator and fed with ore at about the same relative position. This is as far as the comparison can go, however, for the principle of specific gravity does not enter into the work of the sizer as with a table.

By reference to the accompanying drawing and photograph the following description will be understood. The working surface of the sizer is a deck 15 ft. long, 6 ft. wide at the back or feed end and 4.5 ft. wide at the head or discharge end. The deck is heavily trussed to prevent vibration and mounted at an angle of 38° on four inclined toggle supports which, in turn, are firmly supported on a base of I-beams. The deck is attached to a head movement which imparts a longitudinal vibration and its weight is counterbalanced on four compression springs, two under the upper and two under the lower side.

The working surface of the sizer consists of a number of so-called deflectors extending across its width. These deflector surfaces are about 6 in. wide and rise successively from the deck at an angle of about 9°, giving the appearance of a series of steps from one end of the deck to the other. They are not arranged in a uniformly parallel manner on the deck, but their side lines correspond to slightly diverging radii extending from an imaginary center above and behind the deck. Thus the deflectors on either half of the deck have different angles of inclination, corresponding to the positions which would be assumed by a pendulum swinging through a gradually decreasing arc.

The deflectors are covered with corrugated rubber, the line of corrugations extending lengthwise of the deck, or, more exactly, corresponding to numerous arcs described from the imaginary center referred to above. The corrugations are about 1/8 in. deep and number eight to the inch.

In operation the speed of the head movement is 320 r.p.m. The motion imparted to the deck is a combination of the reciprocating head movement, modified by the toggles. On its forward movement the deck is also raised vertically by the toggles and lowered on the return movement. The accompanying Fig. 3 represents graphically the two motions. One marked "stroke" represents the horizontal motion, and the other marked "toggle" shows the vertical throw.

The material to be sized is fed at the upper corner of the

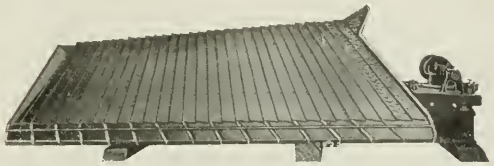


FIG. 1.—WORKING SURFACE OF SCREENLESS SIZER.

deck nearest the head motion. As the particles fall onto the deck they are acted upon by a combination of forces, including the peculiar motion of the machine and the arrangement of deflectors and corrugations, and make their progress across and down the deck. The coarsest particles will not progress beyond the first two deflectors which are covered with metal baffles, as shown in Fig. 1, and which act like a scalping screen

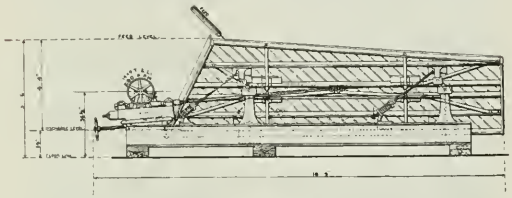


FIG. 2.—REAR VIEW OF SCREENLESS SIZER.

in cutting out an oversize for regrinding. Particles of successively smaller size will progress across and down the deck until they reach a critical point, beyond which they will not pass, whereupon they roll down the deflector and off the deck. The finest particles and dust remain on the deck longest and come off at the last deflectors or over the end of the deck. It must be remembered that this separation is wholly with regard to size and without respect to the specific gravity of the particles.

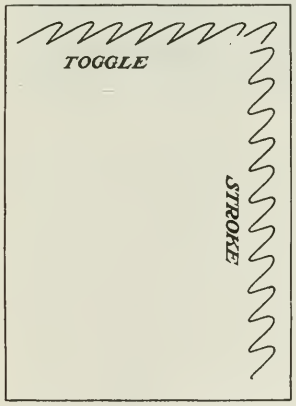


FIG. 3.—DIAGRAM OF DECK MOTIONS.

Along the bottom of the deck is arranged a series of cutting vanes to aid in collecting the various products according to grade. These cutters are adjustable, even when the machine is running. It is found that the zones yielding different grades do not necessarily correspond with a deflector, or any number of them, so the cutters must be set in positions determined by grading analyses to give the desired products. With the adjustments of the sizer once properly made it will continuously give uniform grades. The length of the stroke and inclination of the toggles will vary with requirements, and the capacity of the sizer also will depend in a measure on these adjustments as well as on the size of the material being treated. Recent tests indicate a capacity of from 15 to 20 tons per day of 24 hours on material crushed to about 20 mesh. It is ap-

parent, however, that a double deck can be arranged like an inverted V, which would have double the capacity, be lighter and more rigid, and at the same time require not much more power than the single deck, which is about 1 hp. The life of the sizer should be about the same as that of a standard concentrating table, and the cost of maintenance should be no more. The soft rubber surface of the deflectors should resist the abrasion of the ore very well.

The sizing efficiency of the machine is high and equally good work can be done on the fine and coarse sizes. The following tables show screen analyses made on 15 products obtained in a test. The ore was ground to pass a 24-mesh screen. The first table shows the uniform grading of the various products when divided into plus and minus 50 and 100-mesh portions. The second table gives screen analyses of the different products. It will be noticed that the grading efficiency in some of

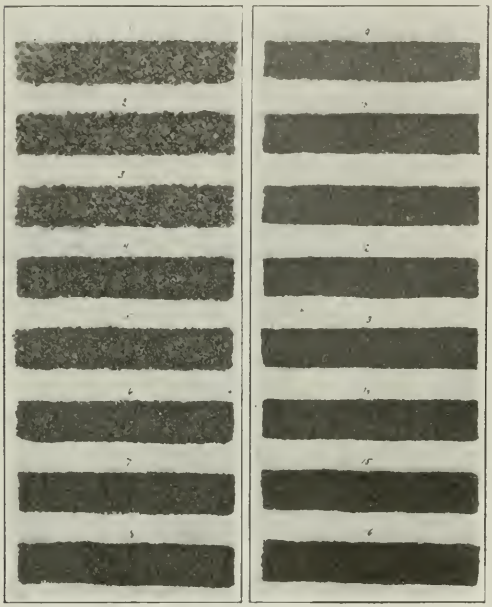


FIG. 4.—CARD SHOWING THE DIFFERENT SIZES MADE.

the products is lower than in others, but it is claimed that this can be remedied by adjusting the cutters to throw the oversize and undersize into the preceding and succeeding grades respectively, and thereby increase the efficiency of the sizing in all grades.

In addition to its use on ore, the sizer can find application in other industrial lines where sizing is required. Among the many materials requiring grading are sand and gravel, glass sand, abrasives, phosphate rock, mineral paints, salt, explosives, grains and seeds.

TABLE I.

Material ground through 24-mesh screen; graded into 15 products, each being screened through 50 and 100-mesh screens.

Product No.	Grade.	Screen.	%	Screen.	%
1	Coarse	+50	100	—50	....
2	"	+50	100	—50	....
3	"	+50	98.44	—50	1.56
4	"	+50	87.50	—50	12.50
5	"	+50	67.18	—50	32.82
6	Medium	+50	6.25	—50	93.75
7	"	+50	1.00	—50	99.00
8	"	+100	99.00	—100	1.00
9	"	+100	93.75	—100	6.25
10	"	+100	81.25	—100	18.75
11	"	+100	75.00	—100	25.00
12	"	+100	50.00	—100	50.00
13	Fine	+100	32.00	—100	68.00
14	"	+100	25.00	—100	75.00
15	"	+100	6.25	—100	93.75

TABLE II.

Grading analyses on 15 products of the McKesson Sizer. Ore originally ground through 24-mesh screen.

Product No.	Screens.	Per Cent.	Efficiency %.
1	— 24	....	
	+ 30	15.60	
	+ 40	81.20	81.20
	— 40	3.20	
2	— 28	....	
	+ 40	93.75	93.75
	— 40	6.25	
3	— 30	....	
	+ 34	10.92	
	+ 40	50.00	82.83
	+ 44	32.83	
	— 44	6.25	
4	— 34	....	
	+ 40	28.13	
	+ 44	37.50	62.50
	+ 50	25.00	
	— 50	9.37	
5	— 40	....	
	+ 44	7.82	
	+ 50	25.00	89.06
	+ 66	64.06	
	— 66	3.12	
6	— 44	....	
	+ 50	6.25	
	+ 66	68.75	90.52
	+ 82	21.87	
	— 82	3.12	
7	— 50	....	
	+ 66	37.50	90.63
	+ 82	53.13	
	— 82	9.37	
8	— 50	....	
	+ 66	9.37	
	+ 82	62.50	87.50
	+ 97	25.00	
	— 97	3.13	
9	— 50	....	
	+ 66	6.25	
	+ 82	50.00	87.50
	+ 97	37.50	
	— 97	6.25	
10	— 66	....	
	+ 82	37.50	84.37
	+ 97	46.87	
	— 97	15.63	
11	— 66	....	
	+ 82	25.00	75.00
	+ 97	50.00	
	+116	21.87	
	—116	3.13	
12	— 66	....	
	+ 82	15.63	
	+ 97	43.75	78.12
	+116	34.37	
	—116	6.25	
13	— 66	....	
	+ 82	6.25	
	+ 97	37.50	81.25
	+116	43.75	
	—116	12.50	
14	— 82	....	
	+ 97	25.00	
	+116	43.75	71.87
	+139	28.12	
	—139	3.13	
15	— 82	....	
	+ 97	6.25	
	+116	18.75	
	+139	43.75	68.75
	+172	25.00	
	—172	6.25	
		Average.....	81.32

Automatic Temperature Control.

The object of the temperature controller described in this article is to maintain the temperature of a liquid, a vapor or a gas automatically and exactly at a desired point. It is made by the C. J. Tagliabue Manufacturing Company, of 396 Broadway, New York City.

The action of this temperature control is based on the fact that different materials have different coefficients of expansion with temperature. If two strips of different metals, but of equal length, are placed side by side and welded together at one end only, then on raising or lowering the temperature the lengths of the two strips will generally change, but not in the

same degree, and the difference in the change of the length of the two strips can be made use of.

This difference in the change of lengths is used to actuate a valve mechanism operated by compressed air.

Fig. 1 shows the general scheme for a case in which a tank of water is to be maintained at a uniform temperature. If left to itself it would cool off, but it is kept at the desired temperature by injecting steam. The amount of steam which is admitted at any moment is regulated so as to keep the temperature constant; and the mechanism of the valve which regulates the supply of steam is operated by compressed air, and the air valve is actuated by the two-metallic expansion stems, as will be described below.

A is the water tank, *B* is the pipe for the steam which heats the water. *C* is the valve in this pipe which regulates the amount of steam which is passed at any moment into the water. This valve is actuated by compressed air from the supply line *D*. The air-valve mechanism which regulates the pressure of

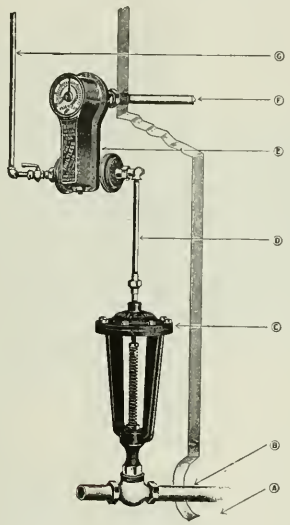


FIG. 1.—TEMPERATURE CONTROLLER.

the air for the valve is shown at *E*, while *F* is the expansion stem of the controller, which expands or contracts more or less according to the temperature of the water in the tank. It supplies the movement for operating the valve mechanism of the controller. *G* is the pipe line which carries the initial supply of compressed air which is required as auxiliary motive force.

In case the temperature of the water would for some reason rise slightly beyond the desired temperature on which the controller is set, then the expansion stem *F* of the controller will expand, and its slight movement will be greatly multiplied by the lever mechanism of the controller and will then be transmitted to the air valve mechanism which now will admit more compressed air to the valve, so that the valve will now admit less steam to the water tank, and so on.

Fig. 2 shows the interior mechanism of the controller with the expansion stem. *J* is the inner rod of the expansion stem, *K* the outer tube part of the stem. The inner rod *J* is made of a material of zero coefficient of expansion; that is, the length of the inner rod remains always the same, whatever the temperature. On the other hand, the outer tube part *K* is made of a material with a high coefficient of expansion so that its length changes considerably with a change of temperature.

The controller is attached to the tank by means of the threaded hub *L* in such a way that the stem *K* extends unobstructedly into the apparatus.

The left-hand end of the inner rod *J* of the stem is constantly pressed by means of the spring *M* against the plug at the left-hand end of the tube *K*; hence when the outer tube *K* elongates or contracts then the inner stem *J* (which remains of constant length) is forced by the spring to move in unison.

To the right-hand end of the inner stem *J* is attached an extension rod *N* which is made of the same material as the expanding outer tube *K*. Hence, the inner and outer parts of the expansion stem to the right of the hub *L* are contracting and expanding equally, so that any changes in temperature outside of the apparatus, the temperature of which is to be controlled, do not affect the functioning of the controller.

It is, therefore, only the inner and outer parts of the expansion stem *to the left** of the hub *L* which are made of

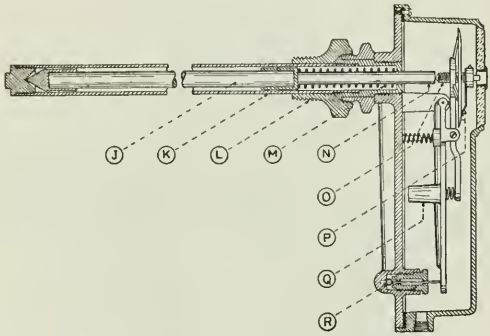


FIG. 2.—EXPANSION STEM AND CONTROLLER MECHANISM.

different materials and which are, therefore, affected unequally by a variation of temperature.

The extension *N* is held against the point of the adjusting screw *O*, the adjustment of which determines at which temperature the controller will function. To the adjusting screw *O* is attached the pointer *P* which indicates the adjustment on a dial beneath, facilitating close adjustments.

Q is the lever mechanism through the short end of which the adjusting screw *O* passes. At this point the lever is held by spring tension against the extension *N*.

The auxiliary motive force (in this case compressed air) is controlled by the valve mechanism *R*. This valve has three ports; one for the admission of the motive force (compressed air), another for supplying the diaphragm motor and another for the escape of the diaphragm-motor supply. The construction is shown on a larger scale in Fig. 3.

U is the inlet port through which the initial supply of motive force (compressed air) enters from the compressed-air supply line; the connections for the line being shown at *T*.

V is the valve-ball, which assumes a position in accordance with the position of the valve stem *X* against which the long end of the lever *Q* in Fig. 2 is held by spring tension, thus regulating the amount of compressed air passing through the ports.

W is the outlet-port connecting with *Y*, the outlet (if the position of the valve-ball *V* allows).

Y is the connection for the line carrying the motive force between the controller and the diaphragm motor.

Z is the waste-port, which provides an escape for the motive force (compressed air) when it is in excess of the amount needed to hold the diaphragm-motor valve open or shut to the desired extent.

*That is the parts within the apparatus the temperature of which is to be maintained constant.

We can now trace in detail, by means of Figs. 2 and 3, what happens if the water in our tank (which is to be held at constant temperature) shows a tendency to rise in temperature.

The temperature will rise, indeed, very slightly. But this will at once result in an elongation of the outer tube *K* of the expansion stem in Fig. 2. As *K* is fixed to the tank at the connection *L*, the left-hand end of *K* will move away from *L* to the left. The inner rod *J* is not affected in its length by the change of temperature and is therefore caused by the spring *M* to move along to the left. The extension *N* moves in unison and consequently the upper arm of the lever *O* moves also, on account of the spring tension which keeps *O* constantly in touch with *N*.

Since the one arm of the lever *Q* moves one way (to the left in Fig. 2), the other arm moves in the opposite direction (to the right in Fig. 2), a pivot being arranged between the two ends of the lever.

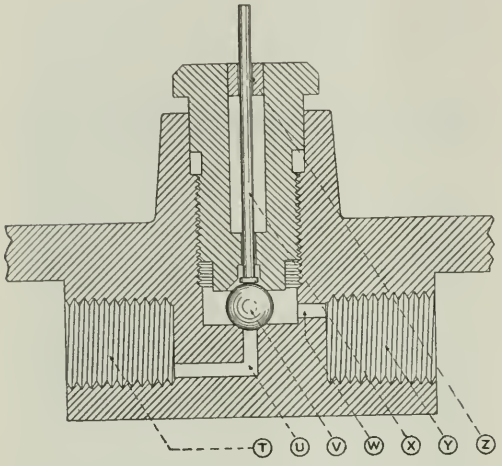


FIG. 3.—VALVE MECHANISM.

Consequently the valve stem *X* will now move also (upward in Fig. 3). The valve-ball *V* being pressed constantly against the head of *X* by the compressed air supply will now also move upward so that more air can enter through the inlet *U* and pass through the port *W* and connections *Y* to the diaphragm motor. Thus a greater pressure of air is supplied to the diaphragm motor, which thereby closes the steam valve to a greater extent and curtails the steam supply to the tank.

Should, however, the water in the tank show a tendency to cool off so that more steam is required, the controller will cause the diaphragm-motor valve to open further (thus letting in more steam) by functioning in just the opposite manner from that above described.

In practice the diaphragm-motor valve is rarely wide open or dead shut. Usually it is "throttling"—the controller being so sensitive to the slightest change in temperature within the apparatus to which the controller is attached that the diaphragm-motor valve is constantly "throttling"—letting in just a little more or just a little less steam, as required to keep the temperature constant.

The above-described design and operation permit of various modifications. Compressed air was assumed as the motive power and is generally best suited for this purpose. The initial supply should be of at least 15 lb. per square inch pressure, but the volume needed is comparatively slight.

The controller may also be built in some cases, however, for operation by water (instead of compressed air) as motive power. But when water is used the construction of the valve mechanism is somewhat different, although the principle is the same.

In the above example the controller mechanism was so arranged that when the stem of the instrument is subjected to a higher temperature the diaphragm motor closes the valve which controls the heat supply. This is the "direct-acting" type of controller.

In other cases in practice it may, however, be required that a rise in temperature be checked by turning on a greater supply of a cooling madium. In such cases the "reverse-acting" type of controller is used. In this type the controller mechanism is so arranged that when the controller stem is subjected to a higher temperature the diaphragm motor opens the valve further.

The many applications of this controller for industrial work, especially in the chemical industries, are so evident that only a few typical applications may be mentioned.

In the case of air to be heated by steam coils to a given temperature the controller stem is placed in the air duct (in which the temperature is to be adjusted) and the diaphragm-motor valve adjusts the steam supply to the steam coils.

If the temperature of a heated air in a closed chamber (an incubator, dryer, etc.) is to be maintained constant the controller may be used with both diaphragm-motor valve and diaphragm-motor lever. In case the temperature tends to rise the diaphragm-motor valve decreases the steam supply for the heating coils, while the diaphragm-motor lever opens the damper for the air supply to a greater extent.

For this controller, which is one of the latest Roesch-designed devices, the following principal points of advantage are claimed:

The inner rod of the stem of the controller is of a material which has a zero coefficient of expansion. If the inner stem has even a slight coefficient of expansion (though much less than the outer portion) the action of the controller would not be changed in principle, but in practice the controller would show the troublesome phenomenon of "creeping" due to the fact that the functioning point of the controller varies as the temperature effect slowly permeates the internal part of the stem. It would, therefore, take such a controller quite a while to settle down. With the above-described controller, with its internal stem remaining rigidly of constant length, all this trouble is avoided.

There is but one valve mechanism in the working parts of this controller, and this valve mechanism is also self-cleansing and self-adjusting. Since the proper working of the device does not depend upon a number of parts being in fine adjustment in relation to each other the effects of wear and tear are greatly reduced.

The cover which incloses the working parts of the controller can be removed while the controller is in use, and thus the working parts be exposed while in operation, thus permitting easy inspection.

No stuffing box arrangements are used in the mechanism. Further, that part of the mechanism which multiplies and transfers the movement of the expansion stem to the valve mechanism is never in contact with the compressed air supply.

Automatic Temperature Control in Electric Ovens.

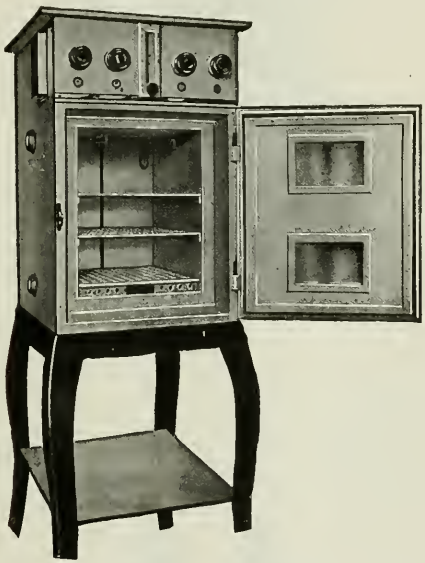
For use in research as well as industrial laboratories the easy and exact temperature control which is possible with electric furnaces is one of their greatest advantages. Usually this temperature control is obtained by some sort of rheostat which is adjusted by hand.

For many purposes it would, however, be greatly advantageous to have automatic temperature control, but the problem in general meets with great difficulties. For the restricted field of electrically-heated ovens with temperatures from that of air to 175° C. (347° F.) the Freas oven, which has just been placed on the market by Messrs. Eimer & Amend, of New

York City, has solved the problem. Even in the restricted field, indicated by the above given temperature limits, the exact automatic temperature control should prove exceedingly useful for many purposes.

The temperature-regulating device in the Freas oven is entirely of metal and its action depends on the expansion of a metallic tube running through the chamber from top to bottom and operating a lever which makes and breaks a contact, proper means being provided to prevent arcing. The lever is extended to serve as an indicator operating up and down a temperature scale on the outside of the oven near its top. By turning the screw at the bottom of this temperature scale the indicator is moved to the desired temperature; this is all that is required of the operator and the regulating device does the rest in maintaining the temperature to which the furnace is set.

The oven, as shown in the adjoining illustration, is built of asbestos wood, which has the triple advantage of being fireproof, a good heat insulator, and not attacked by acids or alkalies. The oven is fitted with one or two shelves, perforated with small holes to distribute the heat uniformly



ELECTRIC OVEN, WITH AUTOMATIC TEMPERATURE CONTROL.

throughout the chamber. This insures uniform temperature, there being less than one degree difference between the top and bottom.

The door is fitted with a mica window, so that parts of the chamber can be inspected without opening the door. Complete ventilation to get rid of fumes and moisture from the substance being dried is obtained by two openings on each side of the oven and two through the top.

The heating resistors are placed at the bottom of the oven and consist of wire coils so insulated that they are not injured by acid fumes, etc.

The furnace can be operated on alternating current as well as on direct-current circuits. The power consumption is 660 watts, which for the lower temperatures can be reduced one-half by means of a switch, provided the closest regulation is desired.

The regulator at any temperature between air and 175° C. is sharp and accurate to less than one degree and remains so indefinitely, while under most favorable conditions it is accurate to one-tenth degree. For continuous operation at such high accuracy the oven is equipped with an extra fan, driven by a motor.

To heat the furnace up to 50° C., about 15 minutes' time is required with maximum current, and 45 minutes to heat it up to 110° C.

A Liquid Flow Recorder.

Liquid flow recorders have become a necessity in recent years chiefly for two kinds of work; firstly in large power stations for the measurement of condensed steam and boiler feed-water, and secondly in chemical and metallurgical works for recording the rate of flow and the quantities of acids and solutions in operation. There can be no doubt that especially in the latter field their use will rapidly increase, when the practical reliability of modern flow recorders is properly understood. In the following we give a description of the Lea liquid flow recorder, which was first developed in England and is placed on the market in this country by the Yarnall-Waring Company, 1109 Locust Street, Philadelphia, Pa.

Liquid flow recorders are essentially based on the laws governing the flow of water over wires and notches. The Lea flow recorder employs for this purpose a sharp-edged V notch to which the formula of Prof. James Thompson applies, according to which the flow in cubic feet per minute through a 90-deg. notch is $0.305H^2VH$, where H is the height of the notch in inches.

For simplicity's sake, let us assume that water is the liquid, the flow of which is to be measured. Then it will be easy to plot a "curve of flow"; that is, a diagram which gives the rate of flow of water for different depths H of the water according to Thompson's formula. If now a float is placed in the water, the rise and fall of this float may be connected to an indicating or recording device, such as a recording pen, through an intermediary mechanism in such a manner that the recording device moves at any instant in direct proportion to the rate of flow of the water in that instant, and therefore plots on a revolving drum a curve which is directly a continuous record of the flow of water during any length of time. Clearly the problem is to devise a suitable and exact intermediary mechanism, as mentioned above. This problem is solved in the Lea recorder in an ingenious manner in the following way:

The float in the water rises and falls with the water level. Above it is the instrument case, and the float spindle passes through the bottom of the instrument case upwards into the latter. Upon this float spindle a rack is cut, which gears into a small pinion upon the axis of a drum, revolving between centers.

Upon the body of the drum is a screw thread, the contour of which is the "curve of flow" (as defined above) for the V notch in connection with which the recorder is used, and just as the flow through the notch increases rapidly with the depth, so the pitch of the screw increases in the correct proportion.

Above this drum is a horizontal slider bar, supported on small pivoted rollers, and carrying an arm, at the upper end of which is a pen or pencil point in contact with a paper chart, upon a clock-driven recording drum, which revolves once in twenty-four hours.

As the float rises or falls, the drum spiral is rotated, and its motion is imparted to the slider bar (and pen arm), by means of a saddle arm, projecting from the latter and engaging at its lower end with the screw thread.

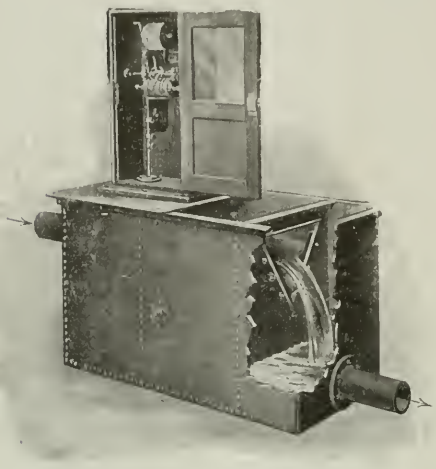
On the body of the spiral drum is a graduated scale, adjacent to the screw thread, and computed in pounds per hour. It is thus seen that the spiral drum serves a double purpose:

Firstly, its screw thread rectifies the motion of the pen, so that it moves equal distances for equal increments in the rate of flow.

Secondly, it provides a magnified scale for making an accurate observation of the rate of flow at any moment.

Thus the actual depth of water in the notch can be observed at any time. The instantaneous rate of flow, in pounds per minute or gallons per minute, can be read off directly. Finally the recording pen which moves in direct proportion to the rate of flow plots a diagram whose area is a measure of the total quantity of liquid passed.

The adjoining illustration shows the lower part of the apparatus in which the water acting on the float passes through



LIQUID FLOW RECORDER.

the V notch, and the upper instrument case with the vertical float spindle and the record drum. The illustration is otherwise self-explanatory.

When the flow to be measured is so small that it would show only a depth of 1 or 2 in. in a 90-deg. V notch, notches having angles less than 90 deg. may be used, like 45-deg., 22.5 deg., etc. Speaking generally, the flow through such notches is approximately proportional to their width, for a given depth.

Hence by means of these acute notches, a large float movement is obtained when only a small quantity is being measured.

Further, a measuring tank and recorder having a maximum capacity of say, 10,000 gal. per hour through a 90-deg. notch, if supplied with two other notch plates—say 1 of 45 deg. and 1 of 22.5 deg.—can be used for measuring either 10,000, 5000 or 2500 gal. per hour.

Moreover, one recorder may be used for measuring large or small quantities of liquids by providing several identical notches, of which a greater or smaller number may be closed according to the requirements. With such a multiple-notch arrangement the total flow at any instant is obtained by multiplying the reading on the recorder by the number of notches open.

It is clear that in principle the operation of the apparatus is exactly the same whether it is used for measuring the flow of water or of acids or any other liquids. But for the use with corrosive liquids it is all-important that nothing but the float is in contact with the liquid, the instrument case itself being above the flow arrangement, out of contact with the liquid.

In the case of a sulphuric acid solution, for instance, the lower tank through which the acid passes, may be made of lead-lined wood, and the notch also of lead. But glass may be used equally well.

Another point which is important for certain chemical operations, is that a float may be placed above the weir to control the inlet, thereby insuring a constant head of the liquid. This is especially valuable in manufacturing processes where a definite percentage of solution has to be added.

There seems no doubt that these liquid flow recorders and adjusters can be of considerable service in large-scale chemical operation.

Carboy Rocker.

The carboy rocker, shown in the adjoining illustrations, is so handy that a brief description of its construction and method of use should be interesting to all users or bottlers of acids, ammonia, etc., from carboys. It is made by Messrs. Rockhill & Vietor, 114 John Street, New York City.

FIG. 1.—ATTACHING ROCKER TO CARBOY.

This carboy rocker is attached to a carboy or detached from it in one-fourth of a minute. One man can roll a full carboy to any desired position on the floor and completely empty it without spattering. In use the carboy can be left resting on the first or second stop, as there is no need of placing it back. It is ready for immediate use.

Fig. 1 shows the method of attaching the rocker to the

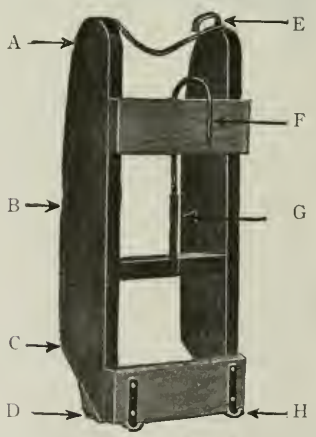


FIG. 2.—STOP CARBOY ROCKER AND VENT TUBE, DETACHED.

carboy. The workman holds the rocker with the right hand by the curved, pointed, sliding hook. He then, with his left

hand, takes hold of the cleat on the side of the carboy, lifts one side of the carboy sufficiently to catch the pointed supporting hooks underneath it, pulls the curved sliding hook up and over and presses it down on top of the rocker. He then removes the stopper of the carboy and inserts the celluloid vent tube, closing the outer end of the tube with the forefinger while inserting it.

Fig. 2 shows the construction of the rocker and vent tube detached: *a* is the last position; this point is in contact with the floor when the carboy is empty; *b* is the middle stop; this point is in contact with the floor when the carboy is half empty; *c* is the first stop; the carboy remains stationary when the stop touches the floor.

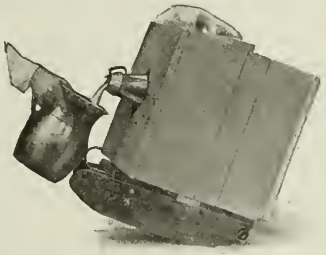


FIG. 3.—FIRST PITCHER FULL.

The wheels *d* facilitate moving. The celluloid vent tube, ready to insert in carboy, is shown at *e;* attention should be called to the fact that it is made of celluloid; *f* is a pointed steel sliding hook; *g* a spiral steel spring, and *h* are supporting hooks.

Fig. 3 shows the position of rocker and carboy when the first pitcherful is taken.

Notes.

Erratum.—The article of Prof. J. W. Richards in our September issue (p. 463) mentions 250 *million* tons of Lake Superior ore running 35 to 50 per cent of iron; this should have been 250 *billion* tons, i. e. 250,000 millions.

The **Scientific Materials Company,** of Pittsburgh, Pa., have recently enlarged their glass-blowing department, increased their force, and added the latest and approved machinery. This department is now in charge of an expert German glass blower, who has made a specialty of intricate laboratory apparatus. An expert engraver takes charge of the graduating of the most precise volumetric apparatus. Special glass apparatus according to drawings may be obtained on short notice, and broken apparatus will be repaired and returned promptly.

The **Zaremba Company,** manufacturers of evaporators and chemical machinery, with main offices in Buffalo, have opened a branch office in New York, Room 95, 97 Liberty Street.

The **Brown Instrument Company,** the first concern to manufacture pyrometers in the United States and whose office and laboratory have been located at 311 Walnut Street, Philadelphia, since 1860, has been forced to move to larger quarters by the great increase in its business. As electrical instruments play an important part in the construction of pyrometers, it was found desirable to be able to superintend the construction of milli-voltmeters for use in their pyrometers. With this end in view, Mr. Brown, the president of the Brown Instrument Company, has now also become president of the Keystone Electrical Instrument Company, of Philadelphia, which, for a number of years, has been manufacturing voltmeters and ammeters, and the shops of the Brown Instrument Company are being combined with those of the Keystone Electrical Instrument Company at Ninth Street and Montgomery Avenue, Philadelphia. This places the Brown Instrument Company in the unique position of controlling the construction of their milli-voltmeters for their pyrometers, even to the manufacture of the raw materials, such as the magnets. By this arrangement the Brown Instrument Company and the Keystone Electrical Instrument Company will manufacture pyrometers, thermometers, speed indicators, draft gages, recording gages, voltmeters and ammeters. In fact, they are now in a position to manufacture or repair practically any type of instrument. In addition to the works and office at Ninth Street and Montgomery Avenue, Philadelphia, an office will also be maintained for the present at 311 Walnut Street, and the branch offices of the Brown Instrument Company at Pittsburgh and Chicago will also be continued.

Carbon Electrode Factory at Niagara Falls.—The new carbon plant which the National Carbon Company, of Cleveland, Ohio, is building at Niagara Falls will be ready for occupancy some time in October. The company has been working for about a year in building this new branch. It will be a thoroughly modern factory in every respect and will engage in the manufacture of carbon electrodes exclusively. This is the second plant which the National Carbon Company has built during the past year, the other being at Toronto. In nearly every one of their nine factories considerable improvement and expansion has been made during this year. The main plant and general offices of the National Carbon Company are located at Cleveland. This plant alone covers about 30 acres of ground, and is engaged in the manufacture of a miscellaneous line of carbon products. The eight outside plants manufacture a special line of product for which the locality, equipment and organization is best adapted. The Cleveland plant is by far the largest industrial carbon factory in the world, while the total facilities of the nine branches make a tremendous organization for the development and manufacture of a complete line of high-class carbon products.

The production of secondary metals, meaning those recovered from scrap metal, drosses, sweepings, etc., amounted to a considerable industry in the United States in 1910, according to figures compiled by the U. S. Geological Survey. Of secondary copper, including that in alloys other than brass, there were produced 25,396 short tons; remelted brass, 27,347 tons; secondary lead, 17,822 tons; recovered lead in alloys, 23,327 tons; secondary spelter, 33,040 tons; recovered zinc in alloys other than brass, 1,181 tons; secondary tin, 2,423 tons; recovered tin in alloys, 3,092 tons; secondary antimony, 27 tons; recovered antimony in alloys, 1,529 tons. The total value of these metals and products was $45,525,500. The production of secondary copper amounted to 17 per cent. of the domestic consumption of new copper. The secondary lead was equal to 11.5 per cent. of the refined lead produced in the United States. The secondary zinc including that in brass equaled 23.2 per cent. of the total production as primary spelter in the United States. The secondary antimony and tin constituted practically the only domestic production of those metals.

Quicksilver in California.—In 1910 California produced 83.5 per cent. of the total output of that metal in the United States. The county leading in production is San Bernardino in which are the New Idria mines, at present the most important producers in the American hemisphere. Fifteen quicksilver mines contributed to the California output in 1910, which amounted to 17,211 flasks of 75 lb. each, valued at $800,484.

The Bureau of Mines has issued within the past month two technical papers; No. 3, Specifications for the Purchase of Fuel Oil for the Government, with directions for sampling oil and natural gas, and No. 4, the Electrical Section of the Bureau of Mines. Miners' Circular No. 4 describes the use and care of mine-rescue breathing apparatus.

PERSONAL

Mr. Louis D. Bastian, of Silverton, Colo., has been engaged as superintendent of the Vermillion mill near Animas Forks, Colo.

Mr. Raymond C. Benner, of the department of chemistry

at the University of Arizona, has gone to the University of Pittsburgh, where he will investigate the smoke problem with Professor Robert K. Duncan.

Mr. Gelasio Caetani, of San Francisco, has been spending some time in the Telluride district, Colorado, making tests and advertising improvements in the milling operations there.

Mr. R. S. Davis, chemist for the Primos Chemical Company, Newmire, Colo., has returned to his work after a short leave of absence.

Dr. Wilbur A. Hendryx, of New York, has gone to South Africa on business for the Hendryx Cyanide Machinery Company.

Dr. H. R. Low, of Platteville, Wis., recently spent several weeks in the San Juan district, Colorado, in the interests of the American Zinc Ore Separating Company.

Mr. A. J. McMillan, formerly managing director of the Le Roi Mining Company, and now closing up the affairs of that company, has returned to Rossland, B. C., after spending three months in England.

Mr. Charles L. Parsons, secretary of the American Chemical Society, and until recently professor of chemistry in the New Hampshire College, Durham, N. H., has been appointed chief chemist of the mineral division of the Bureau of Mines at Washington.

Prof. Robert H. Richards, of Boston, spent the month of September in Denver.

Mr. John E. Rothwell, of Denver, has just completed a tour of the principal western metallurgical centers.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore-Treatment (*Continued*).

399,209, March 5, 1889, Julio H. Rae, of San Francisco, California.

Gold and silver. Treats ore held in suspension by an agitator, the metal being deposited on the cathode. Several cells are shown, each consisting of a tub of wood lined near the bottom by a vertical cathode ring of copper, silver or silver-plated metal. An agitator is journaled on a vertical shaft extending through the bottom of the cell, having radial arms carrying stirrers, some of which are inclined outward to scrape the ore from the cathode ring and throw it to the center, while others are inclined outward. The anodes consist of a number of vertical carbon rods, arranged in two concentric circles and depending from a wood ring into the upper part of the ore-pulp. Each rod has a spring connection and is readily removable.

459,023, Sept. 8, 1891, Curt Schreiber and Hans Knutsen, of Broken Hill, New South Wales.

Gold, silver and antimony. Antimony ore is hand-picked into three classes containing more than 25 per cent of antimony, 5 to 25, and less than 5. Each lot is then crushed to pass a sieve of 900 meshes to the square inch, and is lixiviated with a solution containing sodium monosulfid, in the proportion of 235 parts to each 336 parts of antimony sulfid. Or barium or calcium sulfid may be used in molecular proportions. One cu. yd. of solution, say 200 gal., is sufficient for each ton of ore, the strength being varied according to the percentage of antimony therein. The three lyes from the three classes of ore are differently treated. Each lot of ore is leached with three or more solutions, first with solution used twice previously, then with solution used once, and finally with fresh solution. Strong solution is mixed with the weaker ones, and antimony sulfid is dissolved in the weakest ones. The resulting saturated solutions of antimony double salts are electrolyzed, the conductivity being increased by the addition of a small amount of sodium chlorid. The current is regulated according to the concentration of the lye. Antimony is deposted at the cathode, with production of sodium hydrosulfid which is oxidized at the anode to sodium disulfid. Sodium hydroxid is added from time to time to convert the sodium disulfid into monosulfid, which is returned to the lixiviators. The residual ore, freed from antimony, is amalgamated for the recovery of the gold or silver.

460,354, Sept. 29, 1891, Werner von Siemens, of Berlin, Germany, Assignor to Siemens & Halske, of same place.

Copper, etc. Various cells for carrying out the process of his patent No. 416,576. The first cell is a rectangular wooden pan, covered inside and out with lead, and having a perforated false bottom on which is supported the anode, of corrugated sheet-lead or corrugated and perforated plates of retort-carbon. Lying on the anode is a screen consisting of two spaced-apart sheets of fine wire-gauze, to prevent agitation of the liquid at the anode. The cathode consists of a number of horizontal cylinders of wood coated with wax and then with metal, submerged in the electrolyte and slowly rotated. The solution, containing a salt of the metal to be deposited, a ferrous salt and free acid, is supplied at the top of the cell, and passes from the cathode, where two-thirds of its metallic contents are deposited, down to the anode where the ferrous salts are reoxidized, being drawn off at the bottom through a syphon. The second cell consists of a semicircular or V-shaped trough, its inner surface lined with an anode having small transverse grooves. The cathode is a revoluble cylinder, as before, separted from the anode by the double wire-gauze screen. The solution enters at the top and the resulting ferrous salts are reoxidized, as before, oxidation beginning at the edges of the grooves and being completed at their bottom. The third cell is a circular pan of hard lead, on the bottom of which are supported a number of short vertical carbon rods, or a grate of horizontal carbon rods, constituting the anode. Or the anode may consist of grooved carbon plates. Above the anode is a sheet of wire-gauze, and above this is a horizontal rotating disk cathode of metal. The solution passes downward as before. The motion of the cathodes is necessary for the production of good copper and the use of a high current density.

473,105, April 19, 1892, George J. Atkins, of London, England.

Gold, silver, etc. The first cell illustrated comprises two vertical cylindrical portions, subdivided by a diaphragm of matting, flannel, felt or a porous plate, mounted to slide vertically in grooves. In and nearly filling the anode chamber is an anode comprising a shaft, e.g., of wood or protected metal, carrying a broad-bladed screw of carbon or lead. The cathode compartment contains a revolving cylinder of perforated sheet-copper. Flexible rubber scrapers project from the walls of the compartment against the surface of the cathode. The cell is filled with a solution of sodium chlorid, sodium sulfate, potassius cyanide, or potassium ferro-cyanide. Ore is fed into the top of the anode compartment and descends downward in contact with the screw. The screw may be rotated to accelerate or retard the descent of ore, the contact being sufficient to effect the desired oxidation or chlorination. The solution is re-pumped from the bottom of the anode compartment to the top. The ore, freed from base metals, generally silver and sometimes gold, is withdrawn from the bottom of the anode compartment. If the electrolyte liberates chlorin or other solvent of gold, this is generally reprecipitated by a base metal in solution, such as iron. The metals loosely deposited on the cathode are scraped off and removed below. The gold is recovered by treating the purified ore in an amalgamator. A modified cell consists of a vertical copper cylinder, forming the cathode, containing and separated from the anode screw by a diaphragm of canvas, flannel or felt

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The Abrogation of the Hill Iron Ore Lease.

There are two reasons, each amply sufficient in itself, for the abrogation by the United States Steel Corporation of its lease upon the Great Northern or James J. Hill iron ore lands, the one being that the operation does not promise, from the present industrial position, to be profitable to the Steel Corporation, the other that the lease stands open to question under the Sherman law.

As to the extent of the ores, it may be said that there has come to be reasonable assurance that more than 300,000,000 tons will be found, of fair quality. None of the ore is of good quality, judged by the standard of other ores being mined. Practically all requires washing. The Colerine washer, installed by the Steel Corporation to treat the first of the Hill ores mined, cost to the end of 1910 the tidy sum of $1,461,584.33, established to wash practically the first of the nearly 250,000,000 tons of ore which has thus far been taken out of the Mesabi range. The analyses of the Steel Corporation's Mesabi ores in 1910 were as follows as to iron content:

	NATURAL.	DRIED AT 212 DEG.
Group No. 1	52.9201	60.2171
Group No. 1 (high Mn)	50.0105	56.6986
Group No. 2	54.4243	60.9739
Group No. 3	49.5229	57.0558
Group No. 4	47.2340	56.0731
Group No. 5 (D. M. & N.)	50.4048	56.7034
Group No. 5 (G. N.)	51.5741	57.7271
Group No. 7	46.4202	53.4499

Group No. 5, Great Northern Railway, may be taken as fairly typical of the best of the Hill ore. The royalty rate in the Hill lease is 85 cents for the year 1907, upon ore analyzing 59 per cent metallic iron when dried at 212 deg. The custom in the ore market is to sell ore upon a guarantee in the natural state, the base guarantee being 55 per cent for Bessemer and 51.50 per cent for non-Bessemer. The Hill lease rate advances and declines by 4.82 cents per unit for ore above and below 59 per cent, except that a flat rate of 30 cents is placed upon ore below 49 per cent, that being regarded as the limit for ordinary merchantable ore. For each year after 1907 the royalty rate advances by 4 per cent of the 1907 rate for the ore of iron content involved. The minimum to be mined (and paid for if not mined on the assumption that it would have been 59 per cent ore) is 750,000 tons in 1907 and 750,000 tons more for each succeeding year until 8,250,000 tons is reached in 1917, the minimum remaining at that rate thereafter. A quantity of ore, whether of Hill or other ore, equal to the year's minimum is to be transported over the Hill road, the Great Northern, at the rate of 80 cents, which has been the fixed rate from Mesabi range to Lake Superior ports for years.

Should the open rate decline, the reduced rate is to be paid the railroad, but the royalty rate is to be correspondingly increased, royalty payments being made to the three trustees who act for the holders of "certificates of beneficial interest" distributed in 1907, certificate per share, to the stockholders of the Great Northern Railway Company.

The lease provided that the Steel Corporation could terminate it upon Jan. 1, 1915, provided it gave notice not later than Jan. 1, 1913. Our understanding is that the Steel Corporation purposes to take advantage of this clause, but that, while nothing definite has been done thus far, there is opportunity for negotiation with the Hill trustees, and not impossibly with the aid of the Department of Justice, looking toward an earlier termination.

The general impression seems to be that none of the Hill ores will be found to equal or exceed the base rate of 59 per cent metallic iron, dried at 212 deg., that only a relatively small proportion will fall within a couple of units of that iron content, and that the bulk will fall considerably below it. This alignment operates to depreciate very seriously the apparent value of the Hill ores, considered as an 85-cent royalty mining proposition, because the penalty rate of 4.82 cents per unit does not cover the increased cost of mining, transporting, and smelting the lower grade ores. It is, indeed, very much smaller than is represented in the trade custom in selling ore as that custom is to compute the unit price by taking the price at lower lake dock and adding a 60-cent arbitrary (assumed as the average inland freight rate), making the unit value at the present season's price on Mesabi non-Bessemer ore 7.96116 cents, and ores below 49 per cent iron content, natural state, are subject to penalties at greater than the straight unit rate. The season's price is $3.50 for ore containing 51.50 per cent iron, natural state. Thus the lease is a less desirable one than were the ores actually to be mined rich enough to involve the payment of the base rate.

During its entire life the lease involves a constantly increasing royalty rate, at 4 per cent of the 1907 rate, equal to an annual increment of 3.4 cents on the 85-cent rate. For ten years it involves an annual increase in the minimum to be mined. In 1917 the minimum would be 8,250,000 tons, at $1.19 for base ore, equal to a payment of $9,817,500. Very onerous also is the provision that the ore or an equivalent tonnage shall be carried over the Great Northern road, at 80 cents a ton, for the Corporation has two roads of its own, and it is common report that the service is performed at a cost of not more than 40 cents a ton, so that here is a direct loss, 40 cents on 8,250,000 tons, of $3,300,000 additional. In discussing this phase of the lease reference is frequently made to the possibility of the open rate of 80 cents being reduced. Such reduction would, however, neither increase nor decrease the Corporation's loss through the operation of this lease. In computing the advantages or disadvantages the payment stands in relation to the cost at which the Corporation could perform the service of carrying its ores on its own roads, and not in relation to the open rate charged to outsiders.

It is the common impression that the Steel Corporation entered upon the Hill lease for the direct purpose of preventing these ore deposits from falling under the control of existing or prospective competitors. In that sense, of course, the lease may possibly be interpreted as having been in restraint of trade and perhaps under the ban of the Sherman law. It may be observed that many of the features of the United States Steel Corporation, viewed in the light of a "trust," are in line with those modern developments which are held to represent progress toward a beneficial form of socialism, but it is difficult to see how the Hill lease represented such progress. It would seem that the value of the lease to the lessors was created by the desire of the Steel Corporation to control the properties. There was a distribution, regarded purely as a gift, of 1,500,000 "certificates of beneficial interest" to shareholders of the Great Northern Railway Company, and the stock market value of these certificates at one time rose to above $80, equal to $120,000,000 for the entire issue, a valuation which had dropped to about $75,000,000 prior to the announcement that the lease would be terminated. So it would appear that this value, if it may be regarded as a value, was created chiefly by an effort to monopolize, and shrewd lawyers' logic would be required to defend it as representing progress toward a better form of conducting industry.

Aside from this feature, however, it is pertinent to observe that no evidence has appeared of competition being restricted. The pig iron capacity of the United States in 1906, the year in which the lease agreement was made, was 25,000,000 tons per year. Partly through new erection by the United States Steel Corporation, but largely through new erection by outside interests, the capacity has been increased to between 33,000,000 and 34,000,000 tons at the present writing, with furnaces in course of construction sufficient to make at least 35,000,000 tons total capacity. Present demand and production, however, are at the rate of only about 25,000,000 tons. The excess is such as to preclude any important influence upon prices of pig iron or finished steel products through the existence of this lease, as compared with the condition which would have existed to-day had the Steel Corporation allowed the Hill ore lands to remain upon the market. The Lake Superior region is developed as matters stand to such an extent that it can easily ship 50,000,000 tons of ore, and quite possibly not far from 60,000,000 tons in a season whereas this season's actual shipments promise to fall below 35,000,000 tons, and no season has yet shown a movement of 44,000,000 tons.

The burden of the Hill lease would have been less onerous, through distribution over a larger tonnage, if the Steel Corporation's operations had grown as was expected in 1906. Its capacity has grown in the intervening five years, but its actual output this year is not 10 per cent in excess of that of 1906. Doubtless the expectation in 1906 was that its productive requirements would be doubled by 1917, whereupon the ore required to be mined in that year under the Hill lease would have been less than 20 per cent of the total.

On account of the annual increment in the minimum tonnage until 1917 and the indefinitely continued increment in the royalty rate, the Steel Corporation relieves itself of a large part of the total burden by a discontinuance upon Jan. 1, 1915. We may compute roughly that the total royalty payments in the period 1907 to 1914, inclusive, amount, at the nominal rates, to $27,234,000, and the loss in freight payment, at 40 cents a ton, to an additional $10,800,000, making about $38,000,000. Royalties, though much smaller, would have to be

paid elsewhere if this ore were not mined, but, on the other hand, the development and mining of the Hill ore during this eight-year period are presumably more expensive than the cost would have been elsewhere with continued mining promised. The abrogation of the lease may be effected for an earlier date by arrangements which may yet be perfected. In any event the Steel Corporation will find itself in better condition from a business standpoint, while it is fairly clear in view of the attitude of the Government that it will find itself in a much more comfortable position from a legal standpoint.

The Power-Factor of Electric-Furnace Circuits.

Managers of central stations using an alternating-current system are unpleasantly familiar with the word "power-factor." It connotes to their minds the necessity of buying copper for their generators, distributing mains and transformers to carry the "wattless current" that does no useful service.

In electric furnace plants which are generally operated by alternating current the power-factor must be taken into consideration, and it is especially important to look at the power-factor problem from the standpoint of future increased size of furnace units. In small furnaces such as those which are used for experimental work, say, of 25 kw to 100 kw, the power-factor is high, being from 96 per cent in the first named capacity to 85 per cent to 90 per cent in the second named capacity. But as the wattage rating of furnaces increases, other things being equal, the power-factor decreases, hence the wattless current increases. Consequently we have an untoward state of affairs, quite unlike the case with thermal economy, which increases with the size of the furnace.

An electric furnace has been well called "a short-circuit under control." It has a very low ohmic resistance. The current flow is opposed by the magnetic "reactance" as well as the ohmic resistance. The power in a direct current is equal to the product of the volts and amperes. With an alternating current this product is spoken of as the "apparent watts" or when divided by 1000 as kilovolt-amperes and not as kilowatts. The reading of the wattmeter gives the "actual watts." The ratio of the "actual watts" to the "apparent watts" (which is unity under ideal conditions and otherwise always smaller than unity) is called the power-factor of the circuit.

In an alternating current, therefore, Ohm's law in its simplest form does not apply and a modified form must be employed. Here the amperes are equal to the volts divided by the "impedance." The impedance is equal to the geometrical sum of the resistance and the reactance. The "reactance" can be described as that physical quantity which opposes the current flow by virtue of the electromagnetic properties of the circuit. Now these very electromagnetic properties of the circuit or more properly speaking the "self-inductance" are proportional (1) to the ampere-turns, (2) the number of lines of force that are cut in unit time by circuit and (3) to the frequency.

Let us suppose that the third of these, the frequency, remains constant as it does in any commercial plant and that while the electric furnace increases in size the voltage remains constant; then the amperes and consequently the ampere-turns increase. Furthermore, the number of lines of force increase because they are generated by the electric current in the iron work that encases the firebrick of the furnace. It is plain, therefore, that the "reactance" increases with the increase in size of the electric furnace. However, the resistance decreases with increase in size. Both the decrease of resistance and the increase of reactance tend to reduce the power-factor and therefore to increase the wattless current. Let us consider two electric furnaces of 50 kw and 100 kw capacities respectively. If the emf is 100 volts and there is a power-factor of 100 per cent, the current of the first is 500 amp and the second 1000 amp. The resistances are 0.2 ohm and 0.1 ohm respectively. If the smaller furnace had a power-factor of 96 per cent at 60 cycles it must needs have a current of 521 amp of which only 500 would be useful. But even if the larger furnace were carefully designed its power-factor might be as low as 80 per cent and so it would require a current of 1250 amp, of which only 1000 amp are useful. If the design were careless the power-factor might drop as low as 50 per cent. The current would need be 2000 amp. to produce 1000 amp. of useful current.

Let us interpret this in commercial language. Let the electrometallurgist who utilizes only 1000 amp. picture to himself the busbars, cables and secondary windings of the transformer to carry 2000 amp. and also the increase in copper necessary to carry the primary current and convert this into dollars and cents. The picture is not pleasing, especially when large capacities, say upward of 2000 kw, are needed.

There are several ways to mitigate this evil by proper design. One possible method is to reduce the frequency. But this means electrical machinery of special design and increased cost.

A more practical way (for usually the electric installation is already placed) is to increase the ohmic resistance by change of design of the furnace. Such a procedure results in the increase of the working voltage and of the power factor.

A third way is to increase the magnetic reluctance of the iron work of the furnace by change of design. For instance, it is possible to use large firebrick block 18 in. x 12 in. x 8 in. instead of the small 9-in. x 4½-in. x 2½-in. firebrick backed up by iron plates so often and so conveniently used. With this iron eliminated the magnetic circuit has a greater reluctance and so the reactance is diminished. It might be even possible to have the buck-staves, tie rods and facing plates made of some non-magnetic special steel. All these possibilities must be considered in designing a 1000-kw electric furnace.

We are glad to call the attention of our readers to this most interesting aspect of electric-furnace design. In this twentieth century it is necessary to attain the maximum of commercial efficiency. When viewed from the proper standpoint commercialism has a broad and ideal meaning and a sense quite different from that used by the poor muckrakers. There must be the application of the pure sciences such as chemistry, physics, and economies in the profession of metallurgy. These must be coupled with the use of arts. We have then a pleasing combination of the "knowledge why" and the "knowledge how." It is plain that this combination is needed in electric-furnace developments. We are especially glad to call attention in these columns to the subject of the power-factor of electric furnaces at a time when the electric furnace is invading the metallurgy of iron, steel, copper, lead and brass. For it is in these columns that all the first achievements of electric furnaces have been recorded, and electric-furnace pioneers have always been our readers, contributors, and friends.

The San Francisco Meeting of the American Institute of Mining Engineers.

(*Special Correspondence.*)

The Pacific coast has seldom offered such attractions as it spread before the mining engineers at this meeting. The meeting is, in reality, a fête, a festival of good things.

Starting from Chicago by special train on Sept. 30, the Eastern members, gathering in numbers as they sped westward, paid their first respects to Western scenery by halting two days at the Grand Canyon of the Colorado. The first day was perfectly clear, and the marvelous depths and details of the canyon stood out clear-cut and distinct; the second day clouds and mists, with here and there a rift and patch of sunshine, showed the great gorge in its most playful mood, and, to many, in its most artistic setting.

Los Angeles was reached on the afternoon of Oct. 5, and then followed a strenuous day and a half of unceasing pleasure and unstinted hospitality from the local members. Trolley ride to Pasadena, lunch on the ocean pier at Venice, fish dinner at Redondo and banquet at the Los Angeles Club were incidents of these days, while the local oil field at Bakersfield was also visited. Mr. Mullholland, of the Los Angeles Aqueduct staff, gave a fine illustrated lecture on this gigantic scheme, and its power possibilities. Mr. Comstock, head of the committee on local arrangements, proved a wonderfully efficient manager. A day at Santa Barbara and one on the cliffs at Del Monte or off to Santa Catalina island completed the sojourn in southern California.

Reinforced by the southern California members, the expedition reached the city by the Golden Gate on the morning of the 10th. Here the serious and formal work of the meeting began. The afternoon of the 10th and the mornings of the 11th and 12th were given over to sessions for reading and discussion of papers, sessions more than ordinarily interesting for variety and value of papers presented and vitality of discussion. The full program of these sessions was as follows:

Tuesday Afternoon, Oct. 10.

Chase, C. A.—"Notes on the Liberty Bell Mine."

Aldrich, Jr., T. H.—"Electrolytic Oxygen in Cyanide Solution."

Young, Geo. T.—"Slime Filtration."

Hall, E. J.—"A Modification of the 'Gay Lussac' Method for Silver Bullion Containing Tin."

Durham, E. B.—"Electrolytic Refining at the U. S. Mint, San Francisco."

Daniels, J.—"The Fritz and Coxe Laboratories of Lehigh University."

Bahney, L. W.—"Rapid Estimation of Available Calcium in Lime Used in the Cyanide Process."

Lass, W. P.—"Cyanide Plant at the Treadwell Mines, Alaska."

Christy, S. B.—"Progress in Electro-Amalgamation."

Wednesday Morning, Oct. 11.

Requa, Mark L.—"California Oil."

Janin, Chas. (Introduced by H. Foster Bain)—"Present-Day Problems in Dredging."

Yale, Chas. G.—"Gold Production of California."

Reinholt, Oscar H.—"Fuel Problems of the Pacific."

Reinholt, Oscar H.—"Government Coal Mines in the Philippines."

Dilworth, J. B.—"The Black Mountain Coal District, Kentucky."

Discussion by J. Langdon.

Locke, Augustus—"Notes on the Geological Structure of Tonopah, Nev."

Catlett, Charles—Phosphorus in Coking Coal."

Porter, J. J.—"National Valuation and Quality Efficiency of Furnace Stock."

Thursday Morning, Oct. 12.

Nisho, K.—"Historical Sketch of the Mining Industry in Japan."

Brunton, D. W.—"The Laramie Tunnel.

Locke, Augustus.—"Classification of Ore Deposits for Miners."

Stevens, Blamey.—"The Laws of Igneous Emanation Pressure."

Stevens, Blamey.—"Physical Data of Igneous Emanation."

Irving, J. D.—"Some Features of Replacement-Ore-Bodies and the Criteria by Means of Which They May Be Discovered."

Gamba, F. P.—"Geology of Some Mines in the South of Colombia, S. A."

Keyes, C. R.—"Vadose Ore-Deposition in Arid Regions."

McCollum, Burton.—"Electrical Practice in Mines."

Vallat, B. W.—"Geology and Operation of the Newport Mine."

Read, T. T.—"Mineral Resources of China."

Thursday Afternoon (in the open-air Greek Theater at Berkeley.)

Raymond, R. W.—"Reminiscences of the Beginning of the Institute."

Becker, G. T.—"Biographical Notes on S. F. Emmons." Abstracts of some of these papers, and the discussions will follow in our next number.

The excursions arranged for the meeting were pleasurable and highly interesting. On the afternoon of the 11th, a special train left for Palo Alto, where over two hours were spent at the Stanford University. The next afternoon a similar trip was made to the University of California, at Berkeley, with a short session held in the famous Greek Theater.

Oct. 13 was given over entirely to seeing the gold dredges of the National Consolidated Company, along the American River

near Sacramento. The visit was a revelation of the latest triumphs of this manner of mining gold. The enormous new electrically operated dredge No. 9 was eating into *terra firma* at the rate of 12,000 cu. yd. per 24 hours. Digging to 55 ft. below water level, and taking down 10 ft. above, this enormous 2400-ton craft, operated by 1000 hp, was said to be working over earth carrying 8 cents worth of gold per cubic yard, at a total cost of not over 4 cents per yard, including depreciation and interest on capital invested.

Equally interesting was the effort being made to reclaim this worked-over ground. Working in a pond, the dredge leaves the gravel piled high as heaps of stones, while the soil or silt impregnates the lower layers under water. The most successful scheme seems to be the crushing of the stone for railroad ballast, road material and concrete. This crushed stone is worth 75 to 90 cents per ton at the crushers, and is being shipped all over central California. This disposes of the stones down to the silt or soil. The ground is then leveled off and planted with eucalyptus trees or rubber trees. These grow in the stony ground if silt or soil is not over 2 ft. beneath, and the tract of land becomes green again. An alternative would be to strip 2 ft. or 3 ft. of soil from the land before working, pile it aside and return it to the leveled-off ground. In any event, appropriate legislation should compel all the gold diggers to leave the country in a cultivable condition, as part and parcel of the expense of getting out the gold, and prohibit leaving the worked-over ground a stony waste, more barren than the desert of Sahara.

Returning from Natoma through the beautiful Sacramento valley, another sacrilege was visible from the train in the shape of a tall smelter stack vomiting out a poisonous cloud of white smoke which enshrouded the valley and mountains for miles away. Such a white pest, prejudicial to health, agriculture and natural beauty, should be suppressed with a firm, determined hand. Some practical methods of stopping the smoke are now known; it is simply a question of private greed *versus* public good, and may the public quickly win out!

Saturday, Oct. 14, 1911, is now memorable in California as the day of breaking ground for the great 1915 exposition. The President of the nation was on hand to start "making the dirt fly." Members of the Institute and guests were presented with grand-stand tickets, and joined the throng, 100,000 strong, which gathered in the stadium of Golden Gate Park. In bright sunshine, under a perfect sky, the surrounding hills green with verdure as a background, the tranquil Pacific visible to the west, and to the accompaniment of a military parade, martial music and echoing salutes of cannon, the first sod of the great exposition was turned. Just before, President Taft, in a ringing address audible to thousands, reviewed in brief the history of the canal, told them that 75 per cent of the work on it was now done, and predicted its completion by July, 1913. One sentence of the President's address lodges in our memory as unusually forcible and strikingly true: "Had the sea-level plan of the canal been adopted it is altogether probable that no one who now hears the sound of my voice would have lived to see the canal completed."

Finally, Oct. 15 is now memorable in the remembrances of the visiting mining engineers for the day—one glorious day of throbbing life—in the sacred groves of the red-woods. Bohemia the spot was called. Here the Bohemia Club stopped the woodcutter's ruthless axe and spared a spot in which to bury care, to worship in nature's temple. It is a small valley, the ground springy with its forest carpet, with scores of congenial spirits in the shape of our giant brothers, the big trees. Some are 10 ft. in diameter and over 300 ft. high. Among such we pigmies stalked and strutted and enjoyed our little day. A splendid orchestra added to the charm some lovely music of Bohemia, two of the numbers being conducted by the composers themselves. A more harmonious assemblage of artistic enjoyments, beauteous surroundings and congenial society cannot be imagined. Here the above photograph was taken.

On the evening of the 16th the California section of the Mining and Metallurgical Section met at dinner, with many members of the Institute as guests. Later the withdrawal of the public coal lands in Alaska was discussed pro and con. with considerable spirit. The wide diversity of opinion expressed showed how impossible it must be, in the nature of the case, to secure a uniform verdict on any such matter which would correctly represent the views of the society as a whole. While the discussions are certainly profitable and instructive and lend a proper interest to the meetings, yet a final expression of the opinion of the society as a whole could not be arrived at without great difficulty and arousing of bitter antagonisms. Such a practice is a principle of separation, a source of discord which would ultimately disrupt any society trying to put it into practice.

Tuesday, Oct. 17, saw the fortunate "100" leaving for their seven weeks' trip to Japan, and the more occupied and less fortunate remainder leaving for their homes. The trip to Japan, stopping at Honolulu, promises to be of absorbing interest. Dr. Kanda, of Japan, has come across the Pacific as the advance guard of welcome, and a glorious eighteen days of strenuous experience awaits the expectant engineers in the Kingdom of the Rising Sun—of which more anon. J. W. R.

The Western Metallurgical Field.

Company Reports.

The metallurgical results obtained by El Oro Mining & Railway Company, Ltd., El Oro, Mexico, are made public in the last annual report for the period ending June 30, 1911. The total ore mined during the year was 360,504 tons as compared

with 317,824 tons in the preceding year. Changes have been made so that sulphide ores can be crushed and cyanided separately, thus improving results. Additions were made to plant consisting of six agitation tanks, two thickening tanks and three revolving filters. These were installed principally to aid in the treatment of sulphides and high silver ores.

The percentage recovery for the year was slightly lower than in the preceding period, but this was due to the fact that a lower grade ore was treated in larger quantities, thereby reducing the percentage recovery but making an increased net profit on operations. If the former high recovery had been maintained it would have resulted in a loss on the extra bullion produced. The principal data are in the following table.

Mill in operation, days	349.4
Tons crushed per stamp per 24 hours	10.3
Tons crushed	360,294
Gold value per ton	$5.99
Silver value per ton	$1.64
Per cent gold extracted	91
Per cent silver extracted	73
Per cent total extraction	87

The costs per ton for the year under consideration are as follows:

Mining	$1.55
Development	0.74
Milling	0.17
Cyaniding	0.91
Water supply	0.02
General expense	0.22
Taxes	0.29
Total	$3.90

The Consolidated Mercur Gold Mines Company, Mercur, Utah, is apparently nearing the end of its life as a prosperous concern. The mines of the company have had a most excellent record for the past twenty-five years, but for the last three years it has been apparent that the grade of the ore was becoming lower and the profit less. The net profit in the last fiscal period was less than $500, and this did not come directly from the mill but from outside sources. The average gross value of the ore treated was $3.21, this including the value of old tailings retreated which amounted to 12 per cent of the total. The total quantity of material treated was 239,190 tons, and the value extracted was $2.32 per ton. Loss in tailing was $0.88 per ton. The cost of producing gold was slightly higher than its value, or $20.9218 per ounce. This shows a loss per ounce of approximately $0.25.

Of a more encouraging nature is the August report of the Goldfield Consolidated Mines Company, Nevada. It shows the largest tonnage ever produced by the mines in any one month, the total being 30,580 tons, or nearly 1000 tons per day. The average gross value was $29.77 per ton, and the total net profit was $634,658. Of the total mine production, 30,508 tons were treated in the mill with an average extraction of 94.75 per cent, or $20.75 per ton. The total operating costs amounted to $7.46 per ton. The company is now engaged in constructing a plant to treat both the accumulated and current residues from the present concentrate treatment process. Two Edwards duplex roasting furnaces, one Baker cooler, one 5-ft. by 18-ft. tube mill and other accessory apparatus are being installed in a steel building erected for the purpose. The additional tanks for collecting the roasted product and agitating in cyanide solution will be contained in an addition to the present plant. Tests show that an average of 90 per cent of the value of these residues can be recovered with an annual saving of at least $80,000. The cost of the additional plant will be about $70,000.

Renewed Interest in Mining.

The decisive steps taken by Western commercial and scientific bodies to renew public interest in mining are apparently bearing fruit. From present indications it is certain that mining in general but ore treatment in particular is going to profit by the efforts being made to bring the operator and technical man closer together, and there can be no doubt that the general education of the public will result in attracting money for mining investment on a sane basis.

The latest developments on this line which give promise of improving general conditions are two, viz., the proposed unwatering of the downtown district of Leadville, and the meetings planned jointly by the Colorado School of Mines and the Colorado Scientific Society. Co-operation for mutual benefit is the underlying motive in each case.

The downtown district of Leadville has been unproductive for some years owing to the large volume of water encountered in the mines and the expense of pumping it to the surface. Drainage tunnels have not been feasible, and until recently all proposals for pumping have seemed impossible. The one link necessary to complete the chain of forces which make the plan attractive is the Central Colorado Power Company, which has ample power at its disposal. At a recent meeting held in Leadville the power company outlined a plan for the formation of a pumping company with sufficient capital to purchase and install the necessary machinery. The power company proposes to furnish most of the working capital, and it is expected that the pumping company will ultimately operate at a profit to be derived from a tariff placed on net smelter returns of ore mined as a result of the pumping. Naturally the power company has a selfish motive in proposing this plan, but on the other hand, the plan is such that no one will profit unless valuable ore is produced. It is not expected that any one will benefit at the expense of any other party concerned.

The most important matter is to secure the co-operation of the owners of the mines which it is proposed to unwater, and to receive assurance from them that if they do not wish to work their mines they will make reasonable concessions to leasers. Taken as a whole the plan is very encouraging, for it is known that ore exists in the flooded district, which was only abandoned when the expense of combating the water became unbearable. The power company has competent engineers at work on the details of the plan and it is expected that a definite scheme will be ready for presentation to the mine owners in the near future.

The other plan referred to, that of meetings under the joint auspices of the School of Mines and the Colorado Scientific Society, is expected to bring improved results not only in renewed interest, but also in improved technical work. The plan contemplates holding meetings in the different mining and metallurgical centers in the State at which a selected speaker will open a discussion on some pertinent subject. The discussion will bring out the views of local operators as well as visitors, and the exchange of opinions must undoubtedly do much good. The list of speakers already obtained comprises the best technical talent in the State. Experts on mining, mine management, ore treatment and plant construction will discuss their subjects in districts where they are most appropriate. This will bring the engineer in closer touch with the problems of the various districts, as well as familiarize the local operator with the ideas and experiences of the engineer.

Improvements at El Paso Smelter.

For some time past the American Smelting & Refining Company has been making additions to its El Paso (Tex.) plant, for the purpose of handling the copper concentrates from the Ray and Chino concentrators. Three new oil-burning reverberatories have been built and are probably now in commission. Hitherto the Ray concentrates have been going to Mexican smelters, but will now be diverted to El Paso. The new Ray smelter recently taken over by the American Smelting & Refining Company will not be ready for operation for some time to come, and until then the Ray concentrates will be shipped to El Paso.

Ray Central.

Engineers Weed and Probert have recently estimated the cost of building a 1500-ton concentrating mill at the property of the Ray Central Copper Mining Company, Arizona, and for permanently equipping the mine for large production, at $1,500,000. They estimate a 75 per cent extraction in the mill, concentrating six tons into one, and a smelting charge of $6 per ton of concentrate. Assuming the smelter to pay for 95 per cent of contained copper, less 2.5 cents New York quotations, and a 13-cent copper market, the high grade ore should yield $5 per ton net. The cost of production should be about 7 cents per pound, exclusive of amortization and interest.

Flotation Process in Idaho.

It is reported that the Macquisten tubes installed in the Morning mill at Wallace, Idaho, are giving satisfaction and producing a good grade of zinc concentrate. Reference to this installation was made more fully in our issue for May, 1911, page 236. The material being treated by the tubes is tailing from Wilfley tables, and contains about 10 per cent zinc. Siderite is present to the extent of about 35 per cent of the ore pulp, and is the principal substance to be separated. Its specific gravity is 3.8, while that of blende is about 4. Thus there is little margin for separation by ordinary concentration methods, and the flotation scheme was resorted to as offering the best medium of separation. It is reported that the installation is turning out zinc concentrate containing 50 per cent zinc.

The Iron and Steel Market.

Bookings of orders for finished steel products in October showed no material change in the total as compared with September, and on the whole fell slightly below the production, through orders for rails and fabricated steel being light, while in practically all other lines bookings were up to the production. Output showed a slight gain over September, being thus better than in any preceding month of the half year. The increased outputs were in the main obtained without the putting of any additional productive units in operation, the month being normally one of heavy outputs per unit, on account f favorable weather conditions.

There has been no break in the monotony of the price decline. Plates and shapes have suffered a decline of about $2 a ton, and steel bars and black sheets a decline of about $1 a ton, while tin plate was definitely reduced 20 cents a box or $4 a ton. While nominally open market conditions prevail in the tin plate market, there is definite leadership by the American Sheet & Tin Plate Company, the subsidiary of the steel corporation, which last year made more than 60 per cent of the country's total tin plate output, and its prices are customarily adhered to by the independent producers. Its reduction was made because there had been such a general decline in other steel products and because the shading on small lots for early shipment which had already appeared suggested clearly the impossibility of the old price being maintained for the new season's business. About this time of year the large can makers customarily place their contracts for the first half or the whole of the new year.

In the products outside of tin plates which declined open market prices prevail in practice as well as in theory. Of the country's sheet production last year the American Sheet & Tin Plate Company made less than 40 per cent. In these strictly open-market lines the declines have occurred gradually, through a given low price first being made in the case of a few exceptionally attractive orders, then upon less attractive orders, and finally upon the common run of orders.

Weighting the important finished steel products outside of standard section rails approximately in the order of their tonnage importance, the declines per net ton in the average price of finished steel products during the past four months have been as follows:

July	$0.70
August	.50
September	.95
October	.90
Four months	$3.05

The average decline of $3.05 a ton in the past four months came upon a market which had already declined $1.35 since the beginning of the year and $8.05 since the late months of 1907, when the maximum of the past nine years was reached.

That the cost line has been closely approached in the present market is declared by all producers, and there is no reason to doubt the statement. Attention should be called, however, to the fact that as a rule the mills are accepting specifications of a very much mixed character, and filling them, by prompt shipment, at practically the minimum market prices. That mill costs per ton would be materially lower were operations conducted upon better selected and arranged specifications there can be no question. Buyers are insisting upon prompt shipment against all specifications accepted, and the mills appear to be powerless, on account of the competitive situation, to exact premiums in proportion to the undesirableness of the specification. A great deal is said in the market about specially low prices being made on orders involving "attractive specifications" and of course there is some selection, but in the final analysis specifications being filled result in costs much higher than in periods when mill capacity is strained and orders are so long in filling that judicious selection and arrangement can be practised.

The industry is operating at about 75 per cent of its full capacity, making pig iron at the rate of about 25,000,000 tons a year, approximately the average of the three-year period of great activity immediately preceding the panic of October, 1907. The independents are operating at an average of slightly less than 75 per cent, and the steel corporation at a slightly better proportion, but nearly 15 per cent of its entire capacity is engaged upon export material, the independents conducting very little export trade.

There is no definite promise in general industrial conditions of an improvement in iron and steel consumption in the near future. Purchases, production and actual ultimate consumption are all proceeding at an even pace. Stocks in the hands of jobbers and manufacturing consumers are at a very low ebb, and there is the possibility of present prices proving so attractive to buyers as to induce them to increase their stocks toward normal proportions. No movement of this sort is discernible, but many buyers are openly stating that the moment they observe a hardening tendency in prices, or a slight falling behind in deliveries by the mills, they will promptly place orders for much larger tonnages. A very small event may thus at any time lead to important consequences by way of heavier buying, producing a spurt not altogether healthy through actual ultimate consumption not being correspondingly increased. On the other hand, the approach of winter may operate to curtail consumption in a minor degree and lead to slightly decreased operations. In a word, tonnage is very heavy, prices unsatisfactory and declining, and the market decidedly uninteresting.

Pig Iron.

Sales of pig iron have been light as to tonnage, but a large number of small orders have been placed, and deliveries on contracts are very good. Buyers had contracted very conservatively. There is scarcely any forward buying, no advancing tendency being observed in the market, while occasionally slight declines occur at one point or another. The actual consumption of merchant iron is believed to have been measurably in excess of production during the past two months, effecting a slight decrease in the rather large stocks in merchant furnace yards. These decreases are chiefly at inactive furnaces. An event of considerable interest in October was a break in the price of Bessemer pig iron. This market, the scene only two

years ago of transactions in 10,000 to 25,000-ton lots, held at $15, valley, for more than a year, and then broke early in October on a sale of 1,000 tons at $14.65, valley, iron being later offered at $14.50, valley, without any important sales being effected. The market stands quotable as follows: Birmingham, $10 for No. 2 foundry; delivered Philadelphia, $15 for No. 2 X, $14.50 for basic; f. o. b. Chicago district furnaces, $14 to $14.25 for No. 2 foundry; f. o. b. valley furnaces, 90 cents higher delivered Pittsburgh; Bessemer, $14.50; basic, $12.50; malleable, $13; No. 2 foundry, $13.25 to $13.50; forge, $12.75 to $13.

Steel.

An awkward situation arose in the sheet bar market regarding settlements for October and fourth quarter tonnages. Practically all the consumption stood covered by term contracts, providing for price adjustment monthly or quarterly in a variety of ways, and frequently at the open market, or a slight fixed differential below the market. There was, however, no actual market in existence. There was the tradition of the old price of $22, f. o. b. maker's mill, Pittsburgh or Youngstown. This had not been broken in Bessemer steel, for which the term contracts almost invariably called, but open-hearth bars had sold in small lots as low as $19.50, and the entire finished steel market had declined since the $22 price on sheet bars had first been named. In these circumstances a condition of open competition developed, even though all consumers had contracted for their tonnage, and settlement prices were finally reached at a considerable range, $21, delivered Pittsburgh district or Youngstown, being done in some cases, while in other cases lower prices, down to $20.50, were made. The market is not precisely quotable, and we name the following quotations, which are more or less nominal: Bessemer billets, $20, maker's mill, Pittsburgh or Youngstown; Bessemer sheet bars, $21, delivered Pittsburgh district or Youngstown; open-hearth billets, $19, and open-hearth sheet bars, $20, both f. o. b. maker's mill and subject to frequent concessions.

Finished Material.

Strictly open-market conditions prevail in plates, shapes, bars, and sheets, and on these we quote minimum prices done on fairly attractive orders, these prices being occasionally shaded, while on the least desirable orders slightly higher prices are obtained; in wire products the market is maintained in Pittsburgh and the East, but is frequently if not usually shaded in the South, Southwest and West; in tin plates the market is steadily held. The Pittsburgh basis, as a universally controlling factor, has disappeared in several lines.

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f.o.b. mill, except Colorado.

Plates, tank quality, 1.15 cents, Pittsburgh.

Shapes, 1.20 cents for beams and channels, 3 in. to 15 in., inclusive, zees, and angles 3 x 3 and larger, Pittsburgh.

Steel bars, base, 1.10 cents, Pittsburgh; 1.15 cents, Buffalo; 1.20 cents, Chicago; iron bars, 1.25 cents, Pittsburgh; 1.22 cents, delivered Philadelphia; 1.17 cents, Chicago.

Wire nails, $1.65, base; plain wire, 1.45 cents, base, Pittsburgh.

Black sheets, 28-gage, 1.85 cents; galvanized, 2.90 cents; blue annealed, 1.40 cents, f.o.b. Pittsburgh or Mahoning Valley; 2 cents, 3 cents, and 1.50 cents respectively, Chicago; painted corrugated roofing, $1.35 per square, and galvanized $2.50 per square, Pittsburgh, but frequently shaded.

Tin plates, $3.40, Pittsburgh, for 100-lb. cokes.

CORRESPONDENCE.

An Instructive Experiment with the Flow of Heat.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—The following simple experiment may be of interest, as it shows some of the peculiarities of the flow of heat through bodies.

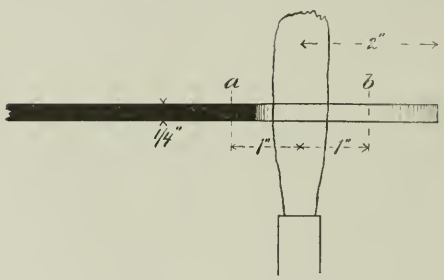
A copper rod ¼ in. in diameter was held in a Bunsen blast lamp flame as shown in the figure, the essential feature being that one end projected only slightly beyond the flame, while the other was practically infinitely long. When the stable state was presumably reached and the rod was at a bright-red heat in the flame, the visibly hot part was unsymmetrical, extending much farther toward the short end, in fact quite to the end of the rod.

This is shown in the figure, the white representing the red-hot part.

The heat enters the rod in the middle of the flame and flows from there to the two ends. It seems to follow, therefore, that more heat flows through the section *a* of the long end than through section *b* of the short end, both being equally distant from the middle of the flame; at *a* it was black and at *b* it was red-hot.

There is, therefore, less loss of heat to the air from the red-hot surface than from the cold surface.

It seems that the heat flowing to the hot end meets with a high virtual resistance at the surface of contact of the metal and the air, and that this resistance is in effect far higher than that of the metal. It would therefore seem to follow that if one side of a tank containing a red-hot liquid were made of a thin copper sheet, while another side was formed by the end of a long massive bar of copper, hence very thick, less heat would be lost through the thin wall than through the thick one.



ARRANGEMENT OF EXPERIMENT ON FLOW OF HEAT.

Or, in other words, the loss through the walls per square inch would be greater than that from the exposed top surface of metal.

If this were also true of the materials which are generally used for furnace walls, it would seriously upset our present ideas that a thick wall insulates better than a thin one. But a repetition of this experiment, using a rod of clay (the stem of an ordinary clay pipe) of the same diameter, in place of the copper, fortunately showed no such dissymmetry. A more carefully made test would probably even have shown a dissymmetry in the other direction.

A carbon rod acted like the copper one, but the dissymmetry was far less.

This apparently very high surface resistance to the flow of heat may be similar to the one existing in steam boilers, the presence of which can be shown by the fact that a postage stamp pasted on the fire side of a water tube will not even be charred by the flames. This high-surface resistance could be made use of in furnace walls in the form of joints between bricks.

The virtual resistance at surfaces seems to be high as compared with that of copper, but may be low as compared with that of the usual heat-insulating materials. Existing data on this subject seem to be deplorably meager, and perhaps quite unreliable in some respects.

If the thermal resistivity of copper decreases greatly with an increase of temperature then the above results might be explained quite differently. Carl Hering.

Philadelphia, Pa.

Reminiscences of Robert Wilhelm Bunsen.

By Dr. Hans Goldschmidt.

The centenary of Bunsen's birth has given me as an old pupil of the great savant, the idea of writing down some reminiscences of my student days at Heidelberg.

Of the many who have taught me in school, at home and at the University, Bunsen is the one teacher who impressed me most deeply as being truly a teacher by nature.

All who worked in Bunsen's laboratory had every possible opportunity of learning every detail of the science of chemistry from the great master, not only theoretically, but also from the purely practical, manual point of view.

Of course, Bunsen only devoted himself in this way to the regular and diligent worker in his laboratory. He was always ready, however, to help the occasional student, and was never otherwise than friendly to the less diligent; but the student who did not work at the laboratory regularly, could not but feel that Bunsen preferred to give his time to those who worked hard.

He put himself at the disposal of the latter out of working hours as well; indeed, I always had the feeling that Bunsen liked to be "bothered" by those students who were in earnest in their work. Strange as it seems, the constantly repeated monotonous teaching never seemed to be a torture to him.

I am speaking too of a time when Bunsen was fully 75 years old. I was, therefore, among the last of his pupils.

Bunsen attached special importance to his pupils being thoroughly conversant with qualitative analysis; he only allowed experiments in quantitative analysis when he knew that the student was complete master of the former.

As in other laboratories, so in Bunsen's, certain numbered mixtures were given out to be analyzed. He had the results put down in a small note book which he always carried about him. I shall never forget his delightful remark, when I asked him to tell me if the results of the qualitative analysis which I put before him were correct. I named the three or four elements which I had found, and the old gentleman gave me a glance full of meaning and said: "I don't want to know what you have found, please tell me what you have *not* found." He then went through the different groups of elements with me and made me describe exactly the process of analysis. I had to explain in each case how the reactions had taken place, and had furthermore to write down the respective chemical equations and put down separately what I had seen, and what I had not seen. From this it can be seen how seriously the great savant dealt even with beginners.

Bunsen was never tired of showing his students even the smallest secrets of the science; how a cracked test-tube can be made fit for use again by the blast lamp, how glass filaments are drawn, how a filter is correctly placed in the funnel, and many another useful piece of knowledge.

The tests known under the name of "Bunsen flame-reactions" were of very special interest. Even at his advanced age and in spite of a somewhat trembling hand, Bunsen could carry out these fine microscopical chemical operations with a skill and a perfection that excited wonder.

The palm of the left hand of the experimenter played an important part in this. Bunsen used it in this way: he rubbed the ingredients into it with the help of a penknife; during the experiment, the fold of the hand made by closing the little finger over the palm, formed the provision store for the mixture, used in various ways during the analysis. The thumb and the first and second fingers of the left hand remained free for use, in spite of the closed little finger.

These researches, which were carried out in the flame of the Bunsen burner, took the place of and were an improvement on the so-called blowpipe analysis, which Bunsen did not like. It was his capacity for obtaining the most striking reactions with the smallest means, that marked Bunsen's teaching and was of the greatest value didactically.

Those who wished to take their degree were not allowed by Bunsen to work out a doctor-thesis, the viva voce examination was enough for him. His point of view was that the young student was not in a position to do independent work, and that, therefore, the doctor-thesis could only be due to the initiative of the teacher; he, therefore, thought it better for the student, as long as he remained at the university, to deepen and widen his knowledge thoroughly in all the fundamental parts of the science placed before him, without trying to break out new ground himself.

As an examiner, Bunsen was particularly thoughtful and kind. Every one who had listened to his lectures attentively and worked through them thoroughly, was certain to get a good place in the examination. In my examination, he questioned me specially on his iodine titration, which he had gone into with me in the laboratory with special care so that he could not but be certain that I had it all at my finger-ends.

I may be allowed here to repeat a remark made by Geh. Rat Kopp and which was current among the students in those days—si non è vero è bene trovato—"Whoever passes his examination in Heidelberg with rite (the lowest class), has it written down in pen and ink that he is an ass." Little Geh. Rat Kopp, who is best known through his great work on the "History of Chemistry," was the faithful companion in walks, and above all on journeys, of the physically much bigger Bunsen.

Bunsen was pleased if any of his pupils, even after the examination, continued to work under him on subjects of inorganic chemistry, which he had worked at himself. I myself, for instance, seized the opportunity of working at the production of calcium and strontium from molten chlorides, by means of electric current.

In those days this was a hard piece of work, as it had to be carried out with a Bunsen battery, and the only electrical measuring instrument was a tangent galvanometer. Not only was it hard, but particularly hot work, and very troublesome, because the chloride had to be melted on an open charcoal fire.

In spite of the heat of summer, Bunsen insisted on watching the experiment for hours, even long after the closing of the laboratory. He even considered it necessary to be present at the separating of the crucibles, in order to see and control the results.

He then went carefully through the theoretical calculations with me and took care too that the small reguli of calcium and strontium (which I still possess) should be fused in petroleum free from water, which he gave me himself out of an old store. He showed me also the process of fusing in the small tubes.

On these occasions the old gentleman sometimes became very talkative and would converse on matters that lay far outside

the realms of chemistry. He liked to talk about his journeys. A visit to the island of Majorca had left particularly pleasant memories behind.[1]

Another time he told me stories of his teaching days at Göttingen, even then fifty years back. Among other things, he told me of a friend who, in consequence of a bet, had deciphered an Assyrian inscription within three weeks. These old memories seem to have struck particularly deep root in him.

Of greatest interest to me was a remark Bunsen once made, when he was discoursing on the advanced technical knowledge of the Egyptians: he said he had been shown small ancient copper bowls from the old wonderland of the pyramids, which by their form had suggested to him the idea that they must have been precipitated with the help of the electric current. This bold suggestion from a man of very sober thought is very striking.

I remember a particularly interesting occurrence when Bunsen allowed me to make some spectral-analytical experiments, such as were often made in his laboratory by the students. The experiment consisted chiefly in this, that the student, who had at his disposal a considerable number of pure substances in small glasses, attached a little of one substance to a platinum wire by dipping it into that substance, and then introduced it into the Bunsen flame, observing the rays or lines in the spectroscope and putting them down on a graduated scale.

The sodium line served as basis. Bunsen would point this one out with a cigar he held in the flame. As is well known, he was a smoker of strong Havanas. He never failed to point out that in every Havana you could see clearly also the red lithium line. If, by chance, a student himself had a cigar which showed no lithium reaction, Bunsen can scarcely ever have missed the opportunity of making a little joke, by pointing out that the student's cigar was no "Havana."

Among the elements I had to analyze were small quantities of the salts of caesium and rubidium discovered by Bunsen.

When Bunsen saw the lines I had put down for caesium, he shook his head doubtfully and expressed his opinion that it was impossible for me to see the two lines in the violet. The test was repeated over and over again, but I always put the lines exactly in the same place.

I never saw the old gentleman so excited and interested as on that occasion. He thought I must have made a mistake, for both the lines observed by me belonged to rubidium. The caesium given me to work with had not been perfectly pure, it contained traces of rubidium.

I could see these rubidium lines clearly, a fact that led Bunsen to declare that my eyes were exceptionally sensitive to violet rays, and he added: "Had I had your eyes, the separation and above all the discovery of rubidium would have been rendered much easier to me."[2]

He then invited me to go with him to his private laboratory. He had there larger quantities of salts coming from the Dürkheim mother liquors which contained only infinitesimal portions of rubidium and caesium.

He wanted to try if I could see the lines in this extreme dilution, though he scarcely thought it possible. And, as a matter of fact, it was impossible to recognize the salts in the substance handed to me by Bunsen. He made certain once more, however, that there was no mistake on my part in what I had seen and put down before in my earlier tests.

The last work Bunsen gave me was purely preparative and related to the preparation of cadmium borotungstate. The solution of this salt with its very high specific gravity was to be used for the separation of minerals.

As a rule, Bunsen was to be met among his students in the laboratory for some hours both in the morning and in the afternoon. During the summer months, he was often even before his first lecture at the disposal of those who wished to learn from him his gas-analytical methods.

Besides this, he gave a daily lecture on experimental inorganic chemistry, a lecture which was probably unique. In one semester he gave a complete survey of inorganic chemistry and what he demonstrated and expounded was from beginning to end his own work, his own experience, his own research.

Bunsen attached the greatest importance to the illustration of his lectures by means of experiments. I cannot remember the failure of a single one. They too were carried out with the simplest means. For instance, Bunsen showed the working of the geysers with a simple apparatus, and prepared before his hearers electrolytically metallic lithium in a porcelain bowl.

One experiment was especially effective: oxy-hydrogen gas was introduced into a small china bowl full of soap and water. Soap bubbles filled with this gas formed at once on the surface. If a light were applied to them, they exploded and the bowl broke in pieces. Bunsen then repeated the experiment by forming the soap bubbles filled with the gas on the palm of his hand. The explosion was quite as loud, but the hand remained absolutely unhurt, because it formed an elastic foundation, not a hard one like the bowl placed on the lecture table.

After such experiments, the students' applause was particularly stormy, it expressed itself in noisy stamping. Clapping of hands was proscribed. These student ovations were also given to Bunsen when he entered and when he left the lecture room, and he always thanked with a friendly glance and a scarcely perceptible nod.

Bunsen's manner of lecturing may be described as classic, in the best sense of the word. His sentences were short, his delivery clear, his voice even in advanced age sonorous and fairly strong. He despised every kind of hollow figure of speech or brilliant comparison. In this respect one could find no greater contrast to him than the orator and philosopher Kuno Fischer, who was lecturing at the Ruperta-Carola, the Heidelberg university, at the same time as Bunsen.

Bunsen's favorite study was neither chemistry nor physics, it was geology. In his lectures he often referred to this branch of science.

But Bunsen was also in possession of more than ordinary knowledge in zoology, botany and above all, in medicine,[3] so that it is not too much to maintain that he belongs to the men of his age who could claim the greatest general scientific culture.

Bunsen was of a meditative nature with an indubitably highly developed inner life. And this explains his dislike to ostentatious display, his wish not to appear in public or be disturbed by social duties more than was unavoidable. He was perfectly natural in his whole manner of being and acting. He was so thoroughly absorbed by his work, so taken up by it, that he felt no desire for things that seemed to him paltry.

With him, one always had the feeling that he placed the problem itself above everything and all that was personal as far out of sight as possible. That was why in his lectures, one never heard the expression: "I have found" or "I have discovered." He always spoke with the impersonal "one." Whether he spoke of the Bunsen burner, of the Bunsen battery, of the Bunsen valve, or of the spectral analysis-inventions that bore his name already while he was yet living—he never put himself forward, never mentioned his own name.

Bunsen was a great, even enthusiastic lover of the beauties of nature; he preferred wandering about on foot, so that he might gain strength in the fresh open air. Heidelberg, with its wooded surroundings, had become very dear to him. There is no doubt that he was deeply attached to his German Father-

[1] He had gone there with the zoologist, Dr. H. A. Pagenstecher, professor at the University of Heidelberg. The latter has given a clear and charming account of this journey in a small book, which is still worth reading, "The Island of Majorca: Sketches of a Journey," by Dr. H. A. Pagenstecher (Leipzig: W. Engelmann, 1867).

[2] Cp. on a fifth element beloneing to the alkali group. By Bunsen. *Berichte* der Akad. der Wissenschaften z. Berlin, 1861, L 27.375.

[3] I may point out here that Bunsen owed his invention of the well-known contrivance the "Bunsen valve" to his medical knowledge of the flaps or valves in the veins.

land, through which he had wandered much in earlier years, but he was free from any onesided overweening appreciation of German intellectual efforts. His views and opinions were not limited by a narrow circle. Our planet did not suffice for his chemical and physical research, the eye of this gifted searcher pierced to the furthest worlds.

Much has been said about Bunsen's great modesty. Since Goethe declared that "only rascals are modest" this quality requires a certain reservation, when it is attributed to a great man. And this is specially called for in the case of our savant. He was an exceptionally high-minded man, most courteously kind in intercourse with others. That was noticeable even in small things. For instance, when a student entered his private laboratory with him, he would never go in first, but obliged the student to do so. If any one met the old gentleman out for a walk, he had to be careful to be in time in bowing first, for Bunsen never waited till he was saluted, but would take off his hat a long way off to the student whose face he recognized.

But together with this well-bred politeness, this simplicity in ordinary intercourse, there was in him a great, one may confidentially say, a too great consciousness of his merits. For Bunsen believed in, and set a value almost exclusively in, what he himself had seen, investigated, and discovered. Others' work he considered doubtful as a matter of course, very often ignored it and even treated with scorn many an investigator who did not deserve such treatment. He expressed this scorn, of course, only in incidental remarks and conversations, for he never wrote a controversial paper. In this depreciation of others' work and the careful avoidance of printed replies, he resembled the great Linnaeus, who, however, unlike Bunsen, took special delight in being himself in the foreground.

As an example, I may quote Bunsen's drastic remark—repeated to me by his late assistant Pagel, who was many years with him—on the periodical system of Mendelejeff, who was in Heidelberg from 1859 to 1860: "If a clever mathematician takes the Stock Exchange column of a newspaper, he also can discover fixed series of values that have no connection with one another." Bunsen never mentioned Mendelejeff's system in his lectures.

The great investigator also stood aloof from the then modern views on atoms and their groupings. His friend Kopp tried to enlighten him in an exceedingly clever and witty publication "From the world of molecules,"[4] which first appeared anonymously. This little book can be warmly recommended to every chemist who has a sense of humor, as far as that can enter into purely scientific questions. Often a very pertinent doubt on the part of the author pierces through the explanations, and this must have made the small work still more savory to the doubter Bunsen.

As Bunsen lectured substantially only on what he had himself investigated, so in his laboratory he only allowed work on subjects he suggested himself, and on which he had worked himself. As soon as a laboratory student chose a subject on his own account, the master's interest ceased. This peculiarity finds its explanation in his conscientiousness as a teacher; he wished to demonstrate and teach in his laboratory only what he was thoroughly acquainted with himself.

There were already a great number of Bunsen anecdotes current during his lifetime. Many of them were meant to prove the helplessness of the savant in things which were likely to be unfamiliar to him, especially as a bachelor, or which showed his awkwardness in society in general. The absentmindedness attributed to every German savant played its part too in these anecdotes. If we were to trust these traditions, even only in their general tendency, our mental picture of the great man would be a gross caricature. Bunsen was a thorough man of the world, who had traveled a great deal and had associated with men of every condition, high and low. He knew perfectly well how to behave correctly and befittingly in every situation in life. On the other hand, the great savant possessed a delicate sense of humor and a great taste for irony, both of which he often turned against himself; he liked to be ironical about himself. Probably those often foolish Bunsen stories owe their origin to this peculiarity of his, his joke having been misunderstood.

The domain of chemistry, especially of inorganic chemistry, that Bunsen was master of, was extraordinarily wide, for he possessed it in all its smallest details, whether it were a problem of the separation of rare earths, of the group of platinum metals, of the most difficult gas analyses, or of smelting processes of every kind. The work he did, as pioneer, in almost every department that he touched upon, was so comprehensive and always so thorough, that he did not seem able to find the time to make himself sufficiently acquainted with the realms of investigation of his contemporaneous colleagues. Thus he had not many collaborators; the best known are Kirchhoff and Roscoe.

To us, who look back, he stands out in solitary grandeur, far above others. His weakness, in ignoring the work and success of others, became finally a source of strength. His immovable trust in his own creative power absorbed him so exclusively, that he gave the world a great deal of new and most valuable knowledge, a fruitful and priceless possession for all time.

Essen-Ruhr, Germany.

[4] Aus der Molekularwelt, published by Winter in Heidelberg.

Proposed Treatment of Sulphide Ores with Nitric Acid, or So-Called Rankin Process.

Editorial Correspondence.

The difficulties attending the economic treatment of ores containing the sulphide minerals of several metals have resulted in classifying such ores as refractory. This term is generally taken to mean that these ores do not yield readily to known methods of treatment, and consequently the newer developments in metallurgy are heralded as means of successfully treating refractory ores.

The proposal to treat mixed sulphide ores with nitric acid is an endeavor to solve this problem in an economical way. The recovery of the nitric acid is a vital feature of the process. In its present state of development it represents the original idea of Mr. H. D. Rankin as modified and improved by his coworkers.

In the following description of the process reference will be had to the treatment of an ore containing some form of the sulphides of iron, copper, lead and zinc, with accompanying gold and silver, either native or combined. The statements given represent the claims for the process as outlined to the writer by Mr. E. H. Westling, metallurgist for the company controlling the process. Experimental work has been done at Pittsburgh, Pa., and Lake City, Colo., and it is announced by the company that a commercial plant is to be built and operated immediately at Chloride, Ariz.

With the exception of gold, the products of the process are not metals ready for refining, but salts of the metals equivalent to high-grade ores suitable for reduction by present processes of smelting. In some instances the precipitated salts may find commercial use as such, as in the case of iron and copper, but for the most part they must be reduced to metals. Briefly, the primary products of the process are: Gold in the form of amalgam, silver as silver chloride, iron as hydroxide, copper as basic carbonate, zinc as hydroxide and lead as sulphate. Thus the process is designed to effect a chemical separation and concentration of the valuable constituents of an ore in a more economical manner than might be done in other ways.

A number of objections and queries will at once arise in the mind of the investigator as to the feasibility of the process. The complete precipitation and washing of salts in large quan-

tities; the economical regeneration and recovery of nitric acid; the completeness of the various reactions; the operation of a process with a gas under pressure; the economical operation of a process combining many separate operations; and the recovery of unfinished products which must stand further cost of reduction and refining—all these are points which call for proof by commercial operation on a large scale. The sponsors of the process, however, feel sufficiently assured on these points and claim that the method can be operated profitably.

Description of Plant.

The process naturally divides itself into three parts: The decomposition of the ore, recovery of the metallic salts and recovery of the nitric acid.

The decomposition of the ore is effected in a digester of cast iron lined with an acid-resisting mixture. This piece of apparatus is egg-shaped and supported in a vertical position. Within is an air lift agitator made of aluminium, connected by aluminium pipe with an air compressor outside. The proportional dimensions of air lift and digester are about the same as in a Pachuca tank. Above the digester, and tightly connected with it, is the ore bin, which can be tightly sealed. In the connection between digester and bin is a revolving cup feeder. In a suitable position above the digester is a receptacle for nitric acid.

The recovery of the metallic salts is accomplished in a series of precipitating and settling tanks, with such filtering and washing apparatus as may be needed. Ordinary amalgamating devices are provided for gold recovery, and a concentrating table is used for the separation of lead sulphate from the gangue.

The acid recovery system consists of a series of condensers and water towers. Aluminium pipe is used in all connections and cooling coils, and the tanks and towers are protected with acid-resisting mixtures wherever exposed to the acid gases. The water towers are constructed of paraffined redwood or cypress and filled with clean quartz. The first of the three towers is half full of water and the others are two-thirds full. At suitable places in the aluminium pipe are two indicators to show the condition of the gases at those points. These indicators are glass sleeves surrounding openings in the pipe, and the first is placed ahead of the first water tower and the second beyond the last tower.

Treatment of Ore.

Although the treatment of ore and recovery of acid are simultaneous operations, they will be described separately for the sake of clearness.

The process is worked intermittently on successive charges of ore and acid. Six tons of ore per charge is a convenient unit of capacity, and the time of treatment of each charge is between two and three hours.

The ore should be ground to about 30-mesh size, and contain at least 2.5 per cent to 3 per cent sulphur. If the sulphur content is below that figure the ore should be concentrated before treatment. At least 5 per cent sulphur has been found necessary if the reaction is to proceed without the application of heat. With 5 per cent or more sulphur the reaction with nitric acid generates sufficient heat for complete decomposition of the ore, but with less than 5 per cent it is necessary to heat the mixture. This is usually done by agitating with live steam instead of air for five or ten minutes at the beginning and end of the treatment.

The acid is run into the digester in quantity proportional to the amount of sulphur in the ore. It is claimed, however, that, if pyrite is present in considerable quantity, 5 per cent less than the theoretical amount of acid can be used, as the ferric salts act as catalyzers in oxidizing the sulphides to sulphates. It is customary to use acid of 1.15 to 1.42 sp. gr. The recovered acid has a sp. gr. of 1.25 to 1.35 and is used at that strength.

With the necessary charges of ore and acid in the sealed ore bin and digester respectively, the feeder is started which slowly delivers the ore in small quantities into the acid. At the same time the air agitator is started and the feeding and agitation continue until the sulphide minerals have been converted into sulphates, all of which are soluble excepting that of lead. Some sulphuric acid may be used with the nitric if the ore contains lime, or to prevent the formation of basic iron sulphate. It is claimed that the conversion of sulphides to sulphates is practically complete, only about 2 per cent of the nitric acid remaining as nitrates of the metals. The following equation is typical of the reaction on which the process is based.

$$3PbS + 8HNO_3 = 3PbSO_4 + 8NO + 4H_2O$$

Throughout the treatment of a charge of ore the digester and acid recovery system are under about 25 to 30 lb. pressure. When the reaction is completed the digested mass is drawn into a tank where the solution is separated from the gangue by filtration or decantation. The residue is washed and passed over an amalgamated plate and Pierce amalgamator for the recovery of the gold. The lead sulphate is then separated from the gangue by concentration on a Wilfley table, the sulphate flowing off as a slime and collecting in settling launders. Naturally it will be contaminated with the fine gangue formed in crushing the ore, but it forms a high grade lead product for smelting.

The metals in solution are precipitated successively in a system of tanks, and the precipitated salts are washed and dried. Silver first is precipitated with salt, giving silver chloride which must he reduced to metallic silver. Iron next is precipitated with pulverized limestone (30-mesh) giving a fine granular precipitate of ferric hydroxide which settles rapidly. This precipitation is carried out at a temperature of 55 deg. C., which is about the normal temperature of the solution at this stage of the process.

A remarkable claim is made in connection with the precipitation of iron and the subsequent precipitation of copper Ground limestone is used as the precipitant in each case, but the temperature is 20 deg. C. higher for copper than for iron It is claimed that at 55 deg. C. limestone will precipitate iron without copper, and that upon raising the temperature of the solution to 75 deg. C. the same precipitant will throw down copper as basic copper carbonate. Wherefore, after the precipitation and separation of iron the solution is heated by passing live steam through it until a temperature of 75 deg. C. is obtained, when copper is precipitated.

The iron precipitate has no value unless circumstances should warrant its use as a paint pigment. A market might be found for the copper precipitate as such, but in all probability if any considerable quantity were produced it would have to be reduced to copper.

With iron and copper removed the zinc is precipitated with milk of lime, giving zinc hydroxide which is to be calcined and smelted. The zinc precipitate will be free from iron but may contain some manganese. It would seem that, inasmuch as lime in some form is the precipitant for iron, copper and zinc sulphates, certain amounts of calcium sulphate must contaminate the precipitates. Calcium sulphate is soluble at ordinary temperatures in about 500 parts of water, the solubility increasing slightly until a temperature of 100 deg. C. is reached, when it decreases rapidly. Hence the salt will precipitate unless enormous volumes of solution are used.

The precipitating tanks for these operations are fitted with pipes for air agitation, bronze pipe being used in all precipitations where copper is still in solution. The entire precipitation system is preferably arranged in duplicate to facilitate cleaning up.

Nitric Acid Recovery.

The principle upon which the recovery of nitric acid is based is shown in the following cycle of reactions. As shown before, the treatment of a sulphide with nitric acid gives nitric oxide

as one of the products. Nitric oxide and oxygen unite to form nitrogen peroxide, which is soluble in water at ordinary temperatures, forming nitric acid and nitric oxide. The latter then combines with more oxygen and the cycle is repeated.

$$3CuS + 8HNO_3 = 3CuSO_4 + 8NO + 4H_2O$$
$$NO + O = NO_2$$
$$3NO_2 + H_2O = 2HNO_3 + NO$$

The oxygen necessary for the reaction is supplied by the air used in agitating the charge of ore and acid. The nitric oxide and oxygen are constantly being forced out of the digester through the acid recovery system, the oxygen being supplied in sufficient quantity so that the gases escaping into the air at the end of the system will show an excess of at least 2 per cent O. It is claimed that 95 per cent of the acid used is recovered.

The gases first enter a large narrow cylinder having about two-thirds the volume of the digester. The function of this so-called mixer is to allow time for thorough mixing of the gases and formation of nitrogen peroxide. Some nitric acid is formed here by reason of the fact that when the reaction in the digester is well under way steam will issue with the other gases, giving the right conditions for the formation of some nitric acid. It is claimed that about 10 per cent of the acid is recovered here.

The gases are next cooled in an aluminium condenser or cooling coil, where another 10 per cent recovery is made. Following the cooler is the first water tower, through which the gases bubble and where the greatest part of the recovery is made, amounting to 60 per cent. Between the cooler and first tower is placed an indicator, where the gases should show reddish brown in color. Following the first tower is another cooler where about 3 per cent of the recovery is made. Finally, the second and third towers make about 12 per cent recovery. The acid drawn from the mixer and coolers is 1.25 to 1.35 sp. gr., while that in the towers gradually builds up to 1.35 to 1.40 sp. gr. before it is drawn for use. The first tower builds up in strength most rapidly and when its contents are withdrawn they are replaced by those of the second tower, and so on. The gases issuing from the last water tower pass through a second indicator, where they should appear colorless if the system has been working properly.

The cost of chemical treatment is determined by the quantity of sulphur in the ore. The acid loss is claimed to be only 5 per cent, and the precipitants used are inexpensive. In addition there will be the usual cost for preparation of the ore, pumping and agitating solutions, filtering, drying and marketing the products.

The results of the first attempt to operate the process on a commercial scale will be awaited with interest

Fiber Testing Plant for Paper Manufacture.—Construction is under way on the paper machine house to form a part of the new laboratory of Arthur D. Little, Inc., engineers and chemists, Boston, Mass. This miniature paper mill will be L shaped, consisting of two rooms measuring approximately 25x35 and 25x52. Being located in the park system of Boston its exterior will be made especially attractive with brick and stucco walls and tiled roof. Although the entire equipment will be a miniature with a capacity of only 5 to 10 per cent of that of an ordinary machine, each piece of apparatus will be perfect in itself. The combination cylinder and Fourdrinier machine which is being made by the Pusey & Jones Company, Wilmington, Delaware, will be constructed along the same lines as the machines made by this company for the Forestry Service and the Bureau of Standards, but will differ somewhat in the method of driving as well as in the number of calendar rolls and suction boxes. Its length over all will be about 44 feet. The balance of the equipment will include two 50-lb. beaters with washers, pulp washer, Jordan, soda, sulphite and rotary digesters, steam generating plant, bleach dissolving system, vacuum pump, stuff chests and pumps.

The Grönwall Steel Refining Furnace.

By Thomas Duncan Robertson.

The development of the electric furnace for iron and steel manufacture is comparable in its rapidity to that of the automobile or the aeroplane. In 1904 there were only four electric furnaces in Europe, while to-day there are over one hundred working in different parts of the world, some reducing iron ores, the majority, however, being used to melt or refine the products of older metallurgical processes and produce the material generally known as electric steel.

It is only natural that many types of electric furnaces have been proposed from time to time; some have existed on paper, some in the experimental stage and others have made iron and steel on a commercial scale. The latter class is generally subdivided into induction furnaces and arc furnaces. The former, it would appear, are on the decline, although they are fairly efficient melting machines when of small size, yet it is found to be very difficult to accomplish with them any appreciable refining. On the other hand, in the arc furnace cold materials are easily melted and the fluid charge can be refined to a degree of purity unequalled by any known process used in the iron and steel industry.

At the present time several types of arc furnaces have proved themselves capable of making good steel economically.

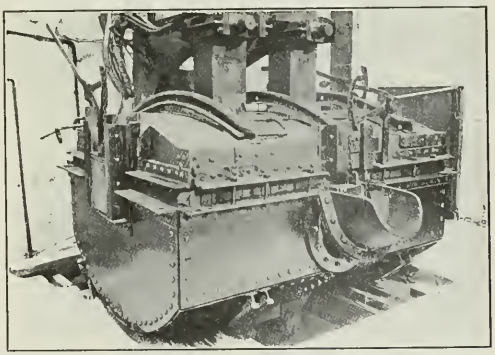


FIG. 1—VIEW OF GRÖNWALL STEEL FURNACE.

One of the latest types to be introduced—and one that possesses several distinct advantages over others—is the Grönwall furnace. This is the invention of three Swedish engineers—Messrs. Assar Grönwall, Axel Linblad and Otto Stalhane, of the Aktiebolaget Elektrometall of Ludvika, Sweden.

Having originally worked with various types of induction furnaces without great success, they decided to design an arc furnace which would embody the mechanical and electrical advantages of existing types, at the same time eliminating their weak points.

The invention consists in the method of supplying electric energy to the furnace, the form used being two-phase alternating current.

Furnace Construction.

As will be seen from the illustration (Fig. 1) the furnace has two carbon electrodes passing through the roof. These electrodes carry the two phases, the current arcing across from them to the charge, passing through this, and then through the basic lining to the neutral return, which is a carbon block fixed in the bottom of the furnace.

The iron tank of the furnace is lined with magnesite brick, and inside this lining the hearth of dolomite or magnesite is rammed. (See Figs. 2 and 3.)

The carbon block forming the lower terminal is set in a casting bolted to the furnace bottom, and good electrical con-

tact is secured by a special mixture which is used as cement. The top of this bottom electrode comes level with the brickwork so that it does not project into or in any way weaken the basic lining. The hearth of the furnace is not broken by any projections, being similar to that of an ordinary basic open-hearth furnace, repairing after each charge being effected in the same manner.

The roof consists of a channel iron framework holding silica bricks, and is detachable, so that when the bricks are worn thin it can be taken off and replaced by a new one, this

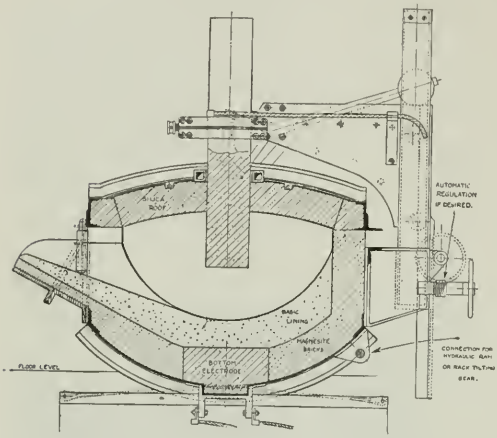


FIG. 2—SECTION OF GRÖNWALL STEEL FURNACE.

operation of changing only requiring about twenty minutes.

There is a working door at each end of the furnace and another one at the tapping spout. At the back of the furnace are the cranes for raising and lowering the electrodes. This is the only method of regulating the current and the electrode cranes can be operated either by hand or by automatic regulators.

The electrode holders for the electrodes passing through the roof offer three distinct advantages over the usual types employed:

1. The electrode holder at the same time grips the electrode and makes electrical contact.

2. The construction is simple and efficient; the holder consists of strong iron castings and T irons, a construction which will stand the rough usage to which it is exposed, and at the same time does away with complicated parts, such as springs, steel bands, etc., which are liable to get out of order.

3. The improvement is introduced of making the electrical contact low down on the electrode, thereby reducing to a minimum the loss of energy due to the resistance of the electrode.

The furnace is mounted on curved rails and is tilted by hand wheel gearing or by hydraulic cylinders. These rails rest on channel irons supported by strong foundations. The furnace construction, although simple, is cheap, strong and efficient.

Electrical Equipment.

The high-tension supply, if three-phase, is transformed by means of two single-phase oil-insulated self-cooling transformers, with Scott's connections to the tension suitable for the furnace, viz., 65 volts. Tappings are, however, provided on the primary side of the transformers for obtaining 50, 60, 70 or 80 volts.

The switch gear consists of an automatic oil switch, with cut-outs for overload, in circuit with the high-tension mains. On the low-tension side there is an ammeter on each phase and one on the neutral return. In addition a voltmeter with a three-way switch shows the voltage of each phase, and also that between phases. The power consumed is measured by a watt-hour meter.

The current is conveyed from the transformers to the electrode holders by means of flexible bare copper cables, and from the bottom casting to the transformers by similar cables.

Advantages of the Two-Phase System.

Practically all power supplies are either two- or three-phase high-tension current. It is, therefore, a great advantage of this system that such current can be used direct by means of static transformers without the intervention of costly running plant.

Where the available supply is two-phase the furnace is coupled directly to the mains through two single-phase static transformers, giving suitable voltage in the furnace. In the case of a three-phase supply, as mentioned above, the current is transformed into two-phase by means of Scott's system of connections. In this manner a great saving in first cost and a high economy in working are gained as compared with the rotary transformers (converters or motor-generators) required for converting three-phase current into single-phase current.

In the Grönwall furnace each phase is connected to one of the top electrodes. Each arc is, therefore, formed independently of the other, so that if one arc is broken the other remains, while in furnaces with arcs in series the breaking of one arc also breaks those remaining. The Grönwall system ensures steadier running. Assuming the furnace to take 500 kw, i. e., 250 kw per phase, if one arc breaks only about 250 kw are immediately switched off the mains, whereas with the arcs in series the whole 500 kw would be cut off, causing a much bigger fluctuation in the power supply. This feature is so marked that in Sheffield, England, where one of these furnaces is installed, the authorities at the municipal power

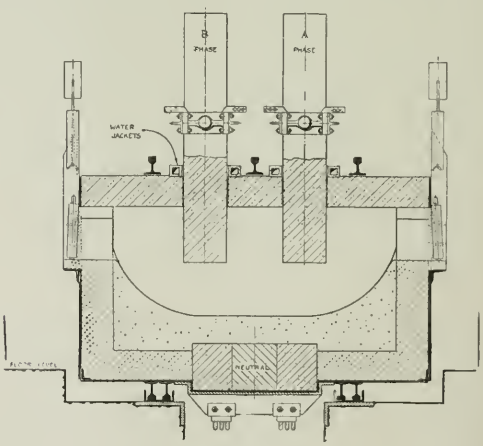


FIG. 3—SECTION OF GRÖNWALL STEEL FURNACE.

station permit the furnace to be started on a cold charge during their peak load.

In all arc furnaces the arc is, of course, the heat generator. During the melting stage the current passes from the electrodes to the charge, and as small pieces of this melt from under the electrodes from time to time the arcs are being continually made and broken. With a single-phase furnace with the arcs in series, this means a continual switching on and off of the entire current, with consequent disturbance of the supply system, loss of time in melting and loss of heat in the furnace by reason of the discontinuity of the supply of energy.

Apart from electrical considerations, it has been found that this two-phase system has decided metallurgical advantages.

The passage of the current through the basic lining uses very little energy, but it raises the temperature of the bottom sufficiently to prevent any metal cooling and solidifying in the bottom of the bath. The lining is not damaged in any way by the passage of the current, and experience has proved that there is no appreciable increase of wear and tear on the bottom.

The arrangement of two arcs in parallel with a neutral return through the bottom produces a vertical as well as a horizontal circulation of the metal in the bath. This facilitates chemical reactions by bringing all parts of the charge successively into intimate contact with the refining slag.

Due to this circulation, heat is taken away from the very hot region below the electrodes, so that the reflection of heat from this part on to the roof is much less than with single or three-phase surface heating, the result being to prolong the life of the roof. The roof of these arc furnaces is made of silica brick, so that when unduly overheated pieces of silica drop down into the basic slag the equilibrium of the reactions would be upset. The longer the roof can be made to last the greater is the reduction in cost and the less silica will pass into the slag per charge.

The process of steel-making in the Grönwall furnace is the same as that employed in the earlier and better known types of furnace. The quality of the resulting steel is so well known that it needs no emphasizing. It is generally admitted, however, that the absence of oxygen from the steel partly accounts for its remarkable properties. The process of deoxidation carried out in the Grönwall furnace is facilitated by the circulation of the charge referred to above.

The advantages of the Grönwall furnace may be summarized as follows: The utilization of two-phase current from a three-phase supply eliminates rotary transformers; an even distribution of heat is obtained throughout the charge, and at the same time a circulation of the molten metal is set up, resulting in improved refining conditions.

Toronto, Ontario.

The Remelting of Ferromanganese and the Deoxidation with Molten Ferromanganese.

In *Stahl und Eisen* of Aug. 24, 1911, appeared an article by J. Bronn and W. Schemmann, of Rombach, Lorraine, giving some valuable information about the use of molten ferromanganese.

In view of articles recently published about this subject (*J. Tr. Rev.*, 1911, March 16, page 526) the authors deem it necessary to give some practical advice. They fear that on account of the difficulties and failures experienced in remelting ferromanganese at the present time in cupola furnaces, open-hearths or electric furnaces, steel men may lose confidence in this process and abandon it, although if properly carried out it has very important advantages.

In 1906 the Rombach steel works began to remelt ferromanganese. The molten metal was poured into the ladle, together with the heat from the converter, thus preventing any waste of ferromanganese due to smaller pieces remaining in the slag. Any reduction of phosphorus from the slag back into the bath, which hardly could be prevented before, was avoided.

Samples analyzed from all the ingots gave not the least differences between the first and the last ingots. This proves a quick and uniform distribution of ferromanganese in the bath.

The casting into ingots proceeded very smoothly and in none of the 1000 heats with molten ferromanganese any difficulties were experienced in casting. Even "dead heats" could be poured.

It is a surprising fact that heats with such a low manganese content as 0.22 per cent to 0.25 per cent could be rolled easily and with only a small percentage of "bad ends."

This permitted a considerable saving of ferromanganese per ton of steel. Further small pieces of ferromanganese, such as result from long storage, could now be used without loss, while in former practice a good deal of the smaller pieces were always lost with the slag.

These favorable results induced the Rombach Steel Works to adopt the process on a larger scale. There were some difficulties involved in the problem how to melt the ferromanganese. Cupola and open-hearth furnaces could not be considered, since too much of the manganese would be oxidized or lost with the slag.

On account of the low boiling point of manganese (1900 deg. C., against that of iron, 2450 deg. C.) the ordinary electric arc furnace is also unsuitable. In a test made with a small electric furnace such an immense smoke of oxide of manganese was developed (the manganese evaporating first as such, but being at once changed into oxide) that the laborers were unable to operate the furnace and the trial was abandoned after half an hour.

Another difficulty was found in the method of feeding electrical energy to the furnace. Girod's bottom electrode did not work at all. After many trials in an electric furnace of 300 kw to 400 kw it was found that the bottom electrodes were exceedingly corroded and penetrated by the ferromanganese. Nearly a dozen different types and sizes of bottom electrodes were tried out, but none lasted more than three days.

Investigation of these steel-bottom electrodes showed a transformation into a high-percentage and very brittle ferromanganese. Observations made by Wüst and Felser have also proven that iron in contact with highly heated ferromanganese absorbs a great amount of carbon and especially of manganese. The use of the bottom iron electrodes was therefore discontinued.

Another problem had to be solved in the construction of the hearth of the furnace so as to prevent leakage. Since only small portions of the bath were taken from the furnace they had to be heated to a high temperature in order to prevent any freezing during transportation. But this hot and very liquid alloy in the furnace penetrates easily into the porous lining of the hearth and causes undesirable damage. These troubles were overcome by a special lining.

Numerous tests were made until a final satisfactory solution was found by using low-tension electric currents. In opposition to the ordinary arc furnaces with a drop of 45 volts to 75 volts between electrode and bath, a "far lower" tension was employed. (The exact voltage is not given in the article.)

The melting process then occurs smoothly without any of the troubles mentioned before. For example, the furnace contained more than 1.5 tons ferromanganese and after six days of continuous operation the furnace content showed not the slightest increase in percentage of iron or decrease in manganese.

With such low-tension current the distance between electrodes and bath surface could be reduced to a minimum without increasing unduly the consumption of electrodes (which was only 1 cm, or 0.4 in., per working hour as a weekly average). But the consumption of electrodes may be still reduced considerably.

It seems that in most cases the saving of manganese, which amounts to 35 per cent of the usual consumption of manganese, pays several times for the costs for remelting.

The results already obtained make it possible, according to the authors, to produce in one heat at reduced cost from Thomas iron an excellent quality of steel which may in many cases replace the ordinary basic open-hearth steel.

The Röchling Iron & Steel Works in Völklingen have also introduced this process in their mills. The remelting was undertaken in a small Roechling-Rodenhauser induction furnace, the results being satisfactory. R. J. W.

A Simple Method for the Determintion of Methane in Producer Gas and Enriched Water Gas.

By S. H. Worrell.

The constantly increasing use of producer gas and enriched water gas for power in gas engines suggests the necessity for either a daily analysis or a b.t.u. determination with a calorimeter. The analysis is necessary to determine, ofttimes, just where the trouble lies in the process of making the gas. The calorimeter is useful in determining the value of the gas when bought on a contract of a rated b.t.u. delivery.

If an analysis is to be made, the simpler the method used the better. The following method is in almost daily use in the University of Texas Testing Laboratory. The apparatus used is a modified form of that put out by the United Gas Improvement Company. Instead of the graduated burette with a bulb and three-way stop cock, a burette calibrated its entire length, without a bulb, and having a two-way stop cock is used with a water jacket to maintain a constant temperature.

One way of the cock is attached permanently, that is, it does not have to be removed during the analysis, to a potassium hydroxide pipette. This leaves the other branch open to attach the various other bulbs in succession during the analysis.

The method in detail is as follows. One hundred cubic centimeters of the gas at atmospheric pressure are taken. Then as each constituent is absorbed the burette is read, the amount so obtained is subtracted from the previous reading and this difference is at once the percentage of the constituent under consideration. This subtraction is all the arithmetical calculation necessary in this method, the final residue being nitrogen.

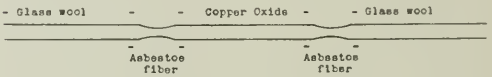
The carbon dioxide is first removed with potassium hydroxide solution contained in the permanent pipette, then the "illuminants" are removed with bromine solution, the gas being run back into the KOH pipette to remove the bromine before the reading is made. It is then run into the phosphorus pipette to remove the oxygen. The fumes of phosphorus pentoxide thus formed are soon absorbed by the water in the phosphorus bulb. The time consumed for these three constituents is from ten to fifteen minutes. A three-minute sand glass will be found to be quite a convenience in this connection. The carbon monoxide is then removed with cuprous chloride in acid solution. This particular constituent requires that the pipette be disconnected from the burette and thoroughly shaken for from two to three minutes to produce complete absorption. The gas is then transferred to the other side of the pipette, a double pipette being used, and shaken for another three minutes. I have never found that the second treatment was necessary but always make it as a precaution. The gas is then run into the KOH bulb again to remove HCl fumes, after which it is measured again, of course. It is next passed through a palladium oxide tube contained in a beaker of water heated to about 90° C. to 100° C. It is best to pass the gas slowly at least three times, renewing the hot water or reheating it as may be necessary. It usually takes about fifteen minutes to make this absorption. The gas is then brought back to room temperature by replacing the hot water with tap water and recirculating the gas, after which it is measured. To regenerate the palladium oxide tube simply attach to the filter pump and pull air through it as long as may be necessary; I usually let it run for about thirty minutes.

The gas, having been cooled, is measured and then run into the KOH pipette. If this treatment causes loss in volume it shows that not all the CO was absorbed by the CuCl solution and the percentage of CO is accordingly corrected by the diminution in volume here shown. Before the next analysis is made the CuCl solution is renewed.

There now remains methane and nitrogen. It is customary to run a measured amount of oxygen or air with a portion of this residue and explode it in an explosion pipette. In either case an involved arithmetical calculation is necessary. One never knows how much of this residue is nitrogen and how much is methane, hence cannot tell how much air or oxygen to add. If too much is added the result may be a dilution beyond the point of a successful explosion. If not enough is added the result is equally unsatisfactory. If the correct amount is added the result may be too satisfactory. Even if the apparatus is not destroyed by a successful explosion, the analyst acquires the habit of throwing the switch with one hand and dodging behind the other.

If the methane is known to be low as it *generally* is in producer gas and *frequently* is in enriched water gas, the hydrogen must either be left in (and there is no check on the CO unabsorbed by CuCl) or a measured quantity of hydrogen has to be added. This involves further arithmetical calculation with results equally as unsatisfactory as above.

To obviate all this the idea was conceived of burning the methane to carbon dioxide and water by passing it through red-hot copper oxide. A fairly thorough search through the chemical journals failed to reveal anything published on the subject. It was then decided to try it, which was done in the following manner. The tip of a hard-glass arsenic tube was cut off, thus giving a straight tube shaped as in the following illustration:



ARRANGEMENT OF TUBE.

The first section is loosely filled to the constriction with glass wool to take up the space and so restrict the volume. The constricted portion is then filled with asbestos fiber. The central section is then filled with wire oxide of copper such as is ordinarily used in organic combustion analysis. It should be ground in a mortar to about forty mesh and the fine dust screened out with an 80-mesh or 100-mesh screen. This allows free passage of the gas but fills the tube thoroughly. The outside of the tube is then wrapped around with about two layers of copper gauze or screen. This prevents the blaze striking directly on the glass and at the same time gives a uniform heating. The gas is then passed slowly through this heated copper oxide at least twice. When it cools to room temperature it should be measured. Any diminution in volume other than that due to the oxygen of the air enclosed in the tube when the combustion was begun is due to hydrogen not removed by the palladium oxide. The gas is then run into the KOH pipette to absorb the carbon dioxide formed. The loss in volume is the methane and is read direct in percentage, since one volume of methane on burning gives the same volume of carbon dioxide. There is, of course, an error introduced by the fact that the tube contains air and the oxygen in this is consumed, thus diminishing the volume by a certain amount. This amount can be determined by filling the apparatus with water, allowing the water to drain out and measure the volume so obtained. This total volume will hardly exceed 1 c.c., and the oxygen content in this cubic centimeter is the correction to be applied if desired.

The copper can be reoxidized by passing air through the red-hot tube. This method has been found applicable to percentages of methane as high as ten. If found necessary a second tube can be used or the same one reoxidized while still hot and the process repeated.

Testing Laboratory.
Bureau of Economic Geology and Technology.
University of Texas, Austin, Tex.

The Elmore flotation process at Sulitelma Mines produced in the month of April, 1911, a total of 3924 tons of concentrate containing 6.9 per cent copper.

Sizing with or Without Screens.

By Carl F. Dietz and Dyke V. Keedy.

The greater interest taken in recent years in the possibilities of the dry concentration of ores has brought out, aside from marked improvements in magnetic and electrostatic processes, several new and ingenious forms of dry concentrators, or rather tables, which have successfully overcome the difficulties of the early attempts at dry concentration. The prejudices against dry concentration in general, which in some instances may be justified by the many failures of the past, are most generally the result of ignorance or a disregard of the fundamental principles involved.

It is not the purpose of this paper to discuss the relative merits of dry vs. wet concentration. It may, however, be mentioned that each has its own field. Only a careful study of the mixture to be separated, together with local conditions, can determine which system is the more desirable. Such decision should, of course, in every case be verified by the treatment of a representative quantity of the material. While it is evident that close and accurate sizing as a requisite to successful dry separation, especially in the case of a complex mixture showing close specific gravity differences, is gradually becoming better understood, it remained for the advent of the Keedy ore sizer to make the application of a system of accurate and close sizing of practical importance. Reference may be had to *Engineering and Mining Journal*, Feb. 5, 1910, "The Keedy Sizer for Classifying Complex Ores."

This machine is based on diametric sizing, that is, the grades are actually measured by the aperture of the screens, limiting each grade, and therefore the ratio in volumes between the largest and smallest particles in any one grade is as the cube of the apertures of the limiting screens. This law, while it is applicable to all minerals breaking into massive particles, is directly affected by the coefficient of fracture. The latter is variable, and as yet extensive investigations have not resulted in determining a suitable factor for each of the usual minerals. Hopes, however, are entertained that something of value in this connection may soon be developed and published. It has been found in practice that the law of cubes gives very satisfactory results and can be relied upon with the exception of certain minerals, such as mica and specular hematite, which break into flat scaly particles, presenting a large area as compared to their volume.

Exception was taken to the cost of sizing an ore into, say, 12 or 15 grades because of the complexity of the apparatus and flow sheet, wear and tear and cost of screen renewals. When considering the old methods of using shaking or bumping screens or reels, the objections are well founded, but are no longer tenable with the more modern form of apparatus. The system is compact, noiseless, dustless and requires but small power. Attention required is at a minimum and screen renewal cost, while varying with the class of material treated, is well within the limits of expense which any given ore can stand if it is worth treating at all.

The objections to the cost of grading in this manner, even though actual operations in a number of instances prove them to be unfounded, still continue and have caused the minds of metallurgists and inventors to seek other means.

Recently there have been given to the mining and metallurgical world some details of a new and ingenious device designed to size ore into any number of grades from 10 to 30 without the use of screens. Details of this new machine, invented by Messrs. C. L. McKesson & B. P. Rice, of Colorado Springs, are presented in *Mining Science* of Sept. 14, 1911, and in Metallurgical and Chemical Engineering of October, 1911.

This new "screenless sizer," in the operation of which the laws of gravity are reported as having but a very slight influence, is capable of some rather remarkable performances, and the published results of an official test will bear some close mathematical analysis. It is barely possible that this machine performs a function as yet apparently not recognized by the inventors or their associates, and if such is true, the apparatus should have a wider range of usefulness than would be accorded to a machine intended to be used as only a sizer.

The measurement of efficiency of any sizing operation is determined by test screening any particular grade through screens which supposedly mark the limits of that particular grade.

In preparing a complex ore for dry table separation it is necessary to make as many sizes as may be demanded by difference in specific gravity between the minerals which vary the least, so that the most perfect stratification may result and that clean zones of minerals, with a minimum of accidental middlings, may arrange themselves on the tables; these conditions can result only when the ratio of volumes between the largest and smallest particles, in any one grade, equals or is less than the ratio between the specific gravity of the two closest minerals.

As an example, in the case of iron pyrite and zinc blende, in which the respective specific gravities are 5 and 4, let Vp be the volume of the pyrite particle, while Vb is the volume of the blende particle. Then

$$\frac{Vp}{Vb} = \frac{5}{4} = 1.25$$

As pointed out, grades in which such ratios exist can be produced by considering the apertures of the screens, and if S_1 is the aperture of the coarse screen while S_2 is the aperture of the fine screen, we have

$$1.25 = \frac{S_1^3}{S_2^3}$$

Of course, a series of screens from, say, 20 mesh to 200 mesh, which throughout would have its aperture cubes bear a relation to each other as 1.25 to 1, leads too far for one sizer and necessitates two machines, as some 22 sizes would be required.

It may be mentioned that in the case of a very complex Mexican ore such a series was employed with the most gratifying results, showing sharp lines of demarcation between the fields occupied by galena, pyrite and blende, so that it was not necessary to cut a zinc-iron middlings.

This, of course, is an ideal condition, but it was found that the system could be much simplified by making only 13 sizes and collecting the accidental pyrite-zinc and zinc-gangue middlings, which, by resizing together over a machine dressed differentially, that is, making between-sizes, prepared them for successful treatment for the reason that the particles were rearranged, and therefore they could not all again report themselves as middlings.

The McKesson screenless sizer is designed to make as many as 30 grades in one operation, which, if the ratios of volumes between the largest and smallest particles in any one grade should prove to be as above shown, would be admirably suited as feed for a dry concentrator.

The aperture dimensions of the screens used to test the various grades must be known, but for illustration we may arbitrarily use the following in discussing the results of the official test on the McKesson sizer.

Mesh.	Aperture inches.
24	.0342
28	.0282
30	.0268
34	.0229
40	.0185
44	.0172
50	.0145
66	.0102
82	.0079
97	.0058
116	.0047
139	.0033
172	.0028

The results on p. 587 show that, except in the two coarse grades, efficiency is based on the material reporting in the two heaviest screen products, which, when taken together, show a large volume-ratio between the largest and smallest particles lying between the limiting screens used in determining measure of efficiency.

The results of a test on the McKesson machine making 15 products are reported to be as follows:

Product No.	Screen.	Per Cent.	Efficiency.
1.	— 24	....	
	+ 30	15.60	
	+ 40	81.20	81.20
	— 40	....	
2	— 28	....	
	+ 40	93.75	93.75
	— 40	6.25	
3.	— 30	....	
	+ 34	10.92	
	+ 40	50.00	82.83
	+ 44	32.83	
	— 44	6.25	
4.	— 34	....	
	+ 40	28.13	
	+ 44	37.50	62.50
	+ 50	25.00	
	— 50	9.37	
5.	— 40	....	
	+ 44	7.28	
	+ 50	25.00	89.06
	+ 66	64.06	
	— 66	3.12	
6.	— 44	....	
	+ 50	6.25	
	+ 66	68.75	90.52
	+ 82	21.87	
	— 82	3.12	
7.	— 50	....	
	+ 66	37.50	90.63
	+ 82	53.13	
	— 82	9.37	
8.	— 50	....	
	+ 66	9.37	
	+ 82	62.50	87.50
	+ 97	25.00	
	— 97	3.13	
9	— 50	....	
	+ 66	6.25	
	+ 82	50.00	87.50
	+ 97	37.50	
	— 97	6.25	
10.	— 66	....	
	+ 82	37.50	84.37
	+ 97	46.87	
	— 97	15.63	
11.	— 66	....	
	+ 82	25.00	75.00
	+ 97	50.00	
	+116	21.87	
	—116	3.13	
12.	— 66	....	
	+ 82	15.63	
	+ 97	43.75	78.12
	+116	34.37	
	—116	6.25	
13.	— 66	....	
	+ 82	6.25	
	+ 97	37.50	81.25
	+116	43.75	
	—116	12.50	
14.	— 82	....	
	+ 97	25.00	
	+116	43.75	71.87
	+129	28.12	
	—139	3.13	
15.	— 82	....	
	+ 97	6.25	
	+116	18.75	
	+139	43.75	68.75
	+172	25.00	
	—172	6.25	
	Average		81.32

The efficiency, determined even in this most favorable manner, cannot be said to be high, and when considering the vitiating effect of the relatively large amount of "through" product recorded, and its detrimental effect on table work, the results cannot even remotely be compared to those obtainable by diametric sizing, when making even a lesser number of products and covering a wider range.

To quote from Metallurgical and Chemical Engineering, page 554: "It will be noticed that the grading efficiency in some of the products is lower than in others, but it is claimed that this can be remedied by adjusting the cutters to throw the oversize and undersize into the preceding and succeeding grades respectively, and thereby increase the efficiency of the sizing in *all* grades."

In the above statement the absence of such positive action as exists when sizing diametrically is clearly admitted.

That the efficiency shown in one grade may be improved by throwing the over and undersizes into the preceding and succeeding grades respectively is obvious, but on the basis of the published screen analyses of the products, instead of thus increasing the efficiency of all the grades, those into which the oversize and undersize are cut would be more likely to suffer, with the result that, while one grade would be improved at the expense of its tonnage, the two grades affected by the improvement of the one would probably show not only a lower efficiency, but would represent a correspondingly larger proportion of the whole. If the action of the machine were positive the explanation of the possibility of improving the grades would have some force which, in view of the results, is lacking.

Taking the basis of the efficiency calculation as given, and applying the law of cubes, we can calculate the ratios existing between the largest and smallest particle in any one product, giving the machine the advantage of using the screens specified, which limit the efficiency calculation. We have then:

Product No.	Limiting Screens.	Per Cent. Efficiency.	Vol. Ratio of Largest to Smallest Particle.
1	— 30 + 40	81.20	3.04:1
2	— 28 + 40	93.75	3.50:1
3	— 34 + 44	82.83	2.30:1
4	— 40 + 50	62.50	2.70:1
5	— 44 + 66	89.06	4.80:1
6	— 50 + 82	90.52	6.20:1
7	— 50 + 82	90.63	6.20:1
8	— 66 + 97	87.50	5.40:1
9	— 66 + 97	87.50	5.40:1
10	— 66 + 97	84.37	5.40:1
11	— 66 + 97	75.00	5.40:1
12	— 82 +116	78.12	4.70:1
13	— 82 +116	81.25	4.70:1
14	— 97 +139	71.87	5.40:1
15	—116 +172	68.75	4.80:1
	Average ratio		4.66:1

In the practical application of close and accurate screening we have adopted, as a standard, a series of 13 screens between 20 and 200 mesh conforming to a harmonic series, thus producing 14 products, including oversize and fines. The average volumetric ratio is 2 to 1, while the efficiency, when measured by the screen aperture actually limiting the grade, averages better than 90 per cent, the only drop under 90 per cent being in the — 150 + 200 grade, which, with the screen area usually exposed, varies between 80 and 85 per cent.

The percentage of oversize is practically nil and the particles passing through the fine limiting screen, in testing the grade, are usually of such a size as to just pass through the aperture, but much too coarse to go through the next screen of the series. If the efficiencies in testing the products from the Keedy sizer were figured in the same manner as those for the McKesson sizer products the former would show an average efficiency of from 95 to 98 per cent, as against 81 per cent for the latter.

The series as developed and used by us as a standard is the following, in which screen cloths have been selected of apertures conforming with the law of cubes:

Mesh.	Apertures, Inches.	Ratio of Volumes of Largest to Smallest Particle.
20	.0410	1.722 to 1
24	.0342	1.722 to 1
28	.0282	1.783 to 1
34	.0229	1.867 to 1
42	.0183	1.959 to 1
50	.0145	2.012 to 1
62	.0116	1.953 to 1
74	.0089	2.214 to 1
86	.0068	2.242 to 1
109	.0054	1.996 to 1
125	.0041	2.284 to 1
150	.0032	2.103 to 1
200	.0026	1.865 to 1
Average		2.000 to 1

A comparison between these ratios and those of the screenless sizer work, entirely aside from the considerably better efficiency of diametric sizing, clearly indicates which products may be expected to yield the better table results.

The results of the McKesson test show that in a general way the efficiency varies inversely as the ratio of maximum to minimum particles in the grades, excepting in the extreme coarse and fines, but the most important characteristic revealed is the gradual gradation by size almost identical in its character with, but somewhat more emphasized than, that resulting from feed-

ing an unsized pulp over a dry table of the Sutton, Steele & Steele type.

It may be especially pointed out that in every instance in the screen analyses the coarse *limiting* screen used is of larger aperture dimension than the fine limiting screen of the preceding grade. The efficiency of products 8, 9, 10 and 11 is measured by exactly the same screens, the only difference in the grades being that they grow finer by reason of the fact that the oversize is gradually reduced while the undersize increases. The latter is also true of products 12 and 13.

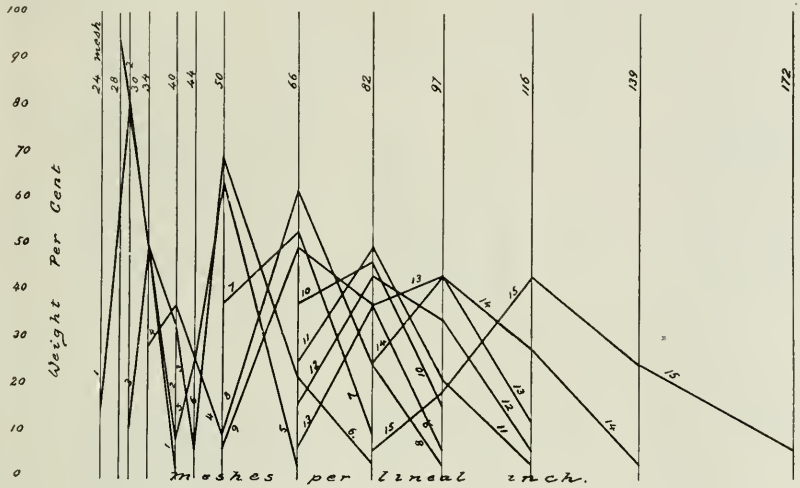


FIG. 1.—GRAPHICAL REPRESENTATION OF SCREEN ANALYSES OF PRODUCTS FROM A MC KESSON SCREENLESS SIZER.

This fact proves that the products very considerably overlap, and thus the sharp line of demarcation existing between diametrically sized grades is wholly lacking in the new method of sizing. The overlap of the products is graphically shown on Plate I, and it is to be clearly noted that no sharp line exists between any two or even three of the grades, while the overlap, in specific cases, is sufficiently great to completely destroy the effect of sizing. For example, products No. 7 and No. 15 are so poorly sized that $-82+97$ mesh material is contained in each.

With an average recorded efficiency of but 81 per cent, and that, too, on an average particle ratio of 4.66 to 1, the actual average particle ratio on the whole product exceeds 6 to 1, which is far too much to permit of anything like desirable table work.

The above discussion is, of course, all predicated on the statement that specific gravity plays an unimportant part in the operation of the McKesson apparatus.

That a machine of this general type will operate as a sizer has been pointed out, but the sizing is of an extremely indifferent quality, and specific gravity actually does play a part well worthy of notice. In one instance, an unclassified ore was treated on a Sutton, Steele & Steele table, primarily for the purpose of noting the sizing effect. The tendency was marked but the gradations along the periphery of the table were so gradual as to be characterized as only an abortive attempt at sizing. The feed to the table and the four products caught at the periphery gave the screen analyses on top of next column which are graphically represented in Plate II.

These results were obtained on a magnetic iron ore, and the products "b," "c," "d" and "e" clearly showed that the finer heavy magnetite particles had a distinct tendency to associate themselves with coarser gangue particles—the result of the ever-operating law of gravity. If 15 products had been cut, the result would have been quite as perfect as that of the McKesson machine.

Mesh.	Feed.	Products B	C	D	E
— 50					
+ 58	1.50	4.1	.10	.25	0
+ 68	.50	2.3	.45	.25	0
+ 90	9.50	16.0	10.00	7.25	2.50
+100	12.00	28.7	13.00	13.75	3.50
+140	14.25	18.0	19.25	14.75	5.00
+163	21.50	10.1	16.25	11.25	6.00
+200	18.00	14.1	24.00	26.75	32.25
—200	20.50	6.5	16.00	24.00	49.00

Another notable instance—in the treatment of a mixture of galena, pyrite, zinc and fluorspar, it was invariably found that the finer galena traveled under the pyrite, the finer pyrite under the zinc, and the finer zinc under the fluorspar. In air, a particle of zinc need be only 25 per cent greater in volume than a particle of pyrite to weigh the same, while a particle of fluorite need be only 30 per cent greater in volume than a zinc particle to weigh the same.

Now these particles of equal weight are acted upon equally, and necessarily produce a middlings which when screened over a cloth of proper aperture results in a concentrated product either on or through, or both, the heavier mineral reporting in the finer product.

The published screen analyses of the products from the McKesson sizer do not indicate that specific gravity plays no part in the operation, and the question obviously arises whether this machine does not perform a function of greater importance than the one which its inventors desire it to perform. Certain it is, on the basis of the recorded efficiencies and the evident gradual classification of the products, that the name screenless sizer is a misnomer. As far as the usual understanding of sizing is concerned, this machine is not a sizer, as its products overlap far too much. It would be more appropriate to call the apparatus a dry classifier, which in effect it is.

Mr. Charles J. Downey, in *Mining Science*, Sept. 14, 1911, page 247, states: "Concerning the theory of the McKesson method, I have no disposition to enter into a discussion of it, beyond what has already been said. An outline of the laws governing the selective action, in which differences of specific gravity play but a very small part (and then only in the case of widely different minerals, according to the engineers) will be the result of systematic and prolonged experiment. That the sizing is effected to a high degree of efficiency is apparent to an observer, while the operation of the machine has all the aspects of simplicity."

Now going back to the thought expressed above that this machine is essentially a dry classifier, all speculations vanish as to a new principle having been stumbled upon, and viewing the action as one of classification is the more tenable, when considering the analogy between the operation of the McKesson machine and the Sutton, Steele & Steele dry table.

It is difficult to conceive of one making separations between minerals of equal size, on the basis of their specific gravity differences, while the other collects together the various minerals according to their size, regardless of their relative specific gravities. These two thoughts are distinctly contradictory.

If the McKesson sizer acts in the nature of a classifier, which is exactly what a table does (only the new machine by means of the deflectors and discharge arrangement takes advantage of and emphasizes the action), it must be shown by the character of the screen products of each grade.

A hydraulic classifier when making 15 products appears to deliver a fine string of perfect sizes, however a screen analysis and assays of the screen products reveal the true action, which is classification both by size and specific gravity.

In each grade it will be found that the lightest mineral reports, to the greatest extent, in the coarsest grade; while the heaviest mineral will be found, to the greatest extent, among the finer particles of the particular product under consideration.

This same action is to be expected in a machine of the McKesson type.

It can readily be verified by testing a mixed sulphide ore and screen—analyzing the products, just as was done in the reported official efficiency test.

The investigations should, however, be carried further, and each of the screen products should be assayed for the principal metals contained in the mixture, thus readily establishing the actual facts of the operation.

We anticipate that results of such an investigation will show each product to be made up of coarse gangue, finer zinc, somewhat finer pyrite, and still finer galena. If this explanation of the operation of the new machine is found to be true, we will no longer be confronted with a new and paradoxical theory to worry the research metallurgist, but will realize the ease with which appearances may deceive and the danger of a too ready acceptance of an apparent overthrow of fundamental principles that have served us long and satisfactorily.

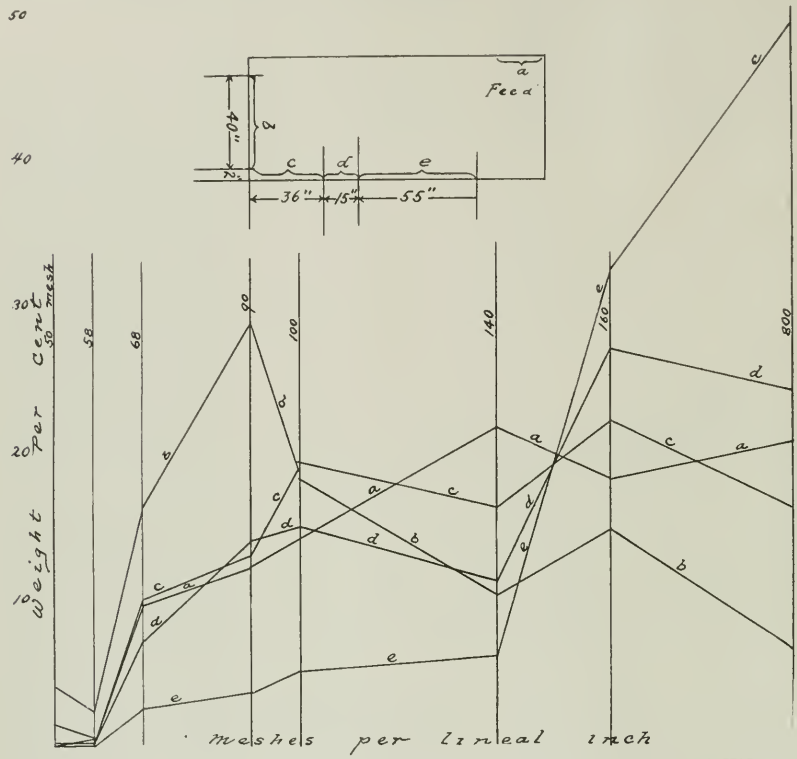


FIG. 2.—GRAPHIC REPRESENTATION OF SCREENLESS ANALYSES OF PRODUCTS FROM A SUTTON, STEELE & STEELE TABLE, WHEN FED WITH UNSIZED ORE.

The diagram on the top represents the Deck of Sutton, Steele & Steele Table, showing Peripheral Discharge of Products b, c, d, e.

The development of a dry classifier should have a wider field of application than a sizer, if its capacity is sufficiently high, since, if the machine acts as herein discussed, its products should make an admirable feed for a wet table.

That the new machine is not in any sense a sizer of mixed minerals seems well established, and that it follows the laws of classification in which specific gravity plays an important part is strongly indicated by an analysis of the published results.

We may express a hope that the inventors will shortly supply the metallurgical world with further, more complete data, preferably along the lines suggested, including a screen analysis of the feed to the machine for purposes of comparison, so that speedily a more definite understanding of the principle involved in the performance of this ingenious new contrivance may be had.

Boston, Mass.

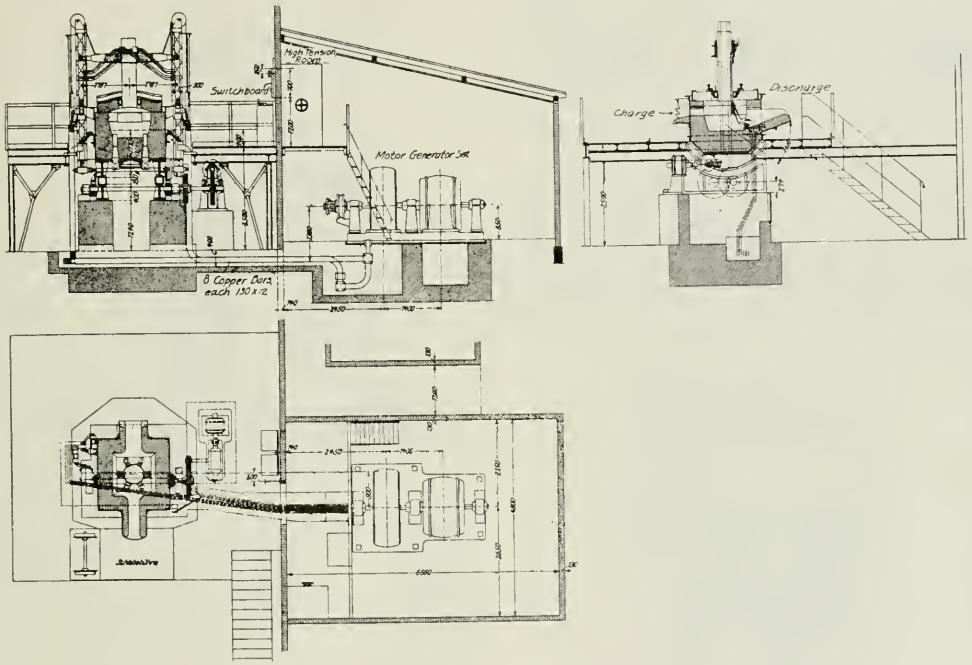
Producer gas is the source of power for seven mines in one of the oldest mining districts in Mexico, viz., in the district of Sultepec in the State of Mexico. Electric power has recently been brought into the district and is furnished at a cost of $170 per hp-year (presumably Mexican currency.) The mine pays for the construction of the substation, and the power discounts 25 per cent of the cost from the power account until the line is paid for.

The Eclipse filter is an Australian device for filtering cyanide or other valuable solution from ore pulp. It consists of a wooden frame 28 ft. long, 4 ft. wide, and having a fall of 3 ft. in its length. The filtering medium is an endless three-ply rubber belt, 3½ ft. wide, with numerous holes punched through it. Over the belt, and attached to it at the sides only, is a strip of filter cloth. In the middle of the machine is the filtering or squeezing gear, which consists of a series of cast iron rollers. The belt travels in a horizontal position except as it passes between the squeezing rollers, when it assumes a vertical position, being folded along its median line. The pulp is fed onto the belt at the upper end of the machine and as the belt passes between the rollers the solution is squeezed out of the pulp, the latter forming a cake which is discharged at the lower end of the machine. If the cake has to be washed it must be passed through a second machine. The capacity is about 900 lb. per revolution, or 650 tons daily. It requires about 3 hp. One is installed and working in the Associated mill, Kalgoorlie, and is more fully described in the August issue of the *Mining & Engineering Review* (Melbourne).

Manufacture of Steel in the Girod Electric Furnace.*

BY DR. A. MUELLER.

The Gutehoffnungshütte has installed a three-ton experimental Girod furnace in their Martin Works No. 1 for the purpose of electrically refining molten steel from their 25-ton open-hearth furnaces.



FIGS. 1, 2, 3.—ARRANGEMENT OF ELECTRIC FURNACE PLANT.

Figs. 1 to 3 show the equipment of the entire electric furnace plant, and the various details of the furnace and the electrical equipment. Fig. 4 is an outside view of the furnace. Only small details have been changed in the original design of Girod. The single-phase current for the furnace is furnished by a motor-generator consisting of a 3000-volt 95-amp. 575-hp three-phase induction motor (with a power factor of 0.92) and a single-phase 75-volt, 6700-ampere, 25-period, 500-kw generator (with a power factor of 0.8) with exciter unit.

These machines, as well as all conductors carrying high-tension currents are placed in a separate room adjoining the furnace room. The electrical conditions can be watched from the furnace by means of ammeters and voltmeters in the low-tension circuit and a kilowatthour meter in the high-tension circuit. The switchboard at the furnace contains the excitation-current regulator for the furnace voltage, the manual and automatic regulator for the electrodes, and the device for tilting the furnace.

The arrangement of the electrical-conductors is of great importance. There are three methods in use for conducting the current to the carbon electrode at the top and the steel electrode in the bottom, employed in the Girod furnace.

In method I (Fig. 5) the shortest path from the motor-generator set to the carbon and steel electrodes has been chosen; all of the cables are on the side of the furnace which faces the motor-generator.

In method II (Fig. 6) the cable to the carbon electrode is divided into two sections which run parallel to each other. The steel electrode is connected by the shortest path with the motor-generator set.

Method III (Fig. 7) is the one used since August, 1910, at the Gutehoffnungshütte. The current is conducted to the carbon electrode in a manner similar to method II. But while in methods I and II the bottom steel electrode is insulated from the furnace body itself (only the metal bath being in the

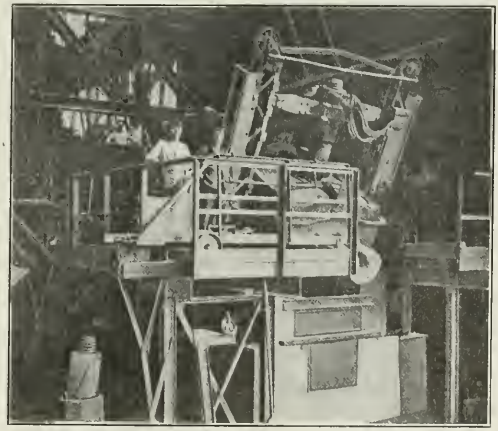
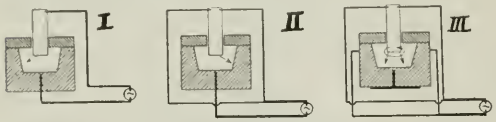


FIG. 4.—OUTSIDE VIEW OF GIROD FURNACE.

*Translated from *Stahl und Eisen*, July 20 and Aug. 3. Slightly abstracted.

electric circuit), the steel electrodes are here electrically connected with the furnace body. The arrangement of the conductors is symmetrical around the furnace.

With the first two methods the electric arc is deflected towards the side marked with the arrow, which results in a non-uniform heating of the steel bath and a rapid destruction of that side of the furnace which is exposed to the excessive direct heat of the arc. Method III avoids all this.



FIGS. 5, 6, 7.—ARRANGEMENTS OF ELECTRICAL CONNECTIONS TO THE FURNACE.

In view of the currents of high amperage and low voltage which with the Girod furnace is operated, there are considerable energy losses in the low-voltage electric conductors depending both on the resistance and the reactance. Method I, with its short conductors, was originally adopted to make these losses small. But the destruction of the furnace walls and roof on the side to which the arc was blown as indicated by the arrow was so excessive that it was cheaper to incur the

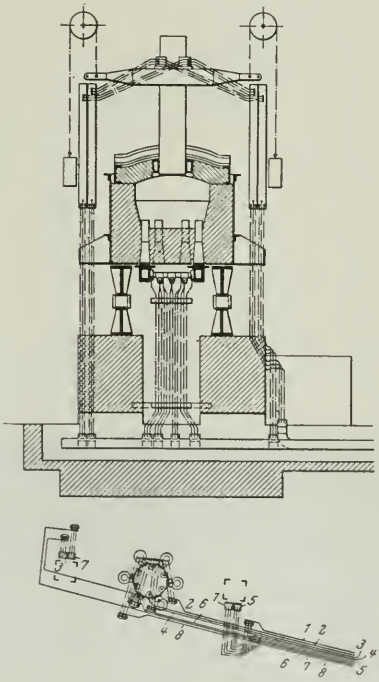


FIG. 8.—OLD ARRANGEMENT.

expense for greater lengths of cables and the losses in the same than the expense for the furnace repair.

It is good practice, therefore, to use as low a frequency of the alternating current as possible, to avoid a material of high permeability for the furnace construction and to try to keep the iron masses as far as possible out of the electric circuit.

The electric arc is forced to distribute its heat uniformly over the bath by arranging the cables symmetrically around the furnace as in Fig. 7, method III. In this way the magnetic fluxes, formed by the iron shell, are reduced to a minimum. The magnetic field around the carbon electrode becomes uniform, the heating effect of the arc becomes uniform, and the current passes through the charge in a more regular order.

The advantages of this arrangement of the cables, from an electrical and metallurgical standpoint, are the following:

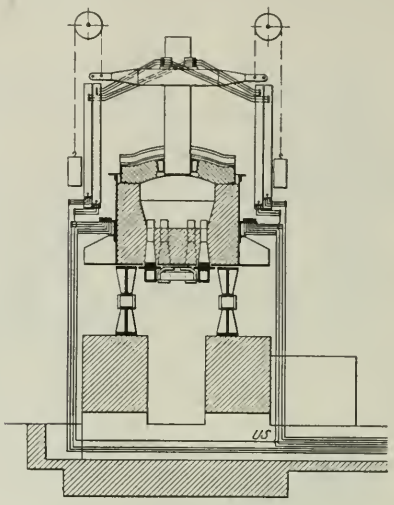


FIG. 9.—NEW ARRANGEMENT.

1. The electric arc circling about the periphery of the carbon electrode causes a strong agitation of slag and metal. This accelerates the speed of reaction between the slag and the iron bath and therefore reduces greatly the time of refining.

2. The arches of the roof as well as the furnace walls receive a more uniform radiation from the electric arc and last longer.

3. The saving in energy consumption is 10 per cent compared with the former arrangement.

4. Copper bus bars can be used instead of cables.

5. The metal bath is heated more uniformly throughout and the resulting product is more uniform.

6. Current interruptions due to the rupture of the arc are avoided, eliminating the resulting rushes of current on the motor-generator set and allowing an easier melting of cold charge than with method II.

7. The consumption of the carbon electrode is more uniform, while with the former arrangement one side of the electrode was consumed more quickly than the other, resulting in greater expense for electrodes.

Fig. 8 shows the older arrangement corresponding to Fig. 6, method II. The current passed in a 23-ft. long circuit through the copper bus bars, marked 8, to the furnace. One half of these bus bars was connected with the bottom electrodes by means of twelve cables, the other half was distributed on both sides of the furnace. With the other electrode connection was made from the conduit both by means of cables and by bus bars. Three cables led on the right side from bus bars 1 and 5, three on the left side of the furnace from bus bars 3 and 7 up to the carbon electrode, while from bus bars 2, 4, 6, 8, three cables each led to the steel electrodes which were electrically connected with each other but insulated from the furnace body.

In the new arrangement (Fig. 9) the six steel electrodes are electrically connected by means of a copper ring and plate with each other and with the furnace body. The entire current is

conducted close to the furnace by eight parallel copper bus bars, and divides at the current distribution point U S in such a way, that opposite currents pass alternately along both sides of the furnace to the pivoting point, where each bus bar is connected by three cables (more recently, by flexible copper bars) either with the furnace body or with the carbon electrode, as clearly shown in Fig. 9. This arrangement with the low voltage of operation overcomes all insulation difficulties and has reduced induction currents considerably. The rotating magnetic fields around the electrode are uniform and very small. This arrangement also permits an easy construction and a saving of cables.

For the first 225 heats the furnace was operated only during the day and in many cases it was possible to tap the furnace five times in 12 hours under favorable conditions (See Fig. 13). In continuous operation it is possible, working day and night, to obtain 25 tons of steel from 8 tappings.

Since the open-hearth furnaces were only tapped every four hours, the electric furnace could not be operated continually. About 3 tons of the metal tapped from the open-hearth was poured into a heated ladle on a weighing scale, and then charged into the electric furnace.

But this method was found to be too expensive, since the temperature drop of the charge from the open-hearth to the electric furnace resulted in a rather high fuel and energy consumption. It was, therefore, finally decided to use a 23-ft. long launder which allows direct charging without any trouble. The weight of a charge can be adjusted with an accuracy of 200 kg. (400 lbs.) by the eye or by a watch.

The operation of the electric furnace is in the main like that of the basic open-hearth furnace, the metallurgical reactions being essentially the same, except the deoxidation. During the charging process, the current is 4000 to 6000 amperes; after the charging launder has been removed from the furnace by a crane, the amount of ore and lime required for the metallurgical reactions is shovelled into the furnace.

The combustion of the phosphorus is now started by a gradual increase in the temperature, while simultaneously the amount of carbon still present is reduced down to a minimum by oxidation. A portion of the sulphur is also removed during this period. This period of refining by oxidation is, of course, the longer the higher the carbon and phosphorus content of the charge.

When this period is over, the oxidation slag is removed by tilting the furnace and the last traces of the slag are carefully scraped off. Then petroleum coke or carbon electrode remnants are added for recarburising and a deoxidation slag is formed by adding lime, fluor-spar, and ferro-silicon. The deoxidation and desulphurisation of the bath now proceed and with proper operation a sulphur-content of 0.01 per cent can be guaranteed. After complete refining, the tap hole is opened, the current cut off, the entire furnace with the carbon electrodes tilted 40° from the horizontal, and the contents poured into a 4-ton ladle.

Lining.

The lining of the furnace must be constructed and maintained with great care. The arches are chiefly subjected to destruction. With the new arrangement of the conductors, which has resulted in greater uniformity of heat radiation from the arc and therefore greater uniformity of the wear and tear of the lining, the silica roof with attached cooling ring will stand 60 or 70 heats. The slightly curved furnace roof of silica brick reinforced by iron bands is insulated from the side walls of the furnace by a thin layer of asbestos and a new roof can be substituted within 20 minutes.

It should be mentioned that the contact areas of the silica roof with its magnesite brick edge resting on the basic walls of the furnace do not reach such a high temperature as to make these materials so-called electrical conductors of the second class, which would result in leakage of the current. Moreover, the carbon electrode usually does not come in contact with the roof-cooling ring so that the insulation is only used as an additional safeguard.

The hearth and walls were at first lined with magnesite, but dolomite is now used.

The ratio of the cross-section of the steel electrodes (Fig. 10) which are embedded in the bottom of the furnace chamber, to the rest of the bottom area is 1 to 16.

A repair of the bottom does not become necessary until it is required to repair the walls. The walls are attacked strongest at the slag line, but the damage can be repaired between charges. After 120 heats the walls must be re-lined. The hearth bottom must also be repaired then and filled up again to the original level, since it has meantime become about 2 in. deeper at the center (where the temperature is highest) than at the periphery.

The cost of an entire dolomite lining is about marks 350 ($87.50), that of an entire magnesite lining marks 700 ($175). Dolomite has proved so efficient that it is now used exclusively. The total maintenance cost for the lining per ton of steel is about marks 1.20 (30 cents.)

Heat Losses Due to Cooling.

Hearth-electrode cooling is still little understood and has even been called a great drawback to the efficient operation of

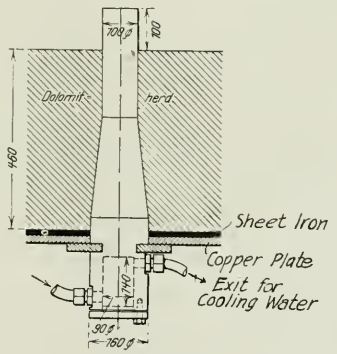


FIG. 10.—WATER-COOLED STEEL ELECTRODE.

the Girod furnace. Fig. 10 shows that only that portion of the soft steel bars which projects below the furnace body, is cooled by water, the bars being connected by a common pipe line.

The difficulties of maintenance of the lining of the Girod hearth are also often exaggerated. The dolomite hearth rammed by compressed-air rammers has lasted through more than 1000 charges and has never given any trouble in spite of different lengths of heats and very different compositions of the charges.

The estimate of the heat losses in the cooling water is often too high, so that the following calorimetric figures should be of interest. The hearth poles were cooled by water running at the rate of 4.9 liters per minute (1.2 gallons per minute), using 637 liters (168 gallons) for a heat of 130 minutes.

The temperature of the water flowing to the bottom electrode was 8° C, that of the water leaving the electrode was 21° C in the first 85 minutes and 23° C in the following 45 minutes.

The heat consumption, due to the cooling of the bottom electrodes was, therefore, $4.9 \times 85 \times (21 - 8) = 5414$ calories (kg), plus $4.9 \times 45 \times (23 - 8) = 3307$ calories (kg), hence the total loss of heat 8721 calories (kg), equivalent to an energy consumption of 10.1 kw hours or 13.7 horse-power hours.

Since according to Joule's law the total energy consumption is equal to $C = 0.2386\ E\ J\ t$ calories, we have in this case with a tension of $E = 55$ volts and a current $J = 4800$ amperes during the total time $t = 7800$ seconds an energy consumption $C =$

$0.2386 \times 55 \times 8400 \times 7800 = 859{,}819$ calories $= 1004$ kw hours (measured by the meter) $= 1364$ hp hours. Hence the loss of heat due to cooling the bottom electrodes is 1.01 per cent of the total energy consumption and when producing 3500 kg (7700 lb) of steel, about 2.9 kw hours or 4 hp hours are consumed per ton of steel for cooling the bottom electrodes.

Let us now calculate the heat loss due to cooling the carbon electrode and the arched roof. 0.2866 liter per second (or 4.5

FIG. 11.—POWER FLUCTUATIONS DURING A HEAT.

(A tapping, B, C, D additions, E carburization, F slagging, G slag readily fluid, H bath boiling, I addition of ore, J slag not fluid, K charging.)

gallons per minute) pass through the cooling ring for the roof, or 2236 liters (590 gallons) for a heat of 130 minutes.

The temperature of the water entering the cooling ring was 9° C, that leaving it was in the first 70 minutes 21° C, and in the following 60 minutes 25.5° C.

This gives a total loss of heat of

$$\frac{2236 \times 70 \times 12}{130} + \frac{2236 \times 60 \times 16.5}{130} = 31.47 \text{ calories, equal to } 36.7$$

kw hours or 49.1 hp hours, so that the loss of heat due to the cooling of the carbon electrode is 3.65 per cent of the total

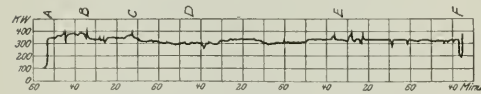


FIG. 12.—FLUCTUATIONS OF CURRENT WITH A HEAT IN THE FURNACE AFTER HAVING BEEN COOLED FOR 12 HOURS.

(F charging, E decarburization, D slagging, A, B, C additions.)

energy supply, 3500 kg. (7700 lbs.) of steel were produced so that 10.5 kw hours or 14.2 hp hours were lost per ton of steel for cooling the carbon electrode.

The Girod furnace with one carbon electrode requires about 0.65 cubic meters of water per ton of steel for cooling the carbon electrode, which is equivalent to 9000 calories. To cool the steel electrodes in the bottom the furnace needs 0.20 cubic meters of water per ton of steel, which is equivalent to 2500 calories. This heat loss is insignificant when compared

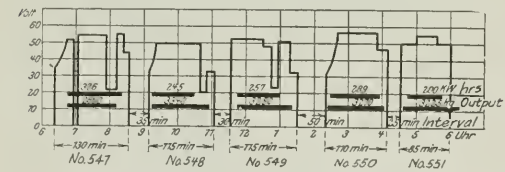


FIG. 13.—FURNACE OPERATION IN 12 HOURS.

(Desulphurization from 0.040 to 0.012%.)

with the radiation losses attendant upon charging, slagging and tapping.

Further, the above figures show that the cooling of the steel electrode requires only one-fourth of the amount of water needed for the cooling of the carbon electrode.

If the water flows at the rate of 5 liters per minute (1.3 gallons per minute), an equilibrium condition is established in the heat exchange as soon as the bottom steel electrodes have melted down to a point which is 20 mm (0.8 in.) below the level of the hearth. These liquid pole points which are gradually solidifying towards the lower end have the composition of the steel bath to be refined. No chemical change is thereby caused in newly charged metal, as they are emptied when the bath is tapped. The pits in the poles are then again filled with newly charged metal. The effect of cooling is a uniform fusing of the pole points, and a homogeneous current density through the same.

On account of the uniform current density and the uniform heating effect of the electric arc, the bath is heated on the hearth surface and the upper layers in such a way that a sticking of steel in the originally cold corners of the hearth is soon eliminated.

Electrical Conditions.

The quiet melting of a cold scrap steel charge has been recognized as a great advantage of the Girod furnace. Figs. 11 and 12 give the fluctuations of the kw consumption as plotted by a recording wattmeter, starting with a liquid charge. The regulation of voltage and current is almost always automatic; hand regulation (with the automatic regulator disconnected) is used only during slagging.

Influence of Temperature and Size of Furnace on Energy Consumption.

The furnace size, the length of intervals between successive heats, the cooling of the furnace during these intervals, the condition of the charge, the desired quality of the end products and other matters can change the energy consumption very considerably.

Fig. 14 shows the influence of the weight of charge in kilograms upon the specific energy consumption in kilowatthours per ton. The results are averages taken from 500 successive

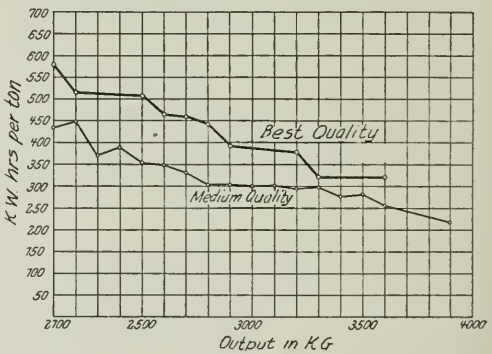


FIG. 14.—SPECIFIC ENERGY CONSUMPTION AS FUNCTION OF WEIGHT OF CHARGE.

heats, including alloy steels. The energy consumption depends a great deal on the weight of the charge and decreases in large furnaces.

The loss of heat through radiation and conduction as also the fluctuations of temperature decrease with increasing size of furnaces. The temperature is much more easily regulated in larger furnaces than in smaller ones.

The minimum economical weight of charge is about 2800 kilograms in Fig. 14, the specific energy consumption not changing materially with increasing weight up to 3300 kilograms.

The specific energy consumption depends still more on the temperature of the furnace and this depends greatly on the length of the intervals between charges. Fig. 15 shows the decrease of the temperature after tapping; this decrease is very rapid compared with that of an open-hearth furnace, the temperature of which sinks down to 1000° C after a 25-hours pause and which cools afterwards at the rate of 20° C per hour. Under such conditions it is possible to get a high thermal effi-

ciency with the Girod furnace only, if it is charged, at the latest, 20 minutes after the last tapping. The interior of the furnace is then still at about 1300° C. The energy consumption per ton is then 188 kw hours for medium-quality steels and 270 for high-quality steels.

Under very favorable conditions it was possible to refine 3654 kg (8039 lbs.) of steel with an energy consumption of 158 kw hours per ton, including refining by oxidation, deoxidation, and desulphurization. Large Girod furnaces are more favorable in this respect, as they can be charged immediately after tapping, i. e. while they are at a temperature 200 degrees higher.

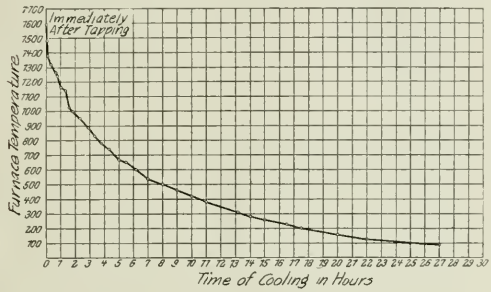


FIG. 15.—COOLING OF GIROD FURNACE.

Metallurgical Reactions.

Fig. 16 is a diagram of the chemical reactions during a heat in the Girod furnace and illustrates the changes of the carbon, manganese, phosphorus, sulphur, and silicon contents.

The results of the analyses of 19 samples are here plotted. Sample 1 is that of the charge, sample 2 was taken after addition of 15 kilograms of ore, sample 3 after addition of 50 kilograms of ore. Sample 4 and 5 were still taken during the period of refining by oxidation, sample 5 just before removing the oxidation slag.

Samples 6 to 19 were taken during the deoxidation period. After addition of 16 kilograms of Girod carbons and 5 kilograms of ferromanganese sample 6 was taken. After addition of 12 kg ferrosilicon (50 per cent Si) and 40 kg refining slag

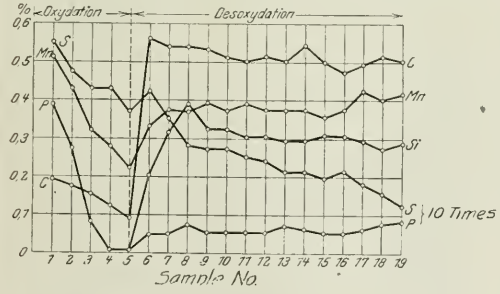


FIG. 16.—ANALYSIS OF C, MN, P, S IN 19 SAMPLES OF METAL DURING ONE HEAT.

sample 7 was taken. After addition of 4 kg ferrosilicon sample 8 was taken; after addition of 20 kg refining slag sample 9; after addition of 3 kg. ferrosilicon sample 10; after addition of 2 kg. powdered petroleum coke sample 11; after addition of 1 kg ferromanganese (80 per cent Mn) sample 12; after addition of 8 kg lime sample 13; nothing was added when sample 14 and 15 were taken; after addition of 10 kg lime and fluorspar and 2 kg powder petroleum coke sample 16 was taken; after addition of 2 kg. ferromanganese sample 17; nothing was added after that, sample 19 being the final sample.

Oxidation Period.

The oxidation period must be preceded in many cases by a melting period of a length corresponding to the interval between charges, as a longer interval will make the steel solidify in parts. During this time ore is gradually added to the bath in the furnace, and carbon, manganese, phosphorus, and sulphur are oxidized.

The carbon oxidizes only slowly, in spite of the high temperature in the furnace. In the heat illustrated in Fig. 16, 0.10 per cent carbon is oxidized in 55 minutes, which is equivalent to 53 per cent of the original carbon content. Although the manganese is present in a more concentrated state than the carbon, only 58 per cent of the original manganese content is oxidized in the same time, which is equal to a decrease in manganese from 0.52 to 0.22 per cent.

Dephosphorization can be carried out to the furthest extent since the concentration of the oxidizing agent can be governed at will and the slag can be maintained very basic. Phosphorus is the easiest to remove of all the impurities in the charge. With progressing decarburization the phosphorus decreases steadily and the original low temperature of the bath favors the oxidation of the phosphorus. Already the third sample taken from the heat in Fig. 16 after a charge of 50 kg. ore, showed only 0.008 per cent phosphorus, against 0.039 at the start, which is a reduction by 80 per cent. Practically all phosphorus is removed before the slagging proceeds.

Sulphur decreases during this period of refining by oxidation, in Fig. 16 from 0.055 per cent to 0.037 per cent, which is a reduction by 33 per cent. Wüst claims that slag containing ferrous oxide dissolves iron sulphide to a certain extent. In stronger concentration it reacts with the ferrous oxide according to the equation:

$$FeS + 2\ FeO = 3\ Fe + SO_2.$$

The sulphur dioxide escapes and can be clearly recognized. But a part of the sulphur may still remain dissolved in the slag together with free ferrous oxide, as is shown by the sulphur of the oxidation slags of table III (p. 587). The figures in table IV (p. 587) show that the average decrease in sulphur during the oxidation period amounts to 26 per cent.

The oxidation slag with its phosphorus is drawn off and the last traces removed by mixing lime with it. This avoids rephosphorization of the bath by the carbon and silicon additions in the following period. One slag is sufficient to remove all phosphorus down to traces.

Deoxidation Period.

Now the deoxidizing slag is produced by adding to the bath about 20 kg. (44 lb.) lime, 3 kg. (6.6 lb.) sand and a like quantity of fluorspar, and also 1 kg. (2.2 lb.) ferrosilicon (50 per cent silicon) per ton of steel. This slag dissolves the ferrous oxide which has meanwhile been formed on the unprotected surface of the metal, and thereby becomes black.

The destruction of the ferrous oxide in the slag is indispensable for the deoxidation and desulphurization of the bath. For the deoxidation the silicon (from the added ferrosilicon) is not sufficient and 1 to 2 kg (2-4 lb) powdered petroleum coke are added to assure complete deoxidation. In this way a slag is obtained which in the air quickly disintegrates into a white powder; this indicates that the ferrous oxide has been reduced down to traces.

With hard charges an amount of petroleum-coke or waste carbon electrode pieces, in a quantity as required for the desired carbon content, are added to the bath for purposes of carburization, prior to the formation of the deoxidation slag. As soon as the slag is readily fluid and free from oxide, the additions are made of ferromanganese, ferrosilicon, etc.

The addition of these alloys which are often impure may increase the phosphorus and sulphur content considerably. Thus in Fig. 16 sample 6 shows that the phophorus has increased from traces to 0.08 per cent, due to the addition of ferromanganese with 0.4 per cent phosphorus.

In following the curve for sulphur in Fig. 16 it will be seen that the amount increases during the carburizing process, showing that the bath has absorbed sulphur from the carburizing agent; this sulphur is then gradually removed from the bath by

TABLE I.—HEAT E 744.

	Sample No.	C%	Mn%	P%	S%	Si%
Oxidation period. Charge	1	0.09	0.54	0.020	0.042	trace
After adding 10 kg ore	2	0.09	0.44	0.016	0.037	"
After adding 15 kg ore	3	0.09	0.34	0.010	0.037	"
After adding 20 kg ore						
Sample just before slagging	4	0.09	0.28	0.005	0.033	"
Deoxidation period. After adding:						
22 kg coke, 60 kg refining slag	5	0.49	0.20	0.005	0.037	"
40 kg refining slag	6	0.53	0.24	trace	0.037	"
2 kg ferrosilicon, 20 kg refining slag	7	0.48	0.24	"	0.035	"
12 kg ferrosilicon; 18 kg ferromanganese	8	0.54	0.75	0.013	0.022	0.20
1 kg powdered lime	9	0.52	0.70	0.010	0.021	0.22
1 kg powdered petroleum coke	10	0.52	0.70	0.010	0.023	0.21
3 kg ferrosilicon	11	0.57	0.74	0.012	0.018	0.26
Final sample	12	0.58	0.70	0.015	0.014	0.24

Final product, 2985 kg electric steel.
Charge, 3030 kg soft steel.
During the refining period by oxidation sulphur was removed as follows:

3030 kg charge (0.042% S) contain	1273 grams S	
3030 kg (0.033% S) just before slagging contain	1000 "	
Sulphur removed	273 grams S	or 21.4%
After slagging 3000 kg steel (0.033% S) contain	990 grams S	
Absorption of sulphur from 22 kg petroleum coke with 0.60% S	132 "	
Sulphur in metal bath	1122 grams S	
At the end of the desulphurization period (0.014% S)	420 "	
Removed from metal bath	702 grams S	

For the production of a deoxidation slag there were added:

80 kg lime with 0.08% sulphur	64 grams S	
30 kg fluorspar with 0.07% sulphur	21 "	
10 kg sand with traces of sulphur	traces "	
3 kg ferrosilicon with traces of sulphur	traces "	
The deoxidation slag contains	85 grams S	
135 kg final slag with 0.5% S	675 "	
The slag has therefore absorbed from the bath	590 grams S	
There were removed from the bath	702 "	
Missing	112 grams S	or 16%

the refining slag. There is a gradual decrease in the sulphur percentage up to sample 13, but no further decrease from sample 13 to 16. Desulphurization only starts again, after the thick highly basic lime slag has been rendered again effective by the addition of fluorspar (before sample 16).

The chemical reactions between the metal and slag occur only under certain conditions. For example, the reaction:

$$\text{Metallic sulphide} + C + CaO = CaS + \text{Metal} + CO$$

occurs only when the slag becomes white and readily fluid, i. e. after the removal of the metallic oxides down to traces. This is evident from table III (p. 587), which shows that in the slag the increase of sulphur runs parallel to the decrease of ferrous oxide and manganese oxide. It is wrong to assume that when a white slag free from metallic oxide is obtained (as is already the case with sample 12) the bath is completely desulphurized, and an interruption of the smelting process at this time would be a mistake, since the desulphurization has only begun and needs time to be complete.

As long as the bath is covered with an oxide slag it takes up oxide out of the slag and becomes thick and sluggish, which may be the reason for incomplete desulphurization in the presence of an oxide slag. When white slag has been formed on the metal bath, it absorbs the oxide from the bath and the oxide is destroyed by the carbon added and the calcium carbide formed under the electric arc.

The bath becomes more readily liquid through the removal of the ferrous oxide, and now the iron sulphide which is suspended in the bath will rise to the surface of the bath and be absorbed by the slag.

It has been observed that during the hottest period of the process, white vapors rise between the carbon electrode and the cooling ring which vapors deposit on the electrode in form of a mouldy, fibrous crust. As these deposits contain as much as 24 per cent SiO_2, it may be assumed that these vapors consist chiefly of silicon sulphide SiS_2. This gaseous compound is oxidized by the air to SiO_2 and SO_2 and the white deposit of SiO_2 is obtained.

As the final finishing slag has only traces of manganese oxide, and no manganese sulphide in its composition, it is to

TABLE II.—HEAT E 784.

	Sample No.	C%	Mn%	P%	S%	Si%
Oxidation period. Charge	1	0.15	0.54	0.034	0.054	trace
After adding 10 kg ore	2	0.14	0.40	0.021	0.048	"
After adding 15 kg ore	3	0.14	0.34	0.016	0.044	"
After adding 25 kg ore						
Just before slagging	4	0.10	0.29	0.008	0.046	"
Deoxidation period. After adding:						
24 kg petroleum coke; 50 kg refining slag	5	0.50	0.26	0.010	0.034	trace
40 kg refining slag	6	0.50	0.26	0.010	0.038	"
30 kg refining slag	7	0.49	0.27	0.010	0.026	"
9 kg ferromanganese	8	0.50	0.49	0.011	0.026	"
1 kg powdered petroleum-coke; 5 kg ferrosilicon	9	0.52	0.52	0.015	0.026	0.07
5 kg ferrosilicon	10	0.52	0.52	0.012	0.018	0.14
4 kg ferromanganese	11	0.56	0.61	0.015	0.010	0.14
Final sample	12	0.56	0.62	0.015	0.010	0.14

Production, 3150 kg electric steel.
Charge, 3200 kg soft steel.
Sulphur lost during refining by oxidation:

3200 kg charge with 0.054% sulphur contain	1728 grams S	
3200 kg steel with 0.046% S just before slagging contain	1472 "	
Sulphur removed by oxidation	256 grams S	or 15%
After slagging, 3200 — 30 = 3170 kg metal with 0.046% S contain	1458 grams S	
Sulphur absorbed from 24 kg petroleum-coke with 0.75% S	180 "	
Present in metal bath	1638 grams S	
At end of desulphurizing period (0.010% S) there are present	317 "	
Removed out of metal bath	1321 grams S	

To form the deoxidizing slag there were used:

80 kg lime with 0.164% S	131 grams S	
30 kg fluorspar with 0.07% S	21 "	
10 kg sand with traces of S	traces	
3 kg ferrosilicon with traces of S	"	
Deoxidizing slag contains	152 grams S	
170 kg finishing slag with 0.89% sulphur contain	1513 "	
Slag has absorbed from the bath	1361 grams S	
Amount removed from metal bath	1321 "	
Surplus	40 grams	

be concluded from the following calculation that during the oxidation period nearly all of the sulphur goes quantitatively into the slag in the form of calcium sulphide.

In the heat represented by Fig. 16, the quantity of electric steel produced was 3100 kg. Allowing 1.5 per cent. for loss in weight, we have 3150 kg metal charged.

	Sulphur in Grams.	
These 3150 kg steel contain 0.055% S or	1732 grams	
Shortly before slagging 0.037% or	1165 "	
Loss of S in refining by oxidation	567 grams S	
Or 33%		
When slagging, about 30 kg metal are lost and 3120 kg metal with 0.037% S contain	1154 grams	
Carburizing with 16 kg electrode remnants, with 1.2% S gives	192 "	
Present in the metal bath	1346 grams S	
Corresponding to sample 6 with 0.043% S.		
The final sample had 0.012% S, or in 3120 kg we have	374 grams S	
Loss in desulphurizing period	972 "	

To form a deoxidizing slag the following materials were added:

65 kg lime, with 0.068% S	44 grams	
8 kg fluorspar with 0.064% S	51 "	
5 kg sand with 0.21% S	10 "	
2 kg ferrosilicon with traces of S	..	
Deoxidizing slag contains	105 grams	
About 83 kg finishing slag with 1.22% S contains	1013 grams	
Deoxidizing slag has removed from the bath	908 grams	
Lost from the bath	927 "	
Missing	64 grams	or 5%

which have probably disappeared in the form of silicon sulphide.

Therefore in this case 64 grams of sulphur escape probably as silicon sulphide, similarly 112 grams in heat E 744 (table 1) as such, although there is the possibility of some inaccuracy in sampling and analysis.

The sulphur decreased during the deoxidation period

Heat Fig. 16	from 1346 to 374 grams i.e. by 72.2%
Heat E 744 (Table 1)	from 1122 to 420 grams i.e. by 62.6%
Heat E 784 (Table 2)	from 1638 to 317 grams i.e. by 80.6%

TABLE III.—COMPOSITION OF SLAGS.

Chemical constituents.	Oxidiz. slags.	Deoxidizing slag: After carburization.	Deoxidizing slag: After main charge.	Finishing slag.
CaO	41.69	59.50	69.45	75.85
SiO_2	9.58	26.78	17.90	13.20
FeO	28.36	1.02	0.27	0.13
MnO	0.99	1.44	0.52	trace
S	0.62	0.39	0.90	1.22
Fe_2O_3	6.43	0.23	....	....
Al_2O_3	1.73	1.67	1.27	1.73
MgO	6.99	5.54	5.78	4.22
P_2O_5	1.47	0.07	0.12	0.09
Oxide of bases	20	20	22	22
Oxide of acids	9	15	10	8
Notes—	Black and streaky.	Grayish brown sand	White sand.	White powder.

TABLE IV.—DESULPHURIZATION EFFECTED DURING OXIDATION PERIOD.

	Ore added.	C%	Mn%	P%	S%	Si%	Decrease of S.
Charge E 152	.....	0.19	0.76	0.02	0.045	trace	
Sample 1	15 kg	0.13	0.48	0.01	0.044	"	26.7%
" just before slagging	25	0.09	0.30	trace	0.033	"	
Charge E 169	.....	0.20	0.64	0.03	0.056	traces	
Sample 1	30	0.20	0.52	0.02	0.056	"	30.4%
" before slagging	35	0.12	0.28	trace	0.039	"	
Charge E 455	.....	0.23	0.62	0.030	0.059	traces	
Sample 1	5	0.22	0.44	0.010	0.048	"	23.7%
Sample 2	10	0.21	0.42	0.010	0.045	"	
" before slagging	15	0.21	0.38	0.005	0.045	"	
Charge E 758	.....	0.17	0.56	0.025	0.052	traces	
Sample before slagging	25	0.09	0.22	traces	0.044	"	15.4%

TABLE V.—ANALYSES, TESTS AND APPLICATIONS OF VARIOUS ELECTRIC STEELS.

W.	Cr.	Ni.	C.	Mn.	P.	S.	Si.	Tensile Strength: Kg.-Sq. Mm.	Tensile Strength: Elongation on 200 Mm. Bar.	Tensile Strength: Reduction of Cross Section.	Applications.
....	....	....	0.13	0.54	Trace	0.014	0.16	41.2	30 %	65.6%	Angle 130x65x8 mm.
....	....	....	0.13	0.70	0.018	0.020	0.08	43	31 %	62.3%	Angle 130x65x8 mm.
....	....	....	0.19	0.56	Trace	0.010	0.13	47	31 %	55 %	Angle 130x130x12 mm.
....	....	....	0.22	0.90	0.010	0.017	0.25	58	27.5%	52.8%	Iron 150x12 mm.
....	....	....	0.25	0.88	Trace	0.017	0.26	60	23.5%	49.1%	I-beam N. P. 30.
....	....	....	0.33	0.74	0.010	0.010	0.20	65	19 %	42 %	Engine and machine parts.
....	....	....	0.64	0.62	0.010	Trace	0.47	86	13.2%	23.5%	50 mm. square blocks.
....	....	....	0.53	0.68	0.008	0.010	0.37	86.9	12.8%	30.3%	Flat steel bars.
....	....	....	0.57	0.70	Trace	0.008	0.40	89.4	10.7%	21.4%	Flat steel bars.
....	....	....	0.72	0.42	0.020	0.010	0.34	97.1	6 %	23.8%	Dies.
....	....	....	0.82	0.48	0.006	0.010	0.25	99.5	5.8%	20.1%	Punches.
....	....	....	0.70	0.72	0.010	0.010	0.36	83.8	11.5%	19.8%	Tramway car-wheel tires.
....	....	....	0.47	0.78	0.010	0.012	0.24	71.5	17.5%	32.9%	Railway car-wheel tires.
....	....	....	0.36	0.78	0.013	0.010	0.17	64	19 %	40.7%	Axles.
....	....	....	0.48	0.78	0.007	0.019	0.28	74.4	13.3%	24.9%	Locomotive wheel tires.
....	....	....	0.52	0.78	0.015	0.018	0.19	78.7	11.1%	20.5%	Locomotive wheel tires.
....	....	....	0.48	0.84	0.007	0.018	0.23	72	17.7%	44 %	Shafting.
....	....	....	....	....	....	....	....	72	16.5%	45.2%	Shafting.
....	....	1.89	0.26	0.78	Trace	0.012	0.12	61.8	23.5%	48.6%	Spindles.
....	....	1.98	0.20	0.72	0.012	0.018	0.13	54	22 %	58.6%	Spindles.
....	1.04	....	1.02	0.58	Trace	0.012	0.16	....	....	....	Knives, shears, chisels.
1.50	0.92	....	0.49	0.82	0.012	Trace	0.32	79.7	15 %	40.4%	Tools.
....	0.85	....	0.46	0.60	Trace	0.012	0.22	79.6	10.2%	32.9%	Pieces for forging.
....	....	....	....	....	....	....	....	62.1	33 %	59.5%	Quenched at 700° C. in oil
....	....	....	....	....	....	....	....	59.6	36 %	63.8%	Quenched at 700° C. in water
....	....	....	0.68	0.32	0.010	0.010	0.23	93.3	9 %	18.8%	Rolls for making grooved rail.

This includes the quantities of sulphur absorbed from the carburizing agents, etc., and again removed.

To sum up, the removal of the sulphur takes place in three different ways:

(1) During the oxidation period sulphur is removed through the action of the slag rich in ferrous oxide, whereby sulphur dioxide is formed which escapes.

(2) In the deoxidation period the white slag, free from ferrous oxide, removes sulphur.

(3) The third agent in desulphurization is silicon, forming silicon sulphide.

It is now generally believed that oxides, gases, and slag enclosures have a much more unfavorable influence on the physical properties of steel than phosphorus and sulphur below 0.02 per cent. The electric steel furnace allows the steel bath to rest a long time under the neutral non-oxide slag, which procedure can be carried out in the crucible only with difficulty without chemical changes, and which cannot be carried out at all in the open hearth. The possibility of removing the oxides, gases, and slag enclosure in this way is, therefore, an important advantage of the electric furnace.

As regards deoxidation, it might be said that it is started by introducing carbonaceous materials. The extent and the speed of deoxidation depend on the temperature, the amount and the action in time of the carbon additions. Complete deoxidation of steel with the aid of carbon alone (even with a slag completely free from ferrous oxide) is possible in the electric furnace only, if the refining period is extended for a very considerable time. It is, therefore, advisable to remove the last traces of oxygen by the use of other deoxidation agents like manganese and silicon in order to bring the oxygen into the slag. The oxygen compounds in the slag are reduced by the aid of ferrosilicon or powdered charcoal.

The calcium carbide formed at the greatest heat center under the arc can be increased so much in amount by raising the tension of the electric arc that during the cooling of the slag ladle the acetylene is ignited with explosive violence.

Coussergues (*Revue de Metallurgie*, 1909, p. 589) claims that the calcium carbide causes the formation of acetylene at the contact surface of metal and slag according to the reaction: $CaC_2 + 2H = Ca + C_2H_2$, thus removing hydrogen from the steel and thus rendering electric steel more free from hydrogen than even crucible steel.

The products of the Girod furnace in the Gutehoffnungshütte are carbon steels of various degrees of hardness, also alloy steels, like nickel-steel, chromium steel, nickel-vanadium steel; chromium-vanadium steel, chromium-tungsten steel for high-class machine and engine parts and tools.

Table 5 gives a short synopsis of the results of tests and analyses, and the application of electric steels made in the Girod furnace.

Oberhausen, Rhineland,
Germany.

Cobalt.—The principal uses of cobalt in the United States are in making glass and pottery. A beautiful blue is given to glass by the oxide of cobalt. Sympathetic inks, according to a report of the United States Geological Survey, are made from cobalt acetate, chloride, and nitrate that are colored when heated or colorless when cold. This interesting phenomenon is due to the change in color of the salts on the absorption of water. When dry they are blue and easily seen on paper; when damp they are pink; and when dilute, colorless. A puzzling application of this principle may be in a doll whose dress is blue in dry weather but changes to pink when subjected to dampness, as in wet weather or when the doll is held in the steam of a tea kettle. Artificial flowers are made to show the same effect.

Progress of British Columbia Zinc Investigation.

The status of the investigation into the zinc resources of British Columbia, now being conducted by the Department of Mines, has been outlined in a recent report to the Director of the department by Mr. W. R. Ingalls, consulting engineer to the commission. The gist of the report follows:

Work has been prosecuted in the field of electric smelting at the metallurgical laboratory of McGill University, Montreal, under the immediate direction of Dr. Alfred Stansfield. A large number of experiments have been made with several forms of furnaces, certain of which have rather elaborate construction, and with a variety of material. The early experiments were directed chiefly toward a discovery of the metallurgical conditions which have hitherto prevented a satisfactory condensation of zinc as molten spelter. While I cannot say that these experiments have afforded us a complete explanation of these conditions, they have taught us a good deal; but in spite of the knowledge acquired we have been so far unable to master the difficulties.

We have indeed produced small quantities of spelter, and in some experiments have condensed a fairly large proportion as molten metal, but we have not yet been able to do that at will. Our experiments have thrown light on the principles of furnace design and have led us to condemn several types that we have tried. Our work has indicated that in order to achieve any material improvement over the ordinary practice of zinc smelting it is necessary to abandon certain features of the latter and contemplate continuous charging of the ore and reduction material, and discharging of the residuum without interfering with the process of distillation. These conditions introduce a multitude of perplexing difficulties which can be worked out only by tedious experimentation.

At the request of the secretary of the Canadian Mining Institute I presented at the meeting of the Institute at Quebec in March, 1911, a paper on the problem of mixed sulphide ores, that concisely summarizes the state of the art in the treatment of such ores, and the natural obstacles that block procedure in certain directions. This paper was published in the *Institute Quarterly Bulletin* No. 15, pp. 23-30, June, 1911.

A careful scrutiny of the work on the treatment of such ores that is done by other metallurgists and investigators has been maintained, and I have examined numerous proposals that have been presented with more or less detail, but I have not discovered anything save one that in my opinion holds out any promise of successful adaptation to the conditions existing in British Columbia. I regard the electric work as being of particular interest as an exploration in a virgin field of unknown possibilities. Doubtless with the same idea a great deal of work is being done in this field by numerous investigators in Europe and America. I have been informed within a few weeks that there are now two electrothermic zinc smelters in operation in Scandinavia, viz., one at Trollhättan, Sweden, using about 7000 hp, and one at Sarpsborg, Norway, using about 4000 hp. Operations at these works were inaugurated five or six years ago, but according to my information the results were for several years commercially unsatisfactory, and it is only recently that it has been claimed to have become possible to make spelter from ore upon an industrial scale. The companies operating these works maintain absolute secrecy, and I have not been able to learn any details of their operation.

Apart from the work in Scandinavia, as to the commercial success of which there is no information available, the electric smelting of zinc ore is, in spite of all claims to the contrary, not only still in the experimental stage, but is in the infancy of the experimental stage. Even if the metallurgical difficulties can be overcome, which is possible, I am of the opinion that no one, except perhaps the Scandinavians, is yet in a position to make any reliable estimate of commercial advantage or commercial results in any way. It is, however, worth while to determine the possibilities and publish the results for the general benefit, unveiling so far as possible the secrecy that is likely to be maintained as to the investigations in this field by private interests, having always in mind, of course, the hope that our work may develop a process that will be commercially applicable to the treatment of the zinc ores of Canada.

Cyanide Tailings Disposal in Mexico.

The discharge of cyanide tailings into streams, the waters of which are later used for domestic, industrial or other public purposes, brings up the possibility of legal complications between metallurgical companies and industrial or agricultural communities.

With a view to gathering information on the metallurgical side of the subject, the Instituto Mexicano de Minas y Metalurgia recently propounded a number of questions to leading metallurgical companies. The questions concerned:

(A) the percentage of cyanide contained in material discharged into streams, and the distance down stream at which these percentages persisted;

(B) the chemical or mechanical possibility of eliminating or reducing the quantity of cyanide in discharged tailings;

(C) the cost of conserving or impounding tailings in stacks or dams;

(D) the possible use of tailings for mine filling;

(E) the advisability and cost of purchasing adjoining lands which might be affected by the discharged tailings and cyanide.

From the replies received by the Institute and published in its *Informes y Memorias*, the following are given as representative opinions.

Mr. Hugh Rose, of the Compania Beneficiadora de Pachuca, writes as follows: "(A) and (B)—I have no information of value on these subjects. (C) Where the location of a mill is such that a convenient dumping ground is available at reasonable expense, I consider it by all means advisable to retain the tailings in dumps or dams, according to the condition in which the tailings are discharged from the plant. Where the tailings can be discharged dry, they may be stacked either by belt conveyors or by cars. Where the tailings are discharged with considerable moisture, the South African practice of retaining the tailings by dams built automatically by the discharged tailings is an easy and cheap method. This method may be also used to recover the water in the pond so formed.

"The cost of conserving the tailings by either of the above methods naturally varies to some extent with the special conditions of each plant, but in any case should not be over 2 or 3 centavos per ton, exclusive of the cost of the land." (1 peso or Mexican silver dollar = 100 centavos).

"In a great many instances the conservation of the tailings is a wise policy, not only as a means of avoiding trouble with land holders, but in preserving perhaps for future generations a valuable asset. This applies particularly to the silver mills of Pachuca. There can hardly be any doubt about further advances in milling metallurgy making later economically available the values now discharged in the tailings.

"(D) The difficulties attached to using the tailings for filling stopes underground are increased where the tailings are very fine, as is the case with all sliming plants. It is not likely that this method can be adopted in general, as it requires such special conditions covering the location of mine and mill, method of mining, etc. It is specially applicable where the topograhy is flat or slightly rolling, as is more the case of the Rand, and in some of the flat mineral deposits.

"(E) The purchase of polluted lands would appear to be a very unwise policy, applicable to advantage only in special cases. Not only would it be difficult to secure the land at fair prices, but the adoption of such a policy generally would naturally result in complications to mills so located where the purchase of such lands would not be desirable or necessary.

"In conclusion, I would say that it is the intention of this company to impound its tailings by dams, as discussed in para-

graph C, for the purpose of not only avoiding controversies with land owners, but also of preserving the tailings for later treatment by this company or its successors."

* * *

Mr. D. L. H. FORBES, metallurgical engineer of the Tigre Mining Company, gives the following opinion:

"The cyanide plant of the Tigre Mining Company, which is now about to start operation, will discharge into the Tigre Cañon 250 short tons per day of slime residues containing 40 per cent moisture. These residues will carry the cyanide equivalent of about 50 lb. of potassium cyanide which will be wasted by the displacement of 42 tons of weak barren solution per day.

"The presence of this amount of cyanide with the residues will give the water in the Tigre Cañon a maximum solution strength of 0.025 per cent KCN during the dry season.

"While it is believed that such a weak cyanide solution would have its poisonous properties completely destroyed before flowing a greater distance than two miles, yet the Tigre Mining Company, in order to avoid the possibility of troublesome damage suits, has acquired a strip of land which takes in both sides of the cañon from the mill to the Bavispe River, a distance of ten miles. The writer is informed by Mr. L. R. Budrow, manager of the Tigre Mining Company, that the total cost of protecting the company's interests in this way was approximately 60,000 pesos.

"It is not the present intention of this company to collect the residues from the cyanide treatment in a dry condition, but the plant has been so designed that this may be easily accomplished whenever it shall be thought advisable. I would estimate that the extra operating cost for stacking slime residues which had been previously air-dried on the filter leaves would be 6 to 8 centavos per dry short ton. In the case of this plant it could be accomplished best by means of a 12-in. carrying belt. The initial cost of such a conveyor installation would be about 4000 pesos."

On the subject of filling the mine with tailings the writer is of the opinion that, aside from the inconvenience and expense of handling the tailings in this manner, such a method of disposal would be inadvisable at this property; both because of the method of mining used here, and on account of the timbering expense and danger that would be involved. As the veins of this property all have a nearly vertical pitch and are comparatively narrow, it has been found most economical to mine the ore bodies by a system of overhand stoping.

"Each block of ore is mined in such a manner that the stopes are kept nearly filled with broken ore until the level above the stope is reached. The ore is withdrawn in quantity only sufficient to compensate for its extra volume when broken, and the stopes are never entirely emptied until the section in which they are located is worked out and ready to be abandoned.

"Even if the filling of the mine with tailings were feasible, the fact that practically all precious metal ores are extracted from the upper workings of the mine first makes it obvious that the storage of the large volumes of mud in a pasty condition would have to be above the main roads and passageways and would therefore constitute an element of danger and an extra expense that would seriously handicap any mining enterprise that might adopt it.

"The practice of rendering escaping cyanide mill solution innocuous by special treatment has not been tried at any of the plants with which the writer has been connected. If such a procedure were made obligatory by statute, undoubtedly several methods would at once suggest themselves, the choice of which would depend largely upon local conditions and the spirit of the law.

"The judicious use of any one of the stronger mineral acids, while it would destroy the most active poison present in the solution, would also be adding another impurity to water that would still be unfit for drinking purposes. If, however, the application of the law should require the complete purification of the water escaping from the mill, a special procedure consisting of precipitation of all salts in solution and subsequent filtration would have to be developed. Such an operation as the latter would involve expense that many companies which are now operating on low-grade ore and tailing dumps with small margin of profit could not bear.

"In most instances, however, it will probably be found that sufficient purification takes place in a small running stream. Wood, leaves, fibrous roots of plants and the carbonic acid present in the atmosphere are all well-known agents that decompose cyanide, and as these are always present in the smaller streams of this country it is evident that waste mill solutions mingling with the water of such streams must soon be rendered harmless in so far as their cyanide properties are concerned."

It should be remembered that the practice, which is becoming prevalent in the gold and silver mills of Mexico, of crushing ore in cyanide solution enables the operator to obtain a closed solution circuit, whereby practically no cyanide is run to waste unless "fouling" occurs. Wherever the ore is of such a character that the extraction of values is satisfactory and the decomposition of cyanide while in contact with the raw ore not excessive, crushing in solution is to be recommended, and it has the advantage of lessening the possibility of legal complications from actual or supposed harmful effects of residues escaping from the mill.

* * *

The following compilation of information from the company's several mills was contributed by Mr. GEORGE BRYANT, resident director of the Guanajuato Development Company:

(A).—*Peregrina Mill,* capacity 500 metric tons per day of tailings about one half sand and one half slime are discharged into a large reservoir which is opened once in two years and then during the occurrence of the heaviest rains, so that both the water impregnated with cyanide and the tailings themselves are taken away by flood water and washed a distance of from 20 to 30 km from the mill almost immediately. The superintendent's answer to question (A) is, that "the waters on leaving the mill contain about 0.03 per cent cyanide, and are practically unchanged on reaching the reservoir.

San Prospero Mill (Mexican Milling & Transportation Company), capacity 150 metric tons per day, discharges slimes immediately into the stream, where they join the drainage waters from the city. The sands, comprising one-half of the tailings, are stacked until the rainy season, when they are also carried away by flood waters. The superintendent states as follows, answering question (A): "The water leaving this mill contains on the average 150 grams cyanide to the ton. This proportion is greatly reduced about 400 m below the point of the discharge."

Pingüico Mill, capacity about 300 tons per day tailings about one-half sand and one-half slime. The slimes are discharged directly into a stream below the mill in which there is sufficient water for about one-half of the year to push them along, while during the dry half of the year they collect until the floods of the rainy season push them further down. The sands collect until the rainy season, when portions of them are washed away by the floods. The superintendent states, in answer to question (A): "The solution discharged from this plant with the tailings carries 0.02 per cent KCN. All traces of solution disappear 50 m below the plant."

Jesus Maria Mill, (Guanajuato Amalgamated Gold Mines Company), capacity 450 tons per day of tailings about one-half sand and one-half slime. The sands are stacked and some of them are carried away during the rainy season. The streams flow as long as there is water in the same to push them, which is about nine months in the year. The superintendent answers question (A) as follows: "The water leaving this plant flows down a cañon having very precipitous sides, and for some 4 km (1 km—0.62 mile) the total land contiguous to the stream and used for agricultural purposes does not total one hectare. From that point on there are orchards and more or less cultivated land.

Upon leaving the company's property the solution titrate is, in terms of KCN and CaO

	KCN	CaO
	0.03%	0.31%
1 km down stream	0.03%	0.26%
2 " " "	0.02%	0.19%
3 " " "	0.015%	0.014%

These samples were taken at five-minute intervals over a period of two hours, which represents a complete cycle on our filter, and gives the average solution discharged. It includes the cyanide of sodium, potassium and the simple cyanide of zinc ($ZnCy_2$), usually estimated as free cyanides.

The total cyanides include KCy.KSCy, K_4FeCy_6, K_2ZnCy_4, and the related sodium cyanide compounds are as follows:

Total cyanide in terms of KCN:

At plant	0.05%
2 km down stream	0.04%
4 km down stream	0.025%

(B).—Three of our superintendents answer that they do not know of any practical chemical or mechanical process for further reducing the small proportion of cyanide contained in the discharged water and tailings. A fourth superintendent answers as follows:

"It is doubtful whether sodium cyanide is especially detrimental to water used for agricultural purposes, as the sodium cyanide undergoes slow decomposition, giving off ammonia and urea (both of which are used in the manufacture of artificial fertilizers), and formic acid. Rather the excess of lime is the damaging material, and this could be neutralized by commercial acid, or possibly by our mine waters, should the pumps work continuously.

"Some years ago one plant let its waste solutions percolate through refuse matter from slaughter-houses, manure and sawdust to render them harmless, but with indifferent success. Then, for some years following the introduction of cyaniding milling in many camps, efforts were made to render the effluent solutions harmless, but I am not aware of marked success attending a single effort, and cannot recommend a process to effectively accomplish that result."

(C).—Peregrina answers that the case of discharging dry tailings would increase the milling cost by 1 peso per ton. Pingüico answers that 60 per cent of the tails discharged are perfectly dry and are washed away each year by the rains. Another answers, "About 45 per cent of our tailings are now discharged dry and do not enter the river. It would be impracticable to discharge the balance dry." I quote verbally from a fourth superintendent's answer: "The slime discharge averages 42.5 per cent moisture at present, our sand is dry for practical purposes. Should the tailings be discharged with a lower moisture per cent than our ore carries, the solution would accumulate by reason of this excess moisture until such a time as all storage capacity would be occupied, and then the excess day by day would run to waste. On the other hand, if not drier than the ore, there would be a continuous seepage of solution rather high in KCN from around the lower edge of the dump, and this would gradually find its way into the stream. Thus, while the objectionable elements would be lessened, it would not cause their elimination."

(D).—At all of our plants, mines and mills are so far separated that use of tailings for filling stopes would be certainly out of the question.

(E).—So far as we know, no damage is done by our tailings to agricultural lands, and up to the present time no claims have been suggested for such damage.

In only one case have I taken this matter into consideration. In that case we found that a total outlay of 46,000 pesos would be necessary to accomplish the purpose suggested in question (F), and figuring the ore supply in that property as 720,000 tons, sufficient for five years' work, we calculated that it would increase the milling cost by 6.4 cents, Mexican currency, per ton.

Possible Reduction of the Power Consumption in Electric Steel Refining Furnaces.

By Carl Hering.

The question of the power consumption in electric furnaces for steel refining is, of course, of great importance and is sometimes even the criterion, hence it deserves the most careful consideration. The present article is an attempt to determine from present practice approximately what the maximum economy is which may be expected in a furnace designed with that end in view. Such a figure would show what the limit of economy is beyond which we cannot expect to go and how near present practice is to this best economy. As the deductions are based on data from an actual furnace they, of course, are subject to the same inaccuracies as the original data. But even if these data are not accurate, and the writer assumes no responsibility for them, they are not likely to be far wrong, and the description of the method of making the deductions may itself be of interest, as it can be applied in the future when better data become available. Even if the present data should have been 100 per cent greater (worse) the results deduced would still be surprisingly favorable.

The chief function of a furnace for steel refining is merely to keep the metal at a constant temperature during the process of refining. In the large Chicago furnace the metal is said to even leave at a slightly lower temperature than that at which it entered. Hence the only energy to be considered here is that necessary to keep the temperature constant; if, therefore, there are any chemical reactions which consume energy, or if there is any melting done, or any increase in temperature, the energy necessary for this must be added. Moreover, the heat lost through the doors, electrodes, etc., is also neglected, hence must be added in an actual case.

When the temperature is maintained constant it means that the total energy supplied to the charge is equal to that lost by it. The heat loss from the charge must, of course, all flow through the bounding surfaces of the charge—that is, through the top, sides and bottom; it is lost by radiation from the top and by conduction from the sides and bottom. For want of better data it is here assumed that the loss by radiation from the top is at the same rate per unit surface as that by conduction from the sides and bottom. Though this is no doubt not strictly correct, it is probably sufficiently so for the present purpose of reaching an approximate figure; this figure can be corrected later when better data become available.

It is also assumed in the present deductions that the insulation of the walls in actual practice is uniform for all different sizes of furnaces—that is, that the heat energy passing out of the metal per square foot of its outside surfaces is a constant and is equal to that in the particular furnace from which the data were obtained. If this insulation can be improved the final figures will be lowered accordingly.

Under these circumstances the heat losses through these bounding surfaces for a constant inside temperature are, of course, dependent only on the extent of this surface and not on the quantity of the metal within the walls.

If, therefore, the loss is a constant per square foot of surface of the metal, and is independent of the quantity of metal, it follows that the loss will be least when the metal is in the shape of a sphere. But as this is impracticable it will be assumed here that the ideal form in practice is a hemisphere, leaving a dome of an equal hemisphere above for working space.

The problem therefore is, on the basis of a constant loss per square foot of metal surface,, what will be the total loss when the body of metal has a hemispherical shape?

This constant loss may be deduced from a recent paper by Mr. Albert Hiorth, read at the recent Toronto meeting of the American Electrochemical Society[1], in which he states that by actual test it required 180 kw to keep the steel melted over

[1]See preceding issue of this journal, p. 534.

night in one of his five-ton, two-ring, single-phase induction furnaces for steel refining. As there are no electrodes, this represents the power which is being continuously lost through the walls. When the furnace was running it took somewhat more, but presumably cold slag was then being melted, and probably the covers were frequently removed; there was also presumably an increase in temperature; these doubtless account for the increased input of power. For the present purposes, therefore, it is probably fairest to use the figure for the all-night run. As these measured results were used by him to design a 30-ton furnace, it may be assumed that they are fairly correct.

He also gives all the dimensions of the crucible and body of metal; hence, one is enabled to deduce from them the outside surface of the charge of metal and the total outside surface of the crucible. These deductions have been made by the present writer and the result is that the average continuous loss of heat from the bo nding surfaces of the metal (sides, bottom and top) is equal to about 13 kw per square meter, or

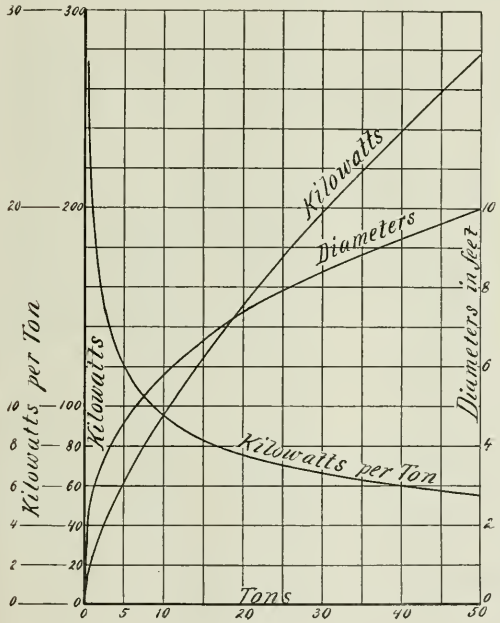


FIG. 1.—MINIMUM KILOWATTS PER TON AS FUNCTION OF SIZE OF FURNACE (CONDITION OF LEAST LOSS).

1.2 kw per square foot. The total surface of the charge was 13.9 sq. m., and the total loss 180 kw. The total exposed outside surface of the crucible was 56 sq. m., or about four times the metal surface.

This average figure, 1.2 kw per square foot of metal surface, is taken as the basis of the following deductions. It is doubtless greater in some parts and less in others, but, as the results are necessarily only approximate, a further refinement is hardly warranted.

In a Roechling-Rodenhauser single-phase, two-ring, induction furnace, the data for which was kindly furnished to the writer for this purpose, the surface of the metal charge was found to be 6.36 sq. meters, and the power necessary to maintain the temperature was given as about 80 kw.; it was a 2-ton furnace, hence considerably smaller than the other one. This reduces to a loss of 12.5 kw. per square meter of metal surface, or 1.17 kw. per square foot. The agreement is herefore confirmatory and is in fact strikingly close.

As these figures can no doubt be improved, an easily remembered rough-and-ready figure for the losses through walls under steel refining conditions is therefore one kilowatt per square foot.

A ton (metric ton equal approximately to a gross ton) of molten iron occupies a volume of about 5.13 cu. ft. From this and the above data the following table has been calculated. It gives the kilowatts which would be required on the above bases to keep steel at the refining temperature, the bottom of the hearth being hemispherical and the metal just filling the hemispherical part. Hence it gives the least number of kilowatts, or kilowatts per ton, required under the best practicable conditions for obtaining the least loss. The curves give the same results graphically.

Capacity.		Diameters		
Gross Tons.	Cubic Feet.	in Feet.	Kilowatts.	Kw. per Ton.
½	2.57	2.14	13.	26.
1	5.13	2.70	20.	20.
2	10.3	3.40	32.	16.
5	25.7	4.61	60.	12.
10	51.3	5.80	95.	9.5
15	77.0	6.65	124.	8.
30	154.	8.38	200.	6.6
50	257.	9.93	278.	5.5

The figures in the last two columns will be found to be far smaller than those of present practice, showing that there is a possibility of very great improvement in reducing the consumption of energy. The actual figure for the Hiorth 5-ton induction furnace was 36 kw per ton as agains a possible 12 kw. In the South Chicago 15-ton furnace it is given as 50 against a possible 8.

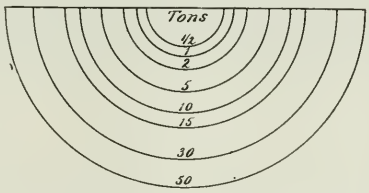


FIG. 2.—INCREASE OF SIZE OF FURNACE WITH INCREASE OF CAPACITY.

The figures in the table are based on the actual results in an induction furnace, but, as the heat insulation per square foot of metal surface of a simple hearth could no doubt be made still higher, the numbers in the last two columns can probably be still further reduced.

These extremely low power consumptions in the table point out the direction in which progress becomes possible if the power consumption is important. They can, of course, be expected to be reached only in those types of furnaces in which it becomes possible to give the hearth its best possible shape and in which the necessary slagging surface is obtained by energetic circulations, as, for instance, in the type of furnace devised by the writer and described in a recent paper read before the American Electrochemical Society[2] and abstracted in the May issue of this journal, p. 277. In other types the results may be approached more or less closely by so designing the hearth or crucible that the bounding surface of the metal is the least possible in that particular type.

The above deductions have nothing to do with the length of time of the refining. If by any improvements, such as rapid circulation, which brings all parts of the metal into repeated contact with the slag, the time of treatment could be reduced, it would, of course, reduce the energy required still more, and in direct proportion; this will reduce the kw-hours, as distinguished from the kilowatts, and it is with the latter only that the above deductions are concerned.

[2]Trans. Amer. Electrochem. Soc. Vol. XIX, 1911, p. 255.

At half a cent per kw-hour it would therefore cost only about 5 cents per hour and per ton of metal to keep it melted for refining purposes in a 10-ton furnace, or 3.3 cents in a 30-ton furnace. As stated above, any power used for melting cold slags, raising the temperature, supplying the losses through open doors, electrodes, etc., must be added hereto. These surprisingly low figures are a good illustration of the improvements which may be expected by a more careful and more rational design, with a view to obtaining the best heat economy.

The following further deductions indicate that even these low figures are probably too high, and that it may be possible to reduce them considerably, possibly to about half these values. In an induction furnace the walls of the crucible which surround the iron cores, are necessarily restricted in thickness; in the one from which these data were taken these surfaces are furthermore water cooled in order to protect the core; moreover, there are 1.65 tons of copper in the primary turns, which are intentionally placed as near as practical above and below the hot metal secondary, to increase the power factor; such a mass of copper will carry off a large quantity of heat. It is likely therefore that the average loss of 1.2 kw per sq. ft. of metal surface in this particular furnace is higher than it would be in one in which the wall thickness is not restricted and in which there is no such mass of copper near the charge. The outside walls of this crucible were about 23 in. thick (60 cm.), the bottom apparently about the same, while the inside ones were only about 14 in. (35 cm.); about one-quarter of the total metal surface was exposed to these thinner walls. The metal charge had about three times the surface exposed to the losses that it would have in a hemispherical hearth.

Taking a five-ton furnace with a hemispherical charge as an illustration, and assuming the wall to be 24 in. thick, the surface on the outside corresponding to 1 sq. ft. on the inside is about 3.5 sq. ft.; hence the flare is about the same as the average (4) in the Hiorth furnace. The mean cross-section of the path of the flow of heat from 1 sq. ft. of the metal surface will then be 1.87 sq. ft., namely, the geometric mean. The thermal resistivity of fire brick deduced from Wologdine's results* is 22.4 thermal ohms for one inch cube. Hence the thermal resistance of such a frustrum of fire brick is $24 \times 22.4/1.87 \times 144 = 2$ thermal ohms; hence if the difference of temperature between the inside and outside is about 1400°C., the flow of heat $W = T/R$, would be $1400/2 = 700$ watts, as compared with 1200 watts deduced from the Hiorth furnace, or only a little over half as much; hence the figures in the table are likely to be nearly double what they should be. Moreover, the Wologdine values for the thermal conductivities are believed to be too high (100 per cent for fire brick, according to Clement & Ely), and if this is true, the flow will be considerably less than 700, perhaps only half as great, in which case the figures in the table would be reduced in the ratio of 1200 to 350, or to less than a third. It is likely, however, that magnesite was used as the lining, and this would increase the flow somewhat, as it conducts far better than fire brick. The rough agreement of the results deduced by these two entirely independent methods shows that the figures are not likely to be far wrong, and are no doubt on the safe side.

An additional check of the order of magnitude of these figures may be obtained from steam heating practice. The amount of heat emitted per square foot of surface of an iron heater, due to radiation and moving air, reduces to about 200 watts when its temperature is about 150°C., and it is said to be about 30 per cent greater from a sandy surface, hence 260 watts. The rate at which the heat would be emitted from the outside surface of the five-ton hemispherical furnace mentioned above is $700/3.5 = 200$ watts per sq. ft., using Wologdine's value, and $1200/4 = 300$, using Hiorth's results, thus again showing very good agreement considering the crudeness of the figures.

*This journal, January, 1911, p. 13.

It, therefore, seems probable that the surprisingly low figures in the table are not only safe, but will in practice turn out to be still lower, and perhaps considerably so, showing the great advantages that may be gained by first finding out what is the best possible design and then approaching it as closely as conditions permit. In an arc furnace the conditions are such that these very high economies can never be expected, owing to the fact that much of the heat of the arc never reaches the metal itself; the above figures refer to the losses of heat from the charge itself, and therefore apply more particularly to resistance furnaces.

Gold, Silver, Copper, Lead and Zinc Production in Western States in 1910.

The United States Geological Survey has recently issued reports showing official estimates of the production of the principal metals in Western States. For convenience the totals have been tabulated as below.

Arizona showed an increased output of gold, but decreased production of silver, copper, lead and zinc. In Colorado the output of lead and zinc increased, but gold, silver and copper showed a decrease. Idaho had an increase in production of silver, lead and zinc, but showed a decrease in gold and copper. In Montana there was an increase in lead and zinc, but a decrease in gold, silver and copper. Nevada had an increase in output of gold, silver and copper, but a decrease in lead and zinc. In Utah there was an increased output of copper and zinc, but a decrease in gold, silver and lead.

	Arizona.	Colorado.	Idaho.
Gold, value	$3,149,366	$20,507,058	$1,096,842
Silver, fine oz	2,566,528	8,509,598	7,369,742
Copper, lb	297,491,151	8,359,535	7,037,292
Lead, lb	2,696,151	76,071,273	228,258,839
Zinc, lb	5,484,899	77,089,648	5,603,114
Total value	$42,731,519	$33,673,879	$16,316,196
Tons mined	3,914,969	2,434,664	1,786,174

	Montana.	Nevada.	Utah.
Gold, value	$3,730,486	$18,878,864	$4,032,085
Silver, fine oz	12,162,857	12,479,871	10,466,971
Copper, lb	284,808,553	64,359,398	127,597,072
Lead, lb	4,106,292	4,871,130	123,597,072
Zinc, lb	31,638,184	2,707,071	16,367,104
Total value	$48,358,253	$34,152,148	$32,199,185
Tons mined	5,079,446	3,655,775	6,380,398

Sixth Congress of the International Association for Testing Materials.

The sixth congress of the International Association for Testing Materials will be held under the patronage of the President of the United States at the Engineering Societies' Building, New York City, during the week beginning September 2, 1912.

The nature of the work for which the congress will meet is concisely described in section 2 of the by-laws of the association as "the development and unification of standard methods of testing; the examination of the technically important properties of materials of construction and other materials of practical value, and the perfecting of apparatus used for this purpose."

Another of the most important functions of the association is the establishment of standard specifications for materials used in manufacture and construction. As these specifications are of greatest value to engineers, it is to be hoped that many American engineers will join the association in order to attend the congress and aid in establishing these specifications.

Prof. Henry M. Howe is the president of the American

Society for Testing Materials and acting president of the International Association for Testing Materials. Prof. Edgar Marburg, of the University of Pennsylvania, Philadelphia, Pa., is the secretary of the American society.

Mr. H. F. J. Porter, 1 Madison Avenue, New York City, is the secretary of the organizing committee of the congress, and all communications concerning the congress should be addressed to him.

An Historical Note on Ore Treatment in San Juan County, Colorado

By Warren C. Prosser.

It is often stated that milling practice in San Juan County is far behind the times, which is probably true. Pressure has, however, been brought to bear within the past few years, by adverse circumstances, which has created the incentive to do better work, improve conditions and institute the best methods of ore treatment. This will no doubt result in successful modern practice at no distant future time.

In picking out the flaws of past and present methods the main fact discovered is that instead of having one problem to solve in arriving at successful extraction of the metallic values, such as might confront a camp mining one general character of ore, there are here a dozen to solve for each character of ore mined.

The early discoveries made in 1861 were abandoned necessarily on account of the violation of treaties with the Indians and it was not until 1873, when the Brunot Treaty was made, that mining operations started in earnest.

The year previous the Little Giant Company was organized in Chicago, anticipating the signing of this treaty, and they immediately took steps to replace their arrastres by stamps and amalgamation plates. Their ore was a brown quartz carrying free gold in quantity.

In these early days silver was high in value, ranging around $1.29 per oz. and ores were mainly sought which carried high silver values. Gray copper and galena ores were found abundantly as snperficial ores.

Mineral Point, where outcrop some of the largest and most extensive veins to be found anywhere, received much attention during 1874 and the year following. The pay streaks were large and some of the ore carried silver in hundreds and thousands of ounces to the ton. Gray copper ores were found in many properties and some rich pockets were opened. The Bill Young mine, for instance, produced $50,000 in one pocke- The Old Lout produced an ore carrying bismuth and associated with a pyrite of botrioidal crystallization which ran $3,000 per ton. The Alaskan mine produced $100,000 in one pocket of this same character of ore.

Early Silver Processes.

The most enterprising of these Mineral Point mines, however, was the San Juan Chief. It was early discovered that much of the value of the ore was being dissipated in excessive freight and treatment charges, and regretting that they did not know of some method of saving the gold, copper, lead and zinc, they decided to institute a method of treatment for silver alone. Before this some of the ore had been treated in a custom mill at Animas Forks with crusher, rolls, jigs and tables but the ore was slimed and great losses resulted. A buddle was put in but this too proved a failure.

Kempton and Thatcher finally built a small testing plant near Mineral Point, where they worked up a leaching process, supposed now to have been designed after the Augustin process, and then they built a larger mill which is now the San Juan Chief mill. Here were placed a Blake crusher, rolls, stamps, roasters and vats. The material from the mine, after being crushed and stamped, was dried in one of the roasters and then given a chloridizing roast in the other. It was then fed hot into a vat of sulphuric and nitric acid solution where it was leached for thirty or forty hours. The solution was then drawn off and run through granulated copper suspended on a cloth screen immersed in water just below the top of another tank. The silver was precipitated by the copper which in turn was recovered by being precipitated on iron sheets hung in the tank below. The silver remaining on the screens was then filter-pressed, melted in large graphite crucibles and cast into bricks. Later it was attempted to crush and roll dry, and still later to concentrate on Cammett tables and canvas.

At Gladstone where the high grade gold-silver ore of Poughkeepsie Gulch was being delivered, and at Crookes mill, two miles north of Silverton, operating on the gray copper ore of the North Star mine on King Solomon Mountain, lixiviation works of the same character were installed a short time following the San Juan Chief operations. It is clearly evident that the method was designed to treat the gray copper ores, tetrahedrite, freibergite and perhaps stromeyerite, which were subject to excessive sliming by ordinary wet concentration.

SILVER LAKE MILL AND TERMINAL EQUIPMENT TWO MILES ABOVE SILVERTON, COLO.

The causes of the failure to treat the Mineral Point and Poughkeepsie gulch ores were varied. The freight on acid and salt was extremely high; the exact figures are not now available but one of the operators of the San Juan Chief tells that it was cheaper to buy a four horse team and wagon in Denver and haul it in. It was thought that the railroad considered the acid so dangerous that in case of a wreck the rails for miles around and the two nearest stations would be entirely destroyed. Again the failure to receive payment for gold and the absolute waste of the copper and lead values worked another hardship. Then, too, the presence of much zinc mineral interfered with successful concentration and reduction.

Smelting.

In 1874 the Green smelter, just north of Silverton, was erected and it was blown in the following year, treating principally the sulphide ores of Arastra Gulch and Hazelton Mountain. This is reported to be the first successful water-jacket furnace in Colorado.

Crooke and Company erected a smelter at Lake City in 1875, and drew sulphide ore from the mines in the northern part of the county around California, Mastodon and Poughkeepsie Gulches and Mineral Point. Mather and Geist of Pueblo also bought ore around the county in competition with the Crooke smelter.

In 1876 a concentrating mill was erected at Animas Forks to treat the ores of the Red Cloud mine near Mineral Point, but this was not successful.

Jas. Cherry built the Eclipse smelter at the mouth of Grouse Gulch in 1880, at a cost of $65,000 and worked only a few tons of lead ore from the Mountain Queen mine.

The Green Smelter about this time was moved to Durango, where it became successively the New York and San Juan Smelting Company, The Durango Smelting Company, The San Juan Smelting and Mining Company, until 1895, when it became the Durango Plant of the American Smelting and Refining Company, now operating under that name.

In 1883 sampling works were operating in Silverton, under the management of E. T. Sweet, T. B. Comstock and Company, and Stoiber Bros. Two years previously the rich copper-silver deposits of the Red Mountain district had been discovered and a smelter was built by Thos. Walsh at Silverton on the site of the Martha Rose smelter ostensibly to handle these ores.

The samplers assisted in securing the production of great quantities of sulphide ores and a large tonnage of low grade ore of this character was developed. Their concentration was attempted by Edward G. Stoiber and Judge J. H. Terry.

Concentration.

Stoiber erected the first Silver Lake mill in Silver Lake Basin where forty stamps were followed by concentration tables. In 1893, a Bleichert tramway was built from this mill to the railroad and in a short time a 100-ton mill was built at the lower terminal, to which milling operations were then removed. In this latter mill were used rolls, jigs, vanners, tables, and later, canvas tables. The mill was burned in 1905, and in 1907 the present modern mill was built on the same ground.

J. H. Terry's operations were first carried on with a ten-stamp mill near Lake Em and the Sunnyside mine. Later a larger mill was built at the forks of Eureka Creek, two miles below and still later a forty-stamp mill was built at Eureka on the railroad, aerial tramways being run to the mine. This mill has been in continuous operation for many years.

The successes achieved by Judge Terry and J. H. Stoiber were emulated by many, some being fairly successful in concentrating and reducing their ores and others using the opportunity thus presented to secure the erection of mills on their properties with no ore in sight. The Gold King, an eighty-stamp mill; North star, twenty stamps; Hercules, twenty stamps; and the Iowa, a roll mill, were highly successful.

The year 1906 witnessed a mill building boom in San Juan County. The Old Hundred mill was built at enormous cost, using 1350 lb. stamps, plates, Card tables, and Sperry slimers. The slimers were soon discarded and amalgamation became their chief effort. Successful concentration was defeated through lack of classification, principally. At the Mogul mill the character of the ore, carrying high percentages of zinc, was the sticking point and the mill closed down after some very expensive work including the installation of Blake machines. The problem here remains to be solved to-day.

In 1905 and 1906 the Gold Prince mill was built at Animas Forks. This is a 500-ton mill of 100 stamps. It was built at a cost of half a million dollars, of steel and reinforced concrete and is a very modern building. It is hard to state just why its operation was not continuous but it is maintained that the ore reserves in the mine were not adequate to meet the enormous tonnage required. It was operated under receivership during 1909-10 and is now awaiting the outcome of litigation.

Very little data is available covering the milling of the sulphide ores. The savings were poor as a general rule. Several factors crept in to render the extraction difficult. A desire to stamp for purposes of amalgamation and to produce concentrates simultaneously, met with a loss of lead and other values in the tailings on account of the fine grinding. The presence of zinc, for many years a bugbear on account of the penalty inflicted by the smelters for its presence, induced further losses of included values upon being eliminated. A rhodonite gangue introduced a waste mineral of high specific gravity, which crept up on the table and interfered with the separation. The Sunnyside mine has had all these factors to contend with. Here an attempt was made to produce a zinc product with Blake machines, which was partially successful, up to the time that the zinc plant was burned.

Prospects for Cyanidation.

The treatment of the siliceous ores of the county, carrying values principally in gold and silver, is to-day presenting a broad field for operation in which past efforts have been mainly experimental. In Ice Lake Basin an attempt was made to treat the dry ores of the overlying volcanics by amalgamation. The gold is coarse and rusty, however, and a saving of about 40 per cent was all that could be obtained. Cyanide tests on these ores gave an extraction of 92 per cent of the gold and 66 per cent of the silver, in eight hours, using compressed air for agitation and aeration. Only a small loss in cyanide was experienced.

On the Eastern Range, including a rather large area, the siliceous ores containing values in free gold associated with argentite were overlooked for many years save for an occasional shipment of high grade ore. The Intersection five-stamp mill is the pioneer in this field and this was built solely for experimental purposes. By amalgamation and the production of a pyrite concentrate on an Overstrom table a saving of 86 per cent of the combined values has been made. That it is a cyaniding field is unquestionably the case. The Esmeralda, Little Maud, Intersection, Ridgeway, Buffalo Boy and Crystal Lake properties have all had highly successful tests showing a good extraction with a small loss in cyanide.

If the reports concerning the extraction of values in some of the mills are true it is not surprising that few reliable records have been kept; but at the same time it must be borne in mind that a great complexity and diversity of ores exists in this district which has tried the skill of many good mill men. Moreover they had at their disposal very little machinery which could do the work, up to a very recent time.

The introduction of electrostatic machines has been the most important step that has been attempted for many years, and that it will be successful in the production of high grade products has already been proved by exhaustive tests, chiefly on zinc-bearing ores. The success of mining operations in the San Juan is now largely dependent on ore treatment, and the backwardness of milling operations has constituted unwittingly a conservation of a greatly diversified mineral wealth.

Silverton, Colo.

Reinforced concrete jigs have been decided upon to replace wooden jigs at the mill of the Cananea Consolidated Copper Company, Cananea, Mexico. They were designed by F. J. Strachan, superintendent of concentrators, and are described in detail in the *Engineering and Mining Journal*, Oct. 7. The inside of the structure is coated with a mixture of lye, alum, cement and water.

The so-called bicarbonating process of treating oxidized or carbonate ores is carried out by dissolving the minerals in a water solution of CO_2 under pressure, forming bicarbonates of the metals. The water solution is separated from the gangue, and the different mineral salts are caused to precipitate by successively reducing the pressure to the critical points at which the various carbonates are claimed to separate out of the solution. It is claimed, for example, that with copper and zinc in solution as bicarbonates under pressure, if the pressure is gradually reduced copper will precipitate before zinc, and that this means of separation can be made quantitative. The whole process is carried out in the cold, and consists of a system of agitating and pumping, with recovery of the CO_2 when the pressure is released from the bicarbonate solutions and when the precipitated carbonates are calcined. The idea has never been carried out commercially, but has been the subject of experimentation by H. D. Rankin and E. H. Westling.

The "Paragon" Electric Furnace and Recent Developments in Metallurgy.*

BY JOH. HÄRDÉN.

Recent developments in the art of steel refining in the electric furnace have called for certain modifications in the design calculated to meet the requirements of the metallurgist.

A retrospective view of the electrometallurgy of recent years will make this clear.

When the late Dr. Kjellin, whose untimely decease we all had to lament on last New Year's eve, designed his induction furnace, about 1894, to 1898, his primary object was to create an efficient melting machine for high-class crucible steel, pure and simple, without any consideration of refining possibilities which were deemed unnecessary with the high-class raw material employed in the manufacture of such steel.

That this furnace has filled this demand, in capacities up to 4 tons and more, has in course of time been amply verified.

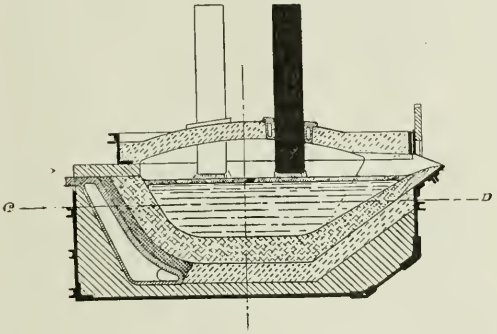


FIG. 1.—VERTICAL SECTION OF PARAGON FURNACE.

though the progress, the author regrets to state, has been very much retarded in England, chiefly on account of the power question, as "surplus" power is scarce in most steel plants and is considered more necessary for other purposes, though it has been repeatedly proved that a clear saving of from £1.10.0 to £2.0.0 ($7.50 to $10) per ton of steel, in the melting, can be effected with this furnace.

It is well known that very little refining as regards sulphur and phosphorus can be effected in the induction furnace on account of its small slag surface, and the comparatively low temperature of the slag (though degasifying is daily carried out in a large Kjellin furnace by a prominent Continental European firm, the Poldihütte A. G. in Bohemia).

The reason for this behavior is quite obvious, but a crude analogy may serve as an illustration. Imagine there are two cups of water on the table, one containing hot and the other cold water, and let us drop a lump of hard crystal sugar into both; as is well known, the sugar will dissolve much more rapidly in the hot water, also if some special salt is added, it may be found to dissolve quite readily in the hot, but not at all in the cold water.

The impurities in the steel may be considered to act precisely in the same way; they dissolve readily in a super-heated slag blanket of a temperature considerably higher than that of the steel, while a slag of the same or lower temperature than the steel may dissolve but little or none of the impurities.

As to the gases, another crude analogy may be taken for comparison. Take, for instance, a vessel containing soda-water, or any other liquid in which gases are occluded. If the flame of a burner is applied over the surface of the liquid, some of the gases may be made to escape somewhat quicker especially if the liquid is stirred, either by the action of the heat or otherwise) but if the flame is applied from beneath the liquid, the degasifying is undoubtedly carried on much more quickly, as at the same time the lower portion of the liquid is rapidly brought to the surface and the gas expanded by the heat.

These considerations, among others, brought the author to work out the design known as the "Paragon" furnace, illustrations of which are herewith reproduced.

In this furnace, the bath is heated, as is easily seen (Figs. 1 and 2), both from the surface of the slag by means of suitably arranged arcs, and also at the same time from the sides and beneath the bath, by means of side plates of "second-class conductors," similar to those which have been used for some four to five years in the Roechling-Rodenhauser furnace. In this manner the metallurgist has it in his hands to apply the maximum heat exactly where he wishes to have it, since both circuits are made to be regulated at will. Thus, during the desulphurization and dephosphorization the slag is heated to a temperature higher than that of the steel, while during the period of degasifying, the bulk of the power is conveyed to the bath through the bottom and sides.

This design brings in other important improvements, viz.: It is quite easy to start the furnace from cold by means of the arcs, which obviates the necessity of filling in liquid charge as a means of starting the non-electrode furnaces. Furthermore, the electrode question in plain arc furnaces may in many cases become a serious one, as large electrodes for furnaces

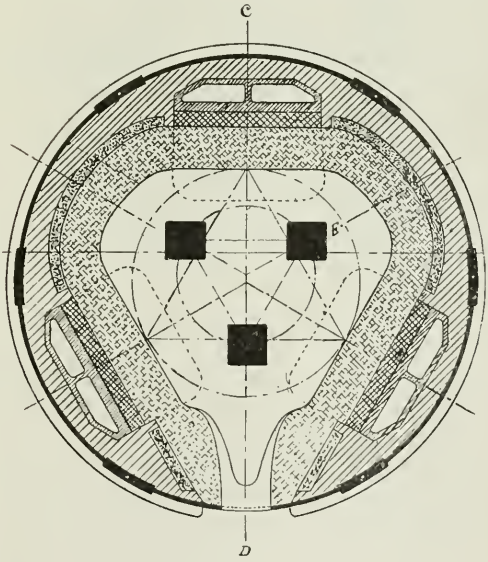


FIG. 2.—PLAN OF PARAGON FURNACE.

of greater capacities are exceedingly difficult to obtain, and always expensive, especially if the losses are considered.

Owing to the nature of the Paragon furnace, where only a smaller part of the power enters the furnace through the electrodes this drawback is considerably reduced. If, for instance, the upper limit of a plain arc furnace is, say 20 tons, on account of the difficulty in obtaining large enough electrodes, it will be found that this capacity can be doubled with the Paragon type, as only about half of the power is required to pass through the electrodes. It is true that with some designs of furnaces the electrodes may be coupled in parallel, but this is also possible in the Paragon, and this statement therefore holds good in this case also.

*A paper read before the Faraday Society on October 2, 1911.

The electrodes for a 30-ton three-phase Paragon furnace should have a cross section of 16-in. x 16-in., or 255 square inches, and in a 50-ton furnace the electrodes should be 24 in. x 24 in., or 576 square inches, which would still only give a maximum of 24.5 to 26 amp. per square inch, which, as experience has proved, is well within reasonable limits, both from an electrical and manufacturing point of view. This would correspond to plain arc furnaces having a capacity of only 12 to 14 tons and 18 to 22 tons. In fact, it may safely be said that as far as the electrodes are concerned any capacity possible with the plain arc furnace may be doubled with the Paragon design.

A further advantage in this design is the greater durability of the roof. It is well known that the roof is the part of an arc furnace which is most rapidly destroyed by the action of

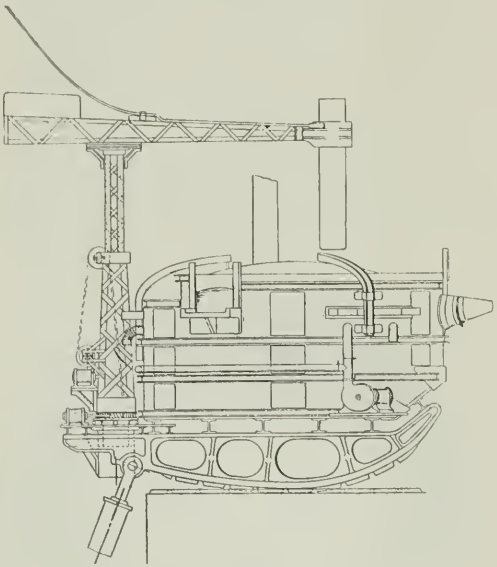


FIG. 3.—30-TON PARAGON FURNACE.

the hot gases. In the Paragon furnace the destructive action is minimized, as only a smaller part of the power is acting on the slag.

In cases where electrodes are more or less unobtainable, the same design may still be used with advantage, when gas firing is used as a substitute for the electrodes, while the electric power is applied only through the side plates. One may perhaps remark that in this case the gas firing alone will be sufficient, but it will be seen on closer consideration that the electric heating is still of great advantage, if the analogy with the liquid containing occluded gas, is remembered, which should be driven out.

Besides, the desulphurization and deoxidation require a considerable temperature, and in order to obtain this one has to blow in a sufficient quantity of air with the gas, which easily renders the flame slightly oxidizing, thus making the deoxidation of the steel difficult.

If electric heating is applied as described, the desired temperature can easily be obtained in the right direction by maintaining a reducing atmosphere, and a more thorough refining should be possible without incurring a higher cost.

Finally, the melting down of cold material with gas should be more economical in this particular case than with the electric furnace proper, since no very high temperature is required to liquefy low-grade iron; but it is the subsequent refining, apart from the dephosphorisation, which demands the higher heat energy, and this can be applied in a more efficient way by means of the side plates conducting the electric power to the furnace.

These furnaces have been patented by the author and the Gröndal-Kjellin Co., Ltd.

A trial furnace is just being built in Germany and designed by Mr. Rodenhauser and the Roechling Iron and Steel Works, where the side plates only are provided for. Experiments have proved that sufficient power can be converted to the bath in this way to give the steel the temperature required. The construction of this furnace has progressed so far that the author hoped to give here some figures of the results obtained; certain delays in the delivery of the electrical parts have, however, frustrated this, but the results are expected to be ready for publication shortly.

It is expected that this design will enable the constructors to provide furnaces up to 50 tons capacity.

Having now broached the question of very large furnaces, in fact of the largest size possible under present conditions, the author wishes to mention a few words about a new furnace of the smallest size, suitable for laboratory purposes, etc.

It is a long-felt want of the steel-maker, to be able to carry out trial melts on a small scale, especially for alloys, before making up a proper charge. This is of great importance in the case of such alloy steels in which expensive ferroalloys are a prominent constituent.

The old method of making a trial in a pot in the crucible furnace is both time-robbing and very uncertain. This new furnace should be a real boon to the crucible steel maker, as it enables him to make melts from a few pounds in weight up to several hundred pounds in a most easy and controllable way.

The furnace is known as the "Helberger" furnace. It consists of an electrically heated ordinary plumbago crucible, lined in such a way as not to react upon the charge. Any form of electric current may be used, but alternating current of from 25 to 60 cycles is preferable.

The furnace and the crucible are open at the top, so that the contents can easily be watched and additions made. The temperature is very easily regulated by means of a handle; temperatures of 3000° C. have been obtained in these furnaces, which will even permit the melting of platinum and other very refractory metals.

The furnaces are made either to be tilted or else with detachable crucible for direct pouring; they are delivered complete, ready for use, at quite reasonable prices.

The melting cost compares very favorably with other methods for small capacities; a few figures will show this:

100 kg.	steel	requires	75	kilowatthours	
100 kg.	copper	"	30	"	
1 kg.	platinum	"	10	"	

The following temperatures of the crucible were obtained:

In 8	minutes	1000°	C.	with	90	amperes,	104	volts.
" 15	"	1470	"	"	160	"	101	"
" 25	"	2150	"	"	165	"	100	"
" 45	"	2750	"	"	248	"	93	"

The temperature was still higher after the last reading, but the pyrometer used did not enable any higher readings.

From this it will be seen that any temperature required for trial purposes in steel-making may be easily obtainable.

With regard to accessories for steel-making, it may be stated that such ferroalloys as ferrotungsten are now being made in the electric furnace on a commercial scale in England by Electric Furnaces & Smelters, Ltd. in their new works at Luton. Three furnaces are in use alternatively, and alloys are produced in considerable quantities. Ferrotungsten, as low as 0.25% in carbon has been made, though 0.5% is more usual.

The power supply is 500 volts direct current which is being converted into alternating current, 25 cycles, 50 to 90 volts, by means of a motor generator.

London, England.

Iron and Steel Institute Meeting.

The autumn meeting of the Iron and Steel Institute was held in London on Oct. 5.

The Duke of Devonshire, President of the Institute, occupied the chair and was supported by two past-presidents, Sir Hugh Bell and Sir Robert Hadfield.

In his introductory address the president expressed great regret that it had been necessary to give up the original intention to hold this meeting in Italy. This was the more regrettable in view of the extensive preparations which had been made in Italy for the entertainment of the Institute. A resolution was adopted "that this meeting desires to place on record an expression of its most sincere regret that circumstances have arisen to deprive the Institute of the honor and privilege of holding the autumn meeting at Turin, under the auspices of its kindred organization, the Association for the Italian Metallurgical Industry."

There were a long list of papers of all of which printed advance copies were available. But only three of them came up for discussion. The others were read by title.

Transformation of Steel by Heat Treatment.

A paper by L. GRENET, of Paris, on "The transformations of steel within the limits of the temperatures employed in heat treatment" was read, in the absence of the author, by the secretary.

The chief conclusions of the author are as follows:

"The iron-carbon alloys partake of the general category of alloys, and no complications due to allotropic transformation of the element iron need be taken into consideration.

"It is quite easy to represent by simple diagrams the mode of influence of the special elements on the temperature of transformation on cooling.

"Quenching (rapid cooling from a temperature sufficiently high that the constituents of an alloy should not be the same as at the ordinary temperature) may, as compared with annealing (slow cooling), act as follows:—

"(*a*) By changing the nature of the constituents and, notably, by preventing wholly or partly the transformation on cooling. In this case, which is rare, so far, that is, as steels are concerned, the quenched state and the annealed state are physico-chemically different states, and their properties cannot be foreseen. It may, however, be said that in the few instances in which such quenchings are known the quenched state is often softer than the annealed state.

"(*b*) Quenching may also act by lowering the temperature of the transformation on cooling. In this, which is the most usual case, the quenched metal is always harder than the annealed metal.

"The following very general law, which satisfactorily groups the facts which have been observed as to steels of the most varied description, and appears to agree with what is known as to alloys, may be formulated. An alloy is the harder in proportion as the transformation which has given birth to its constituents at the time has taken place at a lower temperature, has been more rapid, and as the maximum temperature reached since the last transformation has been at a lower temperature and has been maintained for a shorter period of time.

"The pure metals are alloys in which one of the constituents is in the proportion of nil, and the law applies in this extreme instance, in which there is but one constituent (pure, or nearly pure, iron).

"When it is distinctly higher than the temperature of transformation, the action of the maximum temperature reached since the last transformation preponderates, and it is this which should serve as a guide in the treatment of steel.

"It would appear that the law enunciated above applies equally when the constituents for the time being result from a transformation which causes the metal to pass from the liquid state to the solid state (influence of the method of casting), as, for example, in sand or in a chill: and when they result from a transformation of the metal to the solid state either before or after transformation (as in the quenching of steels).

"The phenomena relating to the hardening of a steel by quenching belong, therefore, to the exceedingly common category.

"So far as the real cause of hardening is concerned, it cannot be said to be known. We are thus led to the conclusion that hardening produced by quenching is due solely to the fineness of the structure. The comparison of the quenching of a solid solution to the quenching of a liquid solution (cast in chills) appears, therefore, to the author to be nearly perfect. In some instances the constituents of a metal vary with the rate of cooling (white or grey pig iron, austenitic steels); but, generally speaking, the constituents of the metal are independent of the rate of cooling, and the law connecting the hardness of the metal with the conditions under which the transformation has been effected applies. Annealing is liquation within a solid, and this liquation would appear to be limited only to the extent of the complete separation of the constituents. Exceedingly slow cooling isolates the constituents in steels or in alloys.

"Between the quenching of metal in a state of solid solution and the quenching of a liquid metal there are naturally enormous differences from the practical point of view. The cooling of a solid solution takes place without the intervention of a mold, and may be very rapid; the constituents separating from the liquid solution are more readily able to segregate by the intervention of the liquid solution not as yet decomposed than the constituents separating from the solid solution. The quenching of a solid solution is therefore more energetic than the quenching of a liquid solution.

"Further, while a liquid solution presents no discontinuity, a solid solution is divided into grains like the metals formed of a single constituent; the grains are larger in proportion as the metal has been heated longer and at a higher temperature. At the time of the formation of the 'cold stable state,' the texture of this state may depend partly on the texture on the 'hot stable state' which has preceded the state at the time being.

"What is important to remember as a guide in the treatment of alloys is the relation—independent of any special conceptions as to the metal iron—which connects the hardness of an alloy with the conditions under which the transformation on cooling is produced, and, in particular, with the maximum temperature to which the alloy has been raised since the setting at liberty of its constituents at the time being."

The paper was discussed at considerable length. Sir Robert Hadfield referred to the author's statement that if quenching acts by preventing wholly or partly the transformation in cooling, the quenched state is often softer than the annealed state. He could confirm this peculiar fact from observations on manganese steel.

Prof. I. O. Arnold said the paper showed that the rival schools of thought were now getting nearer together. He was especially glad that the author had come to the conclusion reached by the Sheffield school of metallographists long ago that the so-called martensite was not a constituent at all, but a structure.

Dr. J. E. Stead criticized the terminology employed by the author as differing in various respects from the usual one.

Dr. W. Rosenhain remarked that the author's opinion on the cause of hardening differed very much from that usually accepted in France or England. The usual view was that the hardening effect in quenching arose from the possible prevention of transformations which would occur on slow cooling. According to Grenet, however, hardness resulted from transformation occurring very rapidly and their products being present in extremely fine division. But this view would require experimental evidence to establish it.

Iron-Manganese Alloys.

Prof. J. O. Arnold, of the University of Sheffield, then presented a paper by himself and F. K. Knowles on "the mechanical influence of carbon on alloys of iron and manganese."

Arnold and Read had formerly expressed the opinion that "the influence of carbon is very considerable, but is by no means the dominating factor, which without doubt is the manganese."

The object of the present paper is to prove the truth of the foregoing enunciation.

Two series of steels were made, in both of which the manganese ranged from 1 to 20 per cent. In one series, however, the carbon was about 0.85 per cent, while in the other it was kept under 0.1 per cent. The results are given of tests of the turning properties, static mechanical (tensile) tests, and dynamical stress-strain tests (alternations induced). Microphotographs of the structure of four typical nearly carbonless alloys are given.

Sir Robert Hadfield, in discussing the paper, gave hardness figures of the manganese steels, obtained by the Brinel method of testing hardness, which he said should find much wider applications, as very valuable practical conclusions could be drawn from it.

Dr. J. E. Stead urged that the investigation should be extended to other properties of the manganese steels.

Temperature Influences on Carbon and Iron.

Mr. E. Adamson, of Sheffield, in his paper on "temperature influences on carbon and iron" endeavored to show that the determination of silicon alone is not sufficient in the chemical grading of iron.

The author gives facts to show that carbon is precipitated as free carbon virtually in the absence of silicon, and in the author's opinion this precipitation in the case of the manufacture of pig iron, primarily depends upon the original temperature to which the iron is raised in the blast furnaec.

The higher the temperature in the blast furnace the higher the carbon in the product—within limits—for it is well known that cold-blast iron is lower in total carbon than hot-blast iron, although the full force of this fact has perhaps been overlooked. What this amounts to is that the temperature of the furnace decides the fracture of grey iron as well as of the lower grades, regardless of the contents of silicon, sulphur, phosphorus and manganese.

The author distinguishes between three different varieties of free carbons and discusses under what conditions they appear. Analyses are given to show that under no conditions can it be said that the percentage of silicon controls the percentage of combined carbon or the graphite or the ratio of one form to the other. Under precisely the same conditions of temperature and time, additions of silicon may show fairly regular variations in the carbons, but under the same constant conditions phosphorus and manganese would do the same.

"The facts stated in this paper are of vital importance in considering the question of the grading of pig iron, and to a large extent explain the objections of many in this country to complete change in the methods used at present to one of grading by practically silicon contents only. The author has shown on several occasions that it is not possible to regulate tests by silicon only, and has no hesitation in saying that the fracture indication is quite as satisfactory as varying the silicon, if the user is fully conversant with fracture indications.

"Continual experience in the use of pig irons in the foundry shows that the same tests can be obtained within very wide ranges of silicon contents. Of course, if it is a question of commerce, there should be no objections to selling irons by silicon contents if this is demanded, and full responsibility is taken by the purchaser: but when it becomes a question as to whether such a method of grading is the best, so as to be of the greatest use to the greatest number, the reply must be an emphatic negative, for it would appear that carbon and its various conditions, due, as shown, to influences entirely outside of analyses, is of primary importance."

In the discussion Dr. J. E. Stead said that the paper was an attack upon analysis and also an attack upon those who depended upon the determination of silicon for pig iron. He thought that all those who had anything to do with foundry work understood thoroughly that the determination of silicon was most important.

Mr. Adamson replied that he was not attacking the chemist, who was a very valuable man as he enabled the iron founder to use grades of iron which he would otherwise not be able to employ. But he attacked grading by analysis pure and simple, and this meant in fact grading by the silicon content.

The other papers were read by title. Nine of them were from the pens of Italian engineers an gave very interesting data on metallurgy in Italy. On account of limitations of space only the two papers on case-hardening are abstracted in this issue. Abstracts of the other papers are reserved for our next issue.

Case Hardening of Steel by Carbon Monoxide and Carbon

Dr. F. Giolitti's paper gives first a summary of the facts established in recent years by the author in co-operation with others on the theory of case-hardening (the original papers having appeared in *Gazzetta Chim. Ital.*, vol. 38, No. 2, 1908; vol. 39, No. 2, 1909; vol. 40, No. 1, 1910; *Rendiconti della Soc. Chim. di Roma*, vol. 6, No. 17, 1908; *Atti della Reali Accad. delle Scienze di Torino*, vol 45, 1910; vol. 46, 1911; *Rassegna Miner., Metall. e Chim.*, vol. 34, No. 18, 1911, and September, 1911; *La Metall. Ital.*, August, 1911, and September, 1911.) These principal facts are as follows:

Where case-hardening is carried out with solid cementing agents having a carbon base, the carburizing effect of the free carbon on the iron, the materials being in simple contact and without the intervention of gaseous carbon compounds, is exceedingly weak, and in any case is entirely negligible in industrial practice.

Where case-hardening is effected with the solid case-hardening agents ordinarily employed in the industry, the specific effect of the nitrogen is very weak. Only with the cementing agents containing a high proportion of the cyanogen compounds (alkaline cyanides, ferrocyanides, etc.), does the direct action of volatile nitrogen comopunds have any marked effect.

As compared with the cementing effect of the solid agents ordinarily employed in the industry, the specific direct carburizing effect of carbon monoxide preponderates enormously over every other carburizing effect.

Pure carbon monoxide carburizes iron at all temperatures within the range (700 deg. to 1300 deg. C.), at which the process of case-hardening can be performed by means of any other medium whatever. Moreover, the rate of case-hardening (by which is understood the depth of carburization which can be obtained in a given time) when working under suitable conditions is greatest when carbon monoxide, or a mixture in which the carbon monoxide can efficaciously exercise its specific carburizing effect, is used as the agent.

This specific carburizing effect exerted by the carbon monoxide on the iron at high temperature is due to a series of chemical reactions, the course and state of equilibrium of which has been actually observed with precision. Moreover, the conditions of equilibrium of the systems in which these reactions take place are in general comprised within the ranges of temperature and pressures ordinarily employed in practice. It is, therefore, possible to obtain with certainty a predetermined result using case-hardening agents whose activity is due, if not exclusively at least very largely, to the specific carburizing action of the carbon monoxide. More particularly is it possible to obtain with such agents carburized zones in which the concentration of the carbon does not exceed a predetermined maximum limit and varies in a well-defined degree toward the inside of the carburized zone. Such definite results, variable at all within sufficiently wide limits, are obtained by varying, in accordance

with fixed rules, the temperature at which the case-hardening is performed, the pressure of the carburizing gas, and the amount of carbon monoxide which, in a given time, comes in contact with the unit surface of the steel to be carburized.

The results obtained by the use of carbon monoxide as an agent vary regularly, other conditions being equal, as the chemical composition of the steel to be carburized is varied.

It is possible to vary regularly and within fairly wide limits the characteristics of the desired product by subjecting the steel to the action of substances along with carbon monoxide which are capable of modifying the conditions of equilibrium of the chemical systems under which the reactions due to the specific carburizing effect of the carbon monoxide are completed. Such substances may be gases, such as hydrocarbons or nitrogen, or they may be solids such as carbon in various forms, and their actions can proceed simultaneously with that of the carbon monoxide throughout the whole cementation period or during a portion only of it.

In particular, by means of the agents, the activity of which is due to the specific carburizing effect of carbon monoxide, it is possible to obtain with ease and certainty—whatever kind of steel is being operated upon—soft case-hardenings and graduated case-hardenings; that is to say, carburized zones in which the concentration of the carbon without being excessive in the outer layers, diminishes slowly and with regularity in the succeeding deeper layers. This is the essential condition for the avoidance of the dangerous phenomena of brittleness and peeling, which defects manifest themselves so frequently in steel pieces case-hardened by the processes ordinarily used in the industry.

The chemical reactions produced by agents in which cyanogen is the active element, are at present but imperfectly understood, particularly as regards their condition of equilibrium upon which depends the concentration of carbon in the carburized zones. It is, however, certain in the conditions under which case-hardening should be performed in practice that the conditions of equilibrium just now alluded to should closely correspond to the strength of the very high concentration of carbon passing into solution in the iron. Thus it happens that the cyanides, ferro-cyanides, and other derivatives of cyanogen, if used alone as case-hardening agents, always give rise to too rapid (energetic) a case-hardening. That is, carburized zones are produced in which the concentration of the carbon is excessively high in the outer layer up to a certain depth and is then suddenly lowered in the succeeding layers. Zones of that type produce brittleness and peeling.

Further, gaseous or volatile hydrocarbons, when used alone as agents, also give rise to too rapid case-hardening, the causes being identical with those referred to in the case of cyanogen and its compounds.

In the light of the facts recited in the foregoing, the great advantage of the use of agents, the activity of which is due, if not exclusively, at least principally, to the specific carburizing action of carbon monoxide, is clear. In order to obtain the best results with such an agent, it is necessary to satisfy the following fundamental conditions:

1. The chemical composition of the agent should be absolutely definite, and should be accurately known.

2. The compounds should be as simple as possible.

3. The reactions which take place during the case-hardening process between the various constituents of the agent and those of the steel should be simple, and should proceed rapidly—under the conditions most easily obtaining in practice—to a well-defined state of equilibrium corresponding to definite concentrations of carbon in the carburized zones.

If pure carbon monoxide alone is used, the concentrations of carbon in the carburized zones are generally too low. It is, therefore, preferable to use along with the carbon monoxide small quantities of hydrocarbons of known composition, or to use along with the carbon monoxide solid carbon in a properly divided state. It is the latter method—the combination of carbon monoxide with free carbon—which is advocated by the author.

Among the numerous advantages claimed for this method by the author are great speed of penetration of the carburized zone; great uniformity in the distribution of carbon in the carburized zone; possibility and ease of regulation of carbon concentration so as to get maximum hardness with minimum brittleness; absolute security against the introduction of foreign substances into the steel; a minimum of deformation and change of volume of the steel pieces.

While the process may be carried out in the ordinary horizontal muffle generally used for case-hardening, the following special furnace offers decided advantages.

As shown in the illustration on p. 600, it consists of cylindrical muffles arranged vertically and heated by producer gas. It is a double-muffle furnace provided with regenerators, of the kind constructed by Messrs. C. M. Stein, of Paris, for the Sampierdarena Engineering Works of Messrs. G. Ansaldo, Armstrong & Co.

The parts of the apparatus particularly applied to the case-hardening process are represented in the illustration which shows a section through the center of the vertical muffles of the furnace with the accessory apparatus completely mounted.

Inside each of the muffles of refractory material is placed a cylindrical retort of mild steel, A, the external diameter of which is about 10 to 20 millimeters less than the internal diameter of the muffle. Mannesmann seamless tubes serve very well for making the retorts, as they are easily obtainable in stock sizes up to 35 centimeters diameter. For retorts of larger diameter, as in the case of the furnace here represented, tubes welded by the autogenous process are used. The retort is supported on a bottom flange attached to a frame fixed to the brickwork of the furnace. The arrangement which serves to hold the retort, whether it is being supported by the bottom flange or by the upper flange, H, intended to receive the cover, is such as to enable one retort to be replaced by another in a few minutes. To the bottom flange is attached a cast-iron funnel B, closed at the bottom by a non-return valve Q.

The steel nozzle C, admitting the carburizing gas, passes through an opening in the side of the funnel, provided with a good-sized boss. The gas is introduced through the screwed tube D, and through the nozzle C. This latter is connected to the hollow cast-steel dish E, intended to support the pieces to be case-hardened by means of the disc F, consisting of a steel plate covered with refractory material.

The carbon monoxide, after passing through the hole bored axially in the nozzle C, is delivered into a small distributing device placed inside the dish E, from which it passes to the case-hardening chamber G, through a number of holes provided in the disc F. During case-hardening the dish E is supported upon a series of projections on the same lower flange to which the funnel B is attached. This ring has thus to sustain the entire weight of the dish E, the disc F and any articles to be case-hardened which may be resting directly or indirectly on the disc. The section shows exactly the position of the various parts during case-hardening.

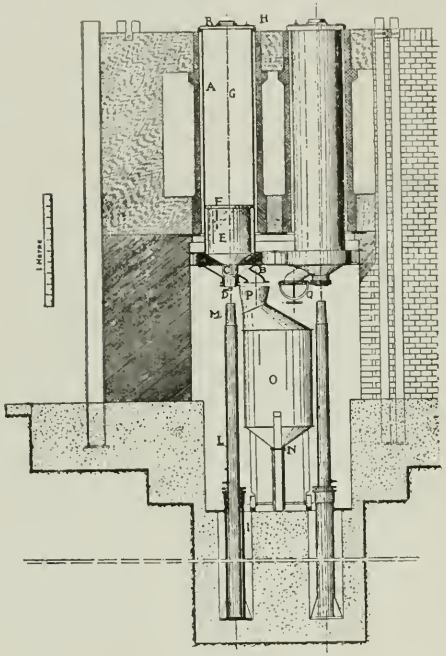
In the case of spur wheels, the maximum diameter of which might be 100 to 150 mm less than the internal diameter of the retort, the first thing to be done is to unscrew the tube D; then by means of the plunger L, actuated by the hydraulic cylinder I, the whole of the parts C, E and F are lifted together, the blunt conical end of the plunger M engaging with these as it rises. The plunger LM is cooled internally with water circulation, so as to avoid the heating of it in case it should accidentally remain raised inside the retort for too long a period.

When the piston L has reached the end of its stroke the upper surface of the disc F being now only about 30 cm below the cover of the retort; the cover is removed, and the wheels to be case-hardened are piled horizontally one on top of the other upon the disc F. As the wheels are inserted

one by one, the apparatus carrying the weighed portions C, E and F gradually sinks, and the charge is complete when the apparatus has reached the lowest position indicated in the figure. In that position the last wheel charged should be at least 30 cm below the upper edge of the retort.

Generally, the articles to be case-hardened are charged hot at a temperature of about 800 to 900°C, this being a matter of prime importance, as it affects most favorably both the quality of the product and the economy and regularity of the operation.

For the preliminary heating of the pieces a small coal-fired muffle will serve, since the uniformity of temperature which



CASE-HARDENING FURNACE.

can be obtained in such a furnace is quite sufficient for the purpose. If, however, the case-hardening furnace is not being worked to its fullest capacity, one of the vertical muffles may be used for the preliminary heating of the articles, while the case-hardening of the charge is proceeding in the other. In this case the best method of working the furnace is to use the muffles alternately, the one for case-hardening and the other for preheating the pieces.

As soon as the articles to be case-hardened have been inserted as described, the cover is placed on the retort and the bottom discharge pipe N, leading from the iron box O, is passed through the central opening in the cover. This box is full of the granular carbon still hot (generally at about 900°C.) which was discharged from the retort after use during the last case-hardening operation just concluded. The box is easily swung above the muffles, being suspended by a tackle from a swinging arm.

The outer walls of the box, notwithstanding that its contents of granular carbon are at a temperature of nearly 1000°C., do not become heated above 200° to 250°C. during the time required for the loading and unloading of the retort, even when working under the most unfavorable conditions. This is due to the small thermic capacity and the very low conductivity of the mass of granular carbon, which, although at a high temperature, can only impart to the walls of the box containing it a small proportion of its heat in a unit of time. The heat is quickly dissipated simply by the cooling action of the air currents passing around it.

The box O having been adjusted as indicated, the butterfly valve which closes the low end of the opening N is gradually opened, and in a few seconds the whole of the free space in the retort surrounding the pieces is filled with glowing granular carbon. The hot granular carbon employed by the author for case-hardening with his mixed medium constitutes a mass which, particularly at high temperatures, is endowed with a mobility comparable to that of a liquid. On this account it is able to penetrate simply by its own weight into all crevices among the articles to be case-hardened, and between these and the walls of the retort. The filling up may be further assisted by means of open rods introduced through holes in the cover and manipulated by the operator who stands on the furnace.

When the granular carbon has been filled in up to a level of 2 cm or 3 cm below the upper edge of the retort, the butterfly valve of the tube N is closed, the box O is lifted, the central opening of the cover is closed, the plunger L is lowered right down and the tube D is screwed into the nozzle C, through which the carbon monoxide or dioxide, pure or mixed with air, is admitted gradually to the distributor E and then into the retort. In order to regulate the current of carbon monoxide, an ordinary gas meter of the dry type is used. The case-hardening then proceeds without further trouble, the only thing that is necessary to be watched and regulated being the current of gas and the temperature.

For the observation of the temperature, an operation which, with the type of furnace described and when adjusted properly, does not often require to be made, an ordinary thermoelectric pyrometer is used by passing it through any one of the holes in the cover from time to time and inserting it into various parts of the mass of granular carbon. The temperature should be maintained quite constantly at a fixed point between 900° and 1100° C., chosen in accordance with the rules laid down by the author in the reports previously referred to.

The gas issuing from the retort through the pipe fixed in the cover can be collected in a gasometer, and after suitable regeneration it will serve for further heats. The necessary length of the heating period can be determined with the greatest precision, and is dependent on the result which it is required to obtain. When the heating has proceeded long enough, the current of carbon monoxide is shut off, the tube D is removed and the discharge of the retort is undertaken.

Where results of a very special nature are not aimed at, the discharging operation begins by withdrawing the whole of the granular carbon at once, and removing it through the non-return valve Q in the box O, now placed underneath.

Where, however, exceptionally graduated case-hardening is required, the specific action of the carbon monoxide is isolated during the last phase of the case-hardening from the direct action of the granular carbon. In that case the box O is placed so that the neck of the funnel B is inserted into the open end of the tube P, the box for this purpose being placed in the chamber below the muffle, where it can be maneuvered on a small bogie.

The non-return valve is then opened gradually and a quantity of the granulated carbon is drawn out from the retort.[1] Only those parts of the case-hardened pieces are uncovered in which a well-graduated case-hardening is required. Sufficient solid carbon is retained in the retort to ensure the desired chemical equilibrium of the gas. The remainder of the carbon is removed from the retort after the second phase of the case-hardening with isolated carbon monoxide has proceeded long enough to attain the desired results.

In either case as soon as the whole of the granular carbon

[1]The carbon contained in the retort falls out from below, passing first through a series of holes in the bottom flange upon which the apparatus C, E, F rest. It was, however, not possible to show all these holes in the accompanying sectional view of the furnace.

has been evacuated from the retort, the cover of the latter is lifted. In performing that operation the operator must be careful to remain at some distance from the upper open end, because the air which enters forms, with the carbon monoxide contained in it, a mixture which explodes suddenly at the high temperature of the retort. The explosion, which in any case is light and not at all dangerous, may be avoided by rapidly scouring the muffle with a little carbon dioxide after the removal of all the solid carbon. Care must be exercised, however, to perform this operation as quickly as possible, in order to prevent the carbon dioxide causing a superficial decarburization of the case-hardened pieces.

When the pieces rest direct on the disc F, as in the present supposed case, they may be lifted by means of the appliances C, E, F, up to the open top of the muffle without first discharging all the granular carbon from the retort. In that manner the light explosion just referred to is avoided, but the picking up of the pieces for quenching is made more difficult, and a considerable quantity of the solid carbon is apt to get burnt in the retort.

At all events, whether the carbon is removed from the retort or not, as soon as the cover R is removed the plunger L is raised until its conical head beds itself in the recess provided for it in the part C. The hydraulic ram then continues to move slowly, gently lifting the weight-supports, C, E, F, with the case-hardened pieces resting on them. It only remains then for the operator standing above the furnace near the open muffle to pick up the case-hardened pieces one by one as they come within reach and quench them in the tank, also placed above the furnace (in case double quenching is desired), or bury them in hot ashes. When all the pieces have been removed, the furnace is ready for a fresh charge.

The method of loading the pieces to be case-hardened into the retort varies greatly according to their shape or size. Pieces of cylindrical shape can be piled on one another as in the foregoing example. If the pieces are of small diameter, such as pinions, cutters, cups and rings for ball bearings, several columns can be built up for one heat; but in that case it is desirable to keep the pieces in position by threading them on iron rods, assuming that the external surfaces only require to be case-hardened.

For cylindrical pieces to be case-hardened internally, such as rings for ball bearings, the rods are placed outside the columns. Pieces of great length or of irregular shape have also been suitably supported in the retort. Finally, very small pieces are placed in cages formed of iron framework held together with iron wire netting. Such pieces can also be laid free on successive layers of granular carbon.

The operation of charging the retort may vary from 4½ to 6 minutes and may be increased to 10 or 12 minutes. The time required for the complete discharge of the retort is 3½ to 5½ minutes, though it may extend to 13 or 14 minutes.

Further, in the case of the process described and in almost all cases occurring in the practical production of case-hardened pieces for machine construction, in which the carburized zone is seldom required to exceed 2 mm in thickness, the period of case-hardening never exceeds 2 hours, and may even be reduced to one hour, the time being reckoned from the moment of completing the charging operation to that at which discharging begins. It, therefore, includes the time necessary for bringing the temperature up to working point, which requires not more than 10 minutes, owing to the fact that both the solid carbon and the pieces are already hot when charged.

The foregoing particulars show that in practice, under ordinary working conditions, the time for a complete heat is less than 2½ hours, and since the weight of the charge, which retorts of the size shown in the accompanying sketch can conveniently hold, varies from a minimum of 100 kg in exceptionally unfavorable cases to a maximum of over 500 kg, it will be seen that the productive capacity of each retort ranges from a minimum of 1 ton to a maximum of 5 tons of case-hardened steel pieces during a working day of 24 hours.

The chief advantages of the furnace described over horizontal muffles are greater speed of working, greater simplicity of operating, and increased uniformity of the treatment of all the pieces forming the charge in the muffle.

The author finally gives two examples which show what can be done in practice with this process, and especially how the distribution of the carbon in the carburized zones can be graduated by isolating the action of the carbon monoxide during a part of the carburizing period.

The following data refer to case-hardenings of medium depth from 5 mm to 10 mm) in which the application of the process in question presents the greatest technical interest, particularly in the case of very deep case-hardening. The tables give the contents of carbon in successive layers of a thickness of 0.5 mm, obtained by case-hardening with the mixed agent, followed by exposure to the action of carbon monoxide isolated in the manner previously described.

First Case.—The material used was an ordinary mild steel containing 0.12 per cent C, 0.06 Si, 0.47 Mn, 0.02S, 0.03 P.

The figures in the column marked *a* give the amount of carbon in the different layers after case-hardening for ten hours at 1100°C. with the mixed agent; *b* gives the amount of carbon after reheating the same carburized zone for five hours at 1100° C. in isolated carbon monoxide; *c* gives the amount of carbon after reheating the same zone for another five hours (altogether ten hours of reheating) at 1100° C. in isolated carbon monoxide

No. of range of single layers analyzed.	Distance of layer analyzed from surface.	Amount of carbon. *a*	*b*	*c*
1	0.5 mm	1.17	0.85	0.70
5	2.5	0.81	0.78	0.67
10	5.0	0.34	0.46	0.53
15	7.5	(0.12)	0.25	0.39

Second Case.—The material used was a chromium-nickel steel of the kind ordinarily used for Krupp armor-plates. On this account the data here given relative to case-hardening at moderate depth are of peculiar interest. The steel contained 0.33 per cent C, 0.06 Si, 1.15 Mn, 0.02 S, 0.015 P, 1.50 Cr, 3.17 Ni.

The results after each single phase of treatment were as follows The figures under *a* give the amount of carbon after case-hardening for ten hours at 1100°C. with the mixed agent, *b* the amount of carbon after reheating the same carburized zone for five hours at 1100°C. in isolated carbon monoxide, *c* the amount of carbon after reheating the same zones for a further five hours (total reheating period ten hours) at 1100°C. is isolated carbon monoxide.

No. of range of single layers analyzed.	Distance of layer analyzed from surface.	Amount of carbon. *a*	*b*	*c*
1	0.5 mm	1.16	0.80	0.86
5	2.5	0.81	0.77	0.87
10	5.0	0.50	0.58	0.75
15	7.5	(0.33)	0.45	0.59

The author concludes that his process permits to obtain carburized zones in which are eliminated the principal causes of brittleness and peeling which always occur, in a greater or less degree, in steels case-hardened by the ordinary process. The principal causes of these defects are excessive concentration of carbon in superficial zones of the case-hardened pieces, and the rapid diminution of the carbon in passing to the deeper layers. It is also worthy of note that this rapid diminution contributes in a high degree to render more intense the phenomenon of liquation of the cementite or of the corresponding complex carbides of the various special steels, and of the ferrite. These phenomena in their turn are productive of the sudden local variations of the concentration of carbon in the carburized zones to which is due the peeling off of the carburized and quenched zones.

Finally the ability to alter the form of the curves of case-hardening, as shown in the two examples, suggests that case-hardening can also be successfully applied to those special steels on the exterior of which the ordinary process of case-hardening yields carburized zones, which are practically useless on account of their excessive brittleness.

Case-Hardening by Means of Compressed Gases.

A paper by F. Giolitti and F. Carnevali, both of Turin, Italy, deals with the effect of gaseous pressure in case-hardening by means of compressed gases, especially by the method (described in the preceding paper in detail) employing the "mixed agent" of carbon monoxide gas and free carbon.

The authors have formerly found that the characteristics of cementation zones vary in notable fashion concurrently with the variation in pressure of the carburizing gas, in the sense that an increase of pressure produces (all other conditions remaining unchanged) an increase in the depth of the cementation zone obtained within a given time and an increased concentration of carbon within the zone itself.

The authors have investigated this phenomenon quite carefully. The arrangement of their experiments is described in detail in the paper and the results of the treatment of six different steels are given. The case-hardening was accomplished by means of a mass of wood charcoal traversed by a slow current of carbon dioxide. This gas supplies with great rapidity and without any possible excess of carbon monoxide, a mixture of carbon dioxide and carbon monoxide of a concentration exactly corresponding to the equilibrium with free carbon, under the conditions of temperature and pressure employed in the operation.

The tests tabulated in the paper show clearly that an increase in pressure of the carburizing gas increases the depth of the case-hardening and the concentration of the carbon in the carburized zone as stated before.

A secondary effect observed is that an increase in the velocity of the current of carbon dioxide tends to cause a diminution in the intensity of case-hardening to such an extent as to eliminate the effects of increase of pressure. This is due to the fact that, on account of the high speed of the carbon dioxide, the carbon monoxide gas does not reach a concentration corresponding to equilibrium.

A third effect which occurs frequently in the course of case-hardening operations carried out under high pressure by means of a case-hardening mixture based on carbon monoxide is the superficial oxidation of the case-hardened steel. This effect is made the subject of a long discussion of the theory of the process.

Finally the authors remark that an understanding of the theory of the process permits to draw valuable practical conclusions.

One example is the fact that the case-hardening of steels with rather high percentages of chromium (more than 4 or 5 per cent) with "mixed cement" at a temperature not higher than 1000 deg. C. (a limit which in many cases cannot be exceeded) is accompanied by considerable oxidation of the metal, even when the process is conducted below ordinary atmospheric pressure.

Now, the results of the tests of the authors suggest immediately a method of avoiding this oxidation; it will suffice, indeed, to reduce the pressure of the carburizing gaseous mixture ($CO + CO_2$) below the minimum pressure of oxidation characteristic of the particular metal at the temperature at which it is being case-hardened. This can be in many cases easily accomplished by "diluting" with air the carbon dioxide which is circulating through the mass of granular carbon. In this way the carburizing gaseous mixture acts under partly reduced pressure, since it is "diluted" by the nitrogen of the atmosphere.

Experience has abundantly confirmed the efficacy of this method.

Abstracts of the other papers presented at the meeting will be given in our next issue.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

The Institute of Metals.

The autumn meeting of the Institute of Metals took place on the 20th, 21st and 22nd of September at Newcastle-on-Tyne. It is noteworthy that the whole of the papers dealt with non-ferrous metals and alloys, and that the subject of corrosion in connection with such metals and alloys received a considerable share of attention. Abstracts of the papers presented were given in the October issue, page 515. In the following we give an account of the discussions:

Non-Ferrous Metals in Railway Practices.

Mr. George Hughes, of the Lancashire & Yorkshire Railway, read a paper on "Non-ferrous Metals in Railway Work" (abstracted in the October issue, p. 516.) The paper extended only to the use of non-ferrous metals in the locomotive and rolling-stock departments, and was peculiarly valuable on account of the author's description of the means of overcoming difficulties liable to occur in course of manufacture. The chief metals dealt with were copper and copper alloys and aluminium.

The discussion was opened by Mr. A. V. Hussey who said that the successful soldering of aluminium yet remained to be achieved; but the autogenous welding process was being largely used with satisfactory results.

Dr. G. H. Desch said that the statement that copper containing 0.3 per cent of arsenic was preferable for fire-box stays was not in accordance with usual experience.

Mr. H. Woore remarked that with phosphor-bronze slide valves running on cast-iron faces, he had found the wear to vary very widely, and he was inclined to ascribe this variation to differences in the cast iron.

Prof. Huntington, with regard to the use of bronze slide valves working with super-heated steam, said that the Lancashire and Yorkshire Railway seemed to have had very good fortune, but as the employment of super-heated steam in locomotives increased it was certain that the discussion of the subject would be revived.

Mr. L. Boeddecker desired to point out that the author had neglected the good points of copper-nickel alloys. The alloy containing 2 per cent of nickel possessed the remarkable property of increasing its tensile strength, after long annealing, from 15.2 to 16.1 tons; and there was a further improvement to 17.1 tons per square inch after five hours' heating in the fire-box of a locomotive.

Prof. H. Louis said that, although the paper compared copper with steel fire-box plates, no comparative figures were given for best Yorkshire iron, and he thought figures for that material were desirable.

Prof. Carpenter pointed out that the specification for copper boiler tubes especially directed that the ends should not be annealed; but this was quite contrary to the proposition of Dr. Beilby that all copper which was to be used for superheating should be annealed at 300° to 350°C.

Sir Gerard Muntz said it was gratifying to know that copper nickel alloys were being experimented with, and it was probable that they would displace the alloys of copper with arsenic. The alloy containing 2 per cent of nickel was stated to retain a tensile strength of 14 tons up to a temperature of 700°C. Pure copper, under the same conditions, gave only six tons; and the usual arsenical copper about ten tons per square inch. There was one point which frequently appeared to be lost sight of in the practical use of copper, and that was the very great influence of the oxygen content on the qualities of the copper.

Hard-Drawn Copper.

A paper on "The Mechanical Qualities of Hard-drawn Copper" was contributed by Mr. D. R. Pye, who directed attention to the want of any satisfactory standard specification for

the properties of hard-drawn copper and the consequent variation of metals so described when obtained from different manufacturers (in October issue, p. 516). He suggested as a satisfactory definition of hard-drawn copper, "Hard-drawn copper, when in the form of circular wires, should have a tensile strength not less than that given by the formula $T = 30 - 20\ D$, and an extension per cent on a marked 10 in. length including the point of fracture, of not less than that given by the formula $e = 5\ D$, where T = tensile strength in tons per square inch of original section, e = extension per cent, and D = diameter of the circular rod in inches."

Mr. A. P. Trotter said that further tests were clearly necessary, as tests of material furnished by one manufacturer only did not constitute a satisfactory basis for action; and manufacturers were chary of making their results public property. They would not give any description of their processes; and users of hard-drawn copper had to be content with what they could get.

Sir Gerard Muntz said it was curious that the amount of work put upon the metals was not taken into account in drawing up specifications. It was his experience that the same specifications were received repeatedly both for 16-gauge wire and for ½-in. wire, and of course the differences of diameter should be allowed for.

Conductivity and Constitution of Alloys.

Dr. W. M. GUERTLER read a paper on "The Electrical Conductivity and Constitution of Alloys" in which he discussed the determination of the constitution of alloys from their conductivity (abstracted in the October issue, p. 517).

Dr. W. Rosenhain opened the discussion and said that the paper was useful in that it directed attention to the limitations of the methods habitually employed by metallurgists in ascertaining the composition of alloys, especially the limitations of thermal analyses. All would welcome the additional method of investigation, but that method also had its limitations. He did not intend to depreciate the value of electrical methods, and thought that metallurgists had unduly neglected those methods in giving preference to thermal results and microstructure.

Dr. C. H. Desch considered that the author had furnished distinct evidence of the existence of inter-metallic compounds in solid solution; and that had recently been extended to the region above the liquidus curve. There was thus proof of the existence of compounds in a liquid state such as had hardly been obtained before with aqueous solutions.

Prof. T. Turner said he was glad that the author had indicated the inutility of expecting to get good conductance in an alloy. Metallurgists seemed to be unable to solve the problem; and it remained to work for the improvement of the mechanical properties of the alloy to a greater extent than the conductivity decreased.

Copper-Tin Alloys.

"Volume Changes in the Alloys of Copper with Tin" was the title of a paper by Mr. J. L. HAUGHTON and Prof. T. TURNER, from which the following conclusions were drawn: (1) the expansion during solidification is closely related to the crystallization interval, except in two regions, but the nature of the relationship is not clear; (2) allotropic changes, during and after solidification, cause variations in the coefficient of expansion; (3) accurate determination of the densities of copper-rich bars on original castings is rendered impossible by blow-holes; and (4) the hardness curve does not fit in with Kurnakow and Shemtschusny's theory, probably owing to the metastable condition in which the bars exist. (See also October issue, p. 515.)

In the course of a brief discussion Dr. Rosenhain remarked that the method described in the paper was suitable for rough analysis, but not for ascertaining the composition of the complicated alloys of the copper-zinc series.

Dr. Desch said that the effect of crystal thrust complicated the interpretation of the results.

Prof. T. Turner, in reply, claimed that the experimental methods adopted were duly adequate; and that the volume changes measured corresponded exactly with those obtained in practice. The results arrived at had moreover been corroborated by the researches of Keep and Wüst.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the eight months ended on the 31st August, put the value of imports of iron and steel, including manufactures, at £7,369,674, and exports at £28,605,040, the respective increases being £1,615,364 and £307,944 as compared with the corresponding period of last year. The imports of other metals, including manufactures, were £18,160,533, and exports £7,071,749; increases of £2,701,540 and £397,887. Imports of electrical goods reached £887,215, with exports of £1,781,520, a decrease in imports of £89,404 and a decrease in exports of £821,516. The value of imported machinery was £3,984,669, and of exports £20,417,462; increases of £986,015 and £1,352,224 respectively. Imports of new ships amounted to £56,796 and exports to £3,703,481, an increase in imports of £39,063 and a decrease in exports of £3,349,499. Taking the month of August only, during which the strikes had a considerable effect on trade, all the exports except those of machinery were materially lower than in August, 1910. Iron and steel fell by £177,452; other metals dropped £75,735; electrical goods fell £400,960; while the value of ships declined by £1,713,224. In imports for August, iron and steel show an increase of £104,314; other metals rose by £3,763, and machinery by £29,339, but electrical goods dropped £43,378 and new ships £1,793.

Electric Steel.

An article entitled "Electric Steel: Its Position and Requirements," which appeared in the *Times* of the 9th August, has given rise to a prolonged and decidedly interesting controversy in the columns of that newspaper, from which it would appear that the views of the writer of the article are widely shared by steel users in England, and to some extent in other countries too.

One correspondent points out that a power consumption of 540 kw per ton in a two and a half ton Héroult furnace, would mean the extraordinarily high furnace efficiency of 79.25, but would only effect the melting of scrap, and not the making of steel or the killing of the metal. Another questions the lasting or fatigue properties of electric steel giving the same analysis and having the same tensile strength and elongation as basic hearth steel. But, without going into the various points discussed, it may be stated that unless and until much greater uniformity of quality and properties than are at present obtainable in electric steel can be secured there is little prospect of the new product making much headway here, especially among tool makers.

A flood of light is thrown on the subject by the remark of a certain German file manufacturer, who after using electric steel for a time handed an order to a Sheffield steel traveller, saying "Thank goodness I am going to have a steel again from which I can get more than two files alike out of a bar."

On the other hand, English steel-makers recognize the possibilities of electric steel; and although the number of furnaces now running is small, it is slowly increasing. The "Electro-Flex Steel Company" is putting down an initial installation of two one-ton Stassano arc furnaces at Dunston-on-Tyne for the production of steel by exclusively electrical methods.

A New Process of Electroplating

Plant has been erected in London for the demonstration of improved methods of electroplating, adopted by Mr. Q. Marino, which are apparently applicable to the deposition of all the ordinary metals and alloys on almost any metal and on non-metallic substances. It is noteworthy that aluminium

receives perfectly adherent coatings either of pure metals or alloys after a special preparatory treatment of brief duration—while for protecting iron with zinc the deposit obtained by the new process has obvious points of superiority over the coating obtained by the ordinary galvanizing. Tin-plate of very superior quality is said to be produced more economically than by the method in general use.

After certain chemical treatment which makes the surface conducting, without employing varnish and plumbago or other similar means, china and glass are coated with a film of adherent metal which can be polished, burnished and subjected to heat treatment.

Market Prices.

September, 1911.

Copper has remained fairly steady on the whole. On August 30th it was £56, and remained at this price till the 8th, then fell off, touching £54.7.0 by the 20th, but since recovered to £55 at which it closed.

Tin has been as erratic as usual. Starting at £192.10.0, it fell rapidly to £186.0.0 by the 5th, to £179.0.0 by the 14th, £176 on the 19th, and £171 on the 25th, closing, however, at £179.10.0, a recovery of nearly £9 in 5 days.

Lead fairly steady, and strong, opened at £14.12.6, rose to £15 by the 14th, and continued rising to the close, at £15.10.0.

Haematite has been flat, with a rise towards the end of the month, opened at 61/-, rose to 62/3 and remained at this price till the 21st, closing at 61/9.

Scotch Pig has been rather lower, opening at 53/-, dropping to 52/9 by the 14th, and closing at 52/6.

Cleveland has also been depressed, opening at 46/10 by the 14th, it ultimately closed at 46/6.

India Rubber (Para fine) has fallen off, opening at 4/9½, it was 4/½ on the 14th and closes 4/8.

	£ s. d.
Alum, lumps, loose, per ton	5.12. 6
Antimony, black sulphide powder, per ton	22. 0. 0
Borax, British Refined Crystal, per ton	16. 0. 0
Copper Ore, 10 to 25 per cent per unit 9/1½ to	9. 7½
Copper Sulphate, per ton	20. 0. 0
Carbolic Acid, liquid, 97/99 per cent, per gal	1. 7
Creosote, ordinary good liquid, per gal	2½
Camphor, 1-oz. tablets	1. 9½
Caustic Soda Ash, 48 per cent ordinary, per ton	5.10. 0
Hydrochloric Acid, per cwt	5. 0
Mica, small slab, original cases, per lb.	6d. to 2. 0
Naptha, solvent, 90 per cent. 160°C., per gal	0. 8
Petroleum, Russian Spot	0. 5½
Quicksilver, per bottle	8.12. 6
Sal Ammoniac, lump, firsts, delivered U.K., per ton	42. 0. 0
Sulphate of Ammonia, f. o. b. Liverpool, per ton	13.17. 6
Sulphur recovered, per ton	5. 0. 0
Shellac, standard T. N. Orange Spots, per cwt	4. 4. 0
Tin Ore, 70 per cent, per ton	£112 to 114. 0. 0
Zinc, Vielle Montague	32.10. 0
Platinum	9. 5. 0

The differences over the month noted are:

Rise	Amount.
Copper Sulphate, ton	0.15. 0
Carbolic Acid, gal.	1
Platinum	12. 6
Fall	
Copper Ore, unit	4½
Sulphate of Ammonia, ton	7. 6
Tin Ore, ton	8. 0. 0

The Braden Copper Company, operating in Chile, has recently completed the first unit of its new concentrating plant. The complete mill will have a capacity of 3000 tons per day, and the first unit of 1000 tons capacity is probably now in operation. The ore is low grade, averaging 2.7 per cent copper.

Synopsis of Metallurgical and Chemical Literature.

Gold and Silver.

Economics of Tube Milling.—The tube mill was first introduced into the metallurgical industry by Dr. Diehl in Kalgoorlie in 1892, but it was not until 1905 that the machine assumed any importance in metallurgy. About the latter year it was extensively introduced on the Rand, where it has continued to grow in favor. The latest reports from the Rand show that 218 tube mills were in use there in June 1911, compared with 178 in the same month in 1910. In spite of the wide use of these machines the most efficient manner of their operation has not been a matter of settled conviction, and consequently the tests made by H. Stanbish Ball at McGill University, Montreal, Canada, are very interesting and valuable. The full results of Mr. Ball's work have been printed in *Bulletin* No. 83, Inst. of Min. and Met. The paper is replete with tabulations and curves which cannot be reproduced here, but the following gives a concise summary of the main points established.

Four series of investigations were carried out to determine the most efficient feed, moisture, pebble load and speed. In each series one of these factors was varied while the other three remained constant. Stadler's proposal for estimating crushing efficiency, based on Kick's law, was adopted in the calculations, and standard screen of the I. M. M. were used in the grading analyses. A general summary of all tests is given in the accompanying table.

General Summary of Tests.

Test.	Pebble Load.	Pebble Vol.	Peripheral Rev. per Min.	Speed Min.	Moisture. %	Feed Tons per 24 Hrs.	H.P.	Net Output Work —120 Mesh Done per Unit.	Grade" per (Tons 24 Hrs.)	Rel. Mech. Eff. per H.P.
Feed tests.										
P	1,200	0.5	41	369	38.0	7.2	6.3	3.89	2.27	4.45
C	1,200	0.5	41	369	38.0	9.6	5.0	3.36	2.34	6.45
B	1,200	0.5	41	369	38.0	12.6	5.1	2.82	2.95	7.02
O	1,200	0.5	41	369	38.0	14.4	5.6	2.91	3.31	7.48
*A	1,200	0.5	41	369	38.0	18.6	5.8	2.58	3.96	8.29
R	1,200	0.5	41	369	38.0	23.0	6.6	2.09	3.79	7.28
Moisture tests.										
D	1,200	0.5	41	369	30.0	12.6	5.7	2.99	2.93	6.61
*B	1,200	0.5	41	369	37.7	12.6	5.0	2.82	2.95	7.11
E	1,200	0.5	41	369	50.0	12.6	6.2	3.05	2.95	6.16
F	1,200	0.5	41	369	58.0	12.6	6.0	2.84	2.97	5.96
Pebble load tests.										
*G	900	0.37	41	369	38.0	12.6	4.9	2.75	2.65	7.07
B	1,200	0.50	41	369	38.0	12.6	5.1	2.82	2.78	7.02
H	1,500	0.63	41	369	38.0	12.6	6.9	3.31	3.40	6.04
Speed tests.										
T	1,200	0.5	33	297	38.0	7.2	6.8	4.19	2.45	4.43
*O	1,200	0.5	37	333	38.0	7.2	5.1	4.17	2.59	5.89
P	1,200	0.5	41	369	38.0	7.2	6.3	3.89	2.27	4.45
S	1,200	0.5	46	414	38.0	7.2	7.5	4.03	2.48	3.87

*Most efficient test of series.

Series I.—Tests to determine the most efficient rate of feed. From the results obtained it was found that a feed of 18 tons per 24 hours is the most efficient one for the laboratory mill, any larger feed apparently causing the crushing efficiency as well as the relative mechanical efficiency to decrease. (The tube used in the experiments had outside dimensions as follows: 4 ft. 8 in. by 3 ft. 5 in. The lining was silex brick set in cement.)

This is a most interesting point, as a critical feed is indicated, above which it seems probable that the fine grinding properties of the mill decrease in proportion approximately to the increase of feed. The theory of a critical feed may have an important bearing in modern practice, as it would seem that more efficient working, with a greater output of slimes would be obtained if a smaller feed were used. In South Africa the tendency at present is to increase the feed in the 5½ by 22 ft. mills, and although they are now feeding at the rate of 400 tons per 24 hours, the belief is held that even heavier feeds would result in increased grinding. The results of the present investigation show the danger of carrying this too far. Comparing the South African mills with the laboratory machine, it appears that the former would give better results if fed with about 300 tons per day instead of 400.

Series II.—Tests to determine the most efficient percentage of moisture in the feed.

The outstanding feature of these tests is the great efficiency of 37.7 per cent moisture over all others. This fact coincides with tests carried on in other parts of the world. In those conducted by Mr. Walter Neal the ideal dilution was determined to be in the neighborhood of 39 per cent moisture, although he believes that the critical point will vary with the size and nature of the mill. The results of the author's tests tend to disprove this belief. Sherrod found that the mill efficiency rose when the moisture was present to the extent of 40 to 45 per cent. G. O. Smart found the best moisture to be 38.5 per cent, while H. W. Fox discovered that, taking the power consumption in conjunction with the fine grinding, the most efficient moisture would be 39.6 per cent. Prof. J. W. Bell suggests that the critical moisture point depends on the percentage of voids in the sand, and that in all probability when the percentage of moisture is numerically equal to the percentage of voids in the sands inside the mill, the moisture is critical.

In connection with the tests on moisture, some novel work was done. From the first series of tests, on the variation in feed, it had been determined that the mill assumed uniform conditions after a period of about 35 minutes from the start of the test. This fact gave rise to the following idea, viz., if the mill was run for a given time at a certain fixed moisture until the conditions had become uniform and the moisture was then suddenly raised to another fixed point without stopping the mill, but allowing it to run for the same period, would not one be justified in assuming that the samples and power readings taken during the transition period of the mill from one state of normal conditions to the next, are representative of its product caused by the amount of moisture in it at that exact moment at which the sample was taken.

To test this theory, samples and power readings were taken at intervals of five minutes throughout three tests and were most satisfactory. It was found that whereas the mill took 30 minutes to adjust itself after the moisture had been changed from 30 to 50 per cent, it took only 12 minutes to assume uniform conditions from a change of 50 to 58 per cent, thus indicating that the time of change was proportional to the amount of change of moisture. With this fact so clearly demonstrated it was a simple matter to calculate what percentage of moisture the mill contained at the precise moment when a transition sample was taken, e. g., since the first transition period was 30 minutes and five samples were taken, the corresponding percentages of moisture would be 33 1/3 per cent, 36 2/3 per cent, 40 per cent, 43 1/3 per cent, 46 2/3 per cent. In the curves plotted by the author, the points obtained for these transition samples coincided with remarkable regularity with the curve drawn through the main points obtained for the actual tests. Hence the adoption of this method was more than justified, as it saved considerable time, material and labor, and gave a number of moisture efficiencies which otherwise would have been impossible to obtain.

Series III.—Tests to determine the most efficient pebble load. The result of this test indicated that the most efficient pebble load is about 7/16 of the volume of the mill, whereas, disregarding power, the most effectual crushing load is about 0.6 of the volume which corresponds to present Rand practice. Whether it would pay to sacrifice power and gain fine grinding, of vice versa, would of course depend upon the conditions of the plant.

Series IV.—Tests to determine the most efficient speed of mill. These tests were most interesting as they proved rather conclusively that 37 r. p. m., at a peripheral speed of 333 ft. per minute, was the most efficient, both in relative mechanical efficiency and grinding efficiency; thus demonstrating that at that speed the grinding was decidedly better and the power required less. If this speed is exceeded or decreased the crushing effect of the pebbles is much reduced, while the power required is increased. The efficient speed determined by the tests, viz., 37 r. p. m., compares with other determinations as follows: Davidson, 34; White, 39; Richards, 38.

General Conclusions.—In connection with the curves obtained from the different tests it is the author's belief that there is a possibility of deducing from them the probable results that would be derived by running the mill under different conditions. It seems that if one complete series of tests was run with a constant feed, the other factors being varied and the respective curves obtained, one would be able to deduce from them all the necessary information regarding any feed, provided that one test at that particular feed had been carried out.

Factors in Mexico's Progress.—Twenty years ago the bulk of Mexican silver ore was treated by the Patio process. Attempts were made frequently to introduce the American pan system, but for various reasons it was not successful. The non-adaptability of the ore; its irregular supply and the lack of mechanical knowledge stood in the way of progress. Changing conditions, however, forced new methods into Mexican metallurgy and have made it what it is to-day. In the *Informes y Memorias* of the Mexican Institute of Mining and Metallurgy, VICTOR M. BRASCHI discusses the influence of some of the factors in Mexican progress. He regards the smelter, electricity and the cyanide process as the three most important of these.

The smelter first proved a benefactor in eliminating the long time usually required for recovery of the value from ore. Instead of the thirty days required for treating the torta, the smelter bought and paid for the ore on delivery. This resulted in abandoning to a large degree the Patio process, and most of the ores were diverted to the smelter. To quote Mr. Braschi's words, "The smelter began by helping the Mexican industry, but it was ending by throttling its very life." It had not only established a monopoly in ore treatment, but it had bought many large mines, and was making the lion's share of the profit.

About this time the cyanide industry was firmly established in the old mining district of Guanajuato, and the development of hydroelectric power in Michoacan combined with the cyanide process to point the way to freedom of the miner from the smelting process. Both of these improvements made much ore available that formerly could not stand the freight and treatment charges of the smelter. Not only ore in place, but much that was in the stopes and on the dump was treated at a profit.

It is Mr. Braschi's opinion that a modern school of mines and metallurgy will be the next step in the development of Mexico's wonderful mineral resources, and that chemical engineering also will be a great aid in utilizing many nonmetallic resources.

Continuous Cyaniding at Natividad, Mexico.—In a paper read at the Wilkes-Barre meeting of the American Institute of Mining Engineers, in June, 1911, Mr. HUNTINGTON ADAMS presented the results of continuous agitation in Pachuca tanks at Natividad, Mexico.

The difficulties encountered in intermittent filling and agitating in tall tanks of the Pachuca type are principally those due to the accumulation of pulp in the cone of the tank. Therefore a continuous flow of pulp from tank to tank has been devised and put in operation in several places. In this way the time of agitation of all the pulp is longer than by intermittent filling, and all particles of ore have the best opportunity to come in contact with the circulating solution.

The main point to be observed in continuous flow is that the composition of the pulp as regards ore and solution shall be maintained as uniform as possible. Otherwise a classification will result and in some tanks heavier particles will accumulate while the lighter particles pass on through the system. The only sure method of obtaining a uniform flow through the system is by arranging the overflow connections from tank to tank so that they will sample the overflow of the air lift pipe. This sampling should make a clean cut of the whole thickness of the air lift discharge. Some previous installations have not ob-

served this point, their connections being placed in the sides of the tank.

At the Natividad mine the concentrate is cyanided with the rest of the pulp after being ground very fine. Johnston vanners concentrate the slime overflow from Dorr classifiers, the concentrate being returned to the tube mill and reground until too fine to be caught on the vanners.

The arrangement of tank connections is as follows: A drop of 4 in. is used from tank to tank, and the central air lift tubes are cut down or added to in order to give this drop. Wooden boxes 7 in. wide, 10 in. long and 6 in. deep are fixed against the central tubes with their tops flush with those of the tubes, and 4 in. pipes placed horizontally pass from the bottom of each box to the next tank in series. Bypass pipes fitted with valves, join each pipe connection with the next in the series, and the 4 in. drop from tank to tank is made in the by passes. Since the level of the inflow pipe in each tank is 2 in. below the tops of the air-lift tube and of the overflow box connected with the outflow pipe, but is slightly above the pulp level in the tank, no part of the pulp entering can pass out of the tank without first gravitating to the bottom, and risen through the air-lift tube, thoroughly mixed with the whole content of the tank.

On the tops of the overflow boxes are sliding iron covers which open at right angles to the direction of the overflowing stream. The regulation of the flow from tank to tank is done entirely by means of these covers, and the valves are used only when it is desired to bypass the tanks. The boxes are sufficiently large for the tonnage passing through agitation, so that the covers need be opened only an inch or two. The openings which sample the pulp stream are thus rectangular, with their long axes parallel to the radial overflow at these points; and while theoretically this is not so correct a shape as would be a sector of a circular ring, in practice it has been found to offer no difficulties.

As the last tank in the series discharges to the pulp tank of a Moore filter, where an intermittent feed is important, it is arranged with two pairs of sliding doors on the central air-lift tube so as to permit agitation at various levels. The pulp is drawn off intermittently to the filter through the bottom discharge openings of the tank, either by hydrostatic pressure in the tank if full enough, or by centrifugal pumps. All the tanks retain their bottom discharge connections with the pumps which lift to the filter, so that whenever it is necessary to empty any of them they may be cut out of the series and discharged to the filter.

The advantages of the Natividad installation over those of Grothe and Kuryla seem to be: Greater simplicity of installation; greater accessibility for handling and supervision; no plugging of the pipe connections can occur and no compressed air connections are necessary to free them; no classification is likely to occur as the boxes by which the flow from tank to tank is regulated provide convenient means of accurately sampling the mixed pulp as it overflows the air-lift tube, and the proportion of solution to slime remains the same throughout the whole system.

Physical Chemistry.

The Specific Heat of Solids at Extreme Low Temperatures.—In 1905 Dewar published a series of interesting results on calorimetric measurements carried out at temperatures of solid carbon dioxide and liquid air. (See *Chemical News*, 92, 181). Recently Herman Barschall reconstructed the Dewar apparatus so as to ensure greater accuracy in the determinations. It consists essentially of three Dewar vessels, one inside of the other. Details and cuts are given in *Zeitschrift für Elektrochemie* of May 1. Only a few grams of the substance, whose specific heat is to be measured, are required to obtain very accurate results. The author found the following values for the specific heat at 183 C; lead, 0.0294; silver, 0.0402; cadmium, 0.0503; sulphur, 0.119; mercury, 0.0316; bromine, 0.073; mercuric iodide, 0.0375; lead iodide, 0.0377; lead bromide, 0.0463.

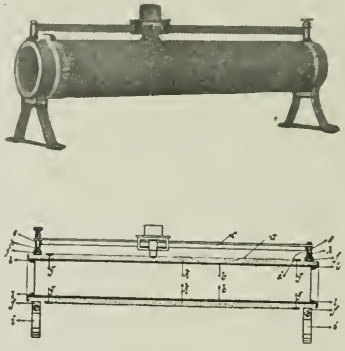
The Heat of Evaporation of Oxygen.—H. Barschall has made a new determination of the heat of evaporation of oxygen (*Zeitschrift f. Elektrochemie*, May 1.) The average of six experiments is 51.30 calories per gram of oxygen boiling at ordinary atmospheric pressure. This value approaches that of Alt 50.97 rather closely, but is considerably smaller than the figure 61.0, published by Shearer in 1903.

Improved Laboratory Rheostat.—A German firm has recently put on the market a new porcelain tube resistance unit particularly adapted for laboratory purposes. As is shown in the accompanying figure the porcelain tube k is lined inside with a metal tube, the contact between metal and porcelain being the best possible. The resistance wire, wound on the outside of the porcelain tube, can be kept at temperatures approaching red heat without serious effects on the tube. The rheostats may be had in lengths of 10 to 100 centimeters. (*Zeitschrift für Elektrochemie*, June 1.)

Chemical Engineering.

Wislicenus Smokestack for Fume Dilution.—Investigation has pretty clearly established the maximum quantity of noxious gas which vegetation can successfully withstand. For SO_2 in relation to sensitive trees such as the spruce, this density is one part of gas in 500,000 parts of air. This is a small percentage, but is not much less than exists in some of our industrial towns other than smelting centers. In the latter the percentage rises much higher, and the necessity of diluting the gases in some manner becomes more important. To this end Dr. Wislicenus, of Germany, has invented a smokestack designed to aid in rapid dilution of discharged gases. The full description appears in the *Zeitschrift* Ver. Deutscher Ing., June 10.

If stack gases are to be diluted so that they will not be injurious they must mix with large volumes of air at the time of their discharge from the stack, as it is well known that from



RHEOSTAT.

high stacks especially the smoke will persist in density for long distances. It is Dr. Wislicenus' idea that if the upper part of the stack were built in lattice form, with a large number of apertures for escape of the gases, a more rapid dilution would occur. Eddies would be formed which are most favorable to dilution. Consequently he has experimented with models and on a large scale, and has designed several forms of lattice structure which break up the dense smoke and mix it immediately with larger volumes of air than would be the case if the dense smoke issued in the ordinary way. One form of lattice consists simply of radial openings, another of tangential openings, and still another with projecting and enlarged vents for steel stacks. Correct construction is necessary to obtain the best effect.

Removing Oil from Return Feed Water.—The presence of any considerable quantity of oil in return feed water is injurious to boiler tubes and plates, reduces the evaporative efficiency and aids in the deposition of scale. Various schemes have been proposed to overcome this, such as submitting the water to an electrical discharge between iron electrodes for the purpose of

coagulating the oil. This scheme, however, has been found to cause deterioration by corrosion, probably due to electrolysis.

A successful plan is described by C. E. CROCKER in the July 1911 *Journal* of the Chamber of Mines of Western Australia. It consists in treating the oily water with alum and common soda in the proportions of 0.35 lb. of alum and 0.30 lb. of soda per thousand gallons of water, for water containing 3.5 grains of oil per gallon. The quantities of chemicals used are, of course, proportioned to the quantity of oil contained in the water. The apparatus should have sufficient capacity to allow the chemicals ample time to act. In the plant referred to the capacity is 50,000 gallons per 24 hours, and the tanks and filters are arranged so that five hours are required for the water to pass through.

The alum and soda are mixed in tanks and allowed to drip into the incoming oily water. The precipitated water is filtered through coke, charcoal, shavings, or any open porous bed. The filters are cleaned as often as necessary. The cost is about 2.3 cents per 1,000 gallons. The plant has been in operation for two years, and the boilers require less cleaning than formerly.

Ozone.

Ozone Generators.—Prof. C. HARRIES, of Kiel University, reports in the *Zeitschrift für Elektrochemie* of August 1, results of experiments made with a Siemens and Halske ozone generator. When using pure oxygen a maximum output of 18 per cent to 19 per cent ozone was obtained at a potential of 7400 volts alternating current, and frequency of 100 at 20° C. The ten glass tubes of the generator were mounted in a wooden instead of the usual metallic frame, and as a consequence an appreciable increase in efficiency was obtained. For parallel connected tubes the most favorable gas velocity was 60 liters per hour and for series connected, 8.6 liters per hour.

The efficiency of the generator increases after repeated using. When air or pure nitrogen was passed through the tubes an intensive blue luminescence appeared which was absent, however, when pure oxygen was used. This blue luminescence is not to be confused with the blue color of highly concentrated ozone gas; it is not due to the presence of traces of oxides of nitrogen, as Warburg and Leithaeuser had assumed, but is the inherent color of the gas. Ozone will react fairly readily with unsaturated hydrocarbons resulting in addition products, such as ethylen ozonide and propylenoxozonide, that are highly explosive. The study of these compounds lead to the assumption that ozone gas contains quadri-atomic oxygen besides the tri-atomic. By careful fractional distillation and analysis the author succeeded in verifying his assumption. The formula of oxozone is either O_4 or O_5.

GIDEON LECHNER has investigated the efficiency of a Berthelot ozonizer. With direct current he finds that the ozone output is directly proportional to the frequency of the interrupter. Increasing the closed-circuit interval of the interrupter increases the mean current intensity and at the same time almost proportionately the ozone output. An alternating current of 42 cycles is more efficient than a direct current of frequency 42 owing to the flatter form of the wave and consequent greater mean current intensity. (*Zeitschrift für Elektrochemie*, June 1.)

Minimum Potential of Silent Discharge.—Prof. E. H. RIESENFELD, of Stockholm, investigated the behavior of a large number of gases when passed through a generator tube of the Jannasch type. The results are recorded in the *Zeitschrift für Elektrochemie*, September 1. The author found that the relative minimum potential necessary to bring about the silent discharge is well defined for every gas and independent of the size or shape of the generator tube, of the velocity of the gas and of the frequency of the current. The relative potential is defined as the potential expressed in terms of the oxygen potential taken as unity. The minimum potential of a gas is proportional to its continuous potential or potential required to effect a steady continuous discharge. The author's values for the relative minimum potential are: Hydrogen, 0.77; nitrous oxide, 0.85; ammonia, 0.97; oxygen, 1.00; carbon dioxide, 1.03; carbon monoxide, 1.032; acetylene, 1.11; ethylene, 1.11; nitrogen, 1.16; hydrogen sulphide, 1.27; hydrochloric acid gas, 1.27; chlorine, 2.42. The luminescence color of each gas is very distinct and different from that of any other. Hydrogen shows a pale blue, oxygen a deep greenish blue, nitrogen a decided purple, and chlorine a very intense green luminescence. There does not seem to be any direct relation between the phenomena of spark discharge and those of the silent discharge.

Electrolytic Production of Ozone.—The electrolytic production of ozone was recently investigated by E. H. ARCHIBALD and H. VON WARTENBERG (see *Zeitschrift f r Elektrochemie*, September 15.) Dilute solutions of sulphuric acid were electrolyzed with direct current and with alternating current superimposed. Owing to the depolarizing effect of the alternating current the output of ozone was greatly increased over that obtained with direct current alone. In some cases the increase was three hundred fold. Platinum electrodes were used. By keeping the intensity of the direct current constant and increasing the alternating current step by step the amount of ozone rose steadily. The maximum output was obtained at about 6 amperes, alternating current, for all values of direct current between 0.25 and 1.00 amperes. An increase in the frequency of the current was accompanied by an increase in the ozone output. The principle involved is commercially utilized by the Norddeutsche Affinerie in their gold-silver parting process (Wohlwill process.) Formerly, when using merely direct current, a considerable part of the chlorine escaped the bath as gas instead of combining with the silver; this loss, however, is now entirely avoided by superimposing an alternating current. (See our vol. viii, p. 82).

Recent Metallurgical Patents.

Iron and Steel.

Nodulizing and Desulphurizing Kilns.—For the purpose of desulphurizing and nodulizing iron ores in one rotary furnace, Mr. DAVID BAKER, of Philadelphia, Pa., has patented a furnace. The invention consists primarily in constructing the furnace in two sections, in one of which desulphurizing takes place, and in the other nodulizing or sintering. Invention also lies in combining with a rotary furnace a scraper or cleaning device to prevent the accumulation of rings in the hottest part.

The desulphurizing end of the kiln consists of a cylindrical furnace, having a slight pitch so that the ore will travel through the furnace as it revolves. The nodulizing portion consists of an extension of the kiln in the form of a truncated cone, which is relatively shorter than the cylindrical portion. The smaller end of the truncated cone encircles the end of the cylindrical portion so as to receive the ore discharged therefrom. Different speeds are imparted to the two parts of the furnace, so that the desulphurizing end runs slower than the nodulizing portion, there being less heat and longer time required for desulphurizing and higher heat and shorter time for nodulizing. A satisfactory speed is 1 r.p.m. for the cylindrical portion and 4 r.p.m. for the truncated cone. The scraper is a bar sustained in such a position that it is in close proximity to the side of the nodulizing kiln. It is cooled. (1,004,340, Sept. 26, 1911.)

Still another means for preventing the formation of rings in sintering furnaces has been patented by Mr. HARTLEY C. WOLLE, of Westmont Borough, Pa. It consists in embedding cooling pipes in the interior walls of the kiln. The object of the cooling pipes is to so cool the furnace in their vicinity that a continuous ring can never form. Sintered accumulations may form, but they will be in the form of an arch without a keystone, and in the natural revolutions of the furnace they will tend to fall off. One or more of these cooling pipes may be embedded in the furnace wall, thus causing one or more breaks in the continuity of any ring which might form. The coolers are cooled by circulating air or water through them. (1,005,396, Oct. 10, 1911.)

A similar effect is obtained by Lewis P. Ross, of Standish, N. Y., by supporting a T-bar in the furnace to detach material adhering in the form of a ring or annulus. The bar is suspended so that it is not rigid, and is cooled by water pipes running its length. (1,004,085, Sept. 26, 1911.)

Electric Steel Furnace.—Among the more recent combination furnaces for electric steel refining is that of Hans Nathusius, which has been employed in commercial tests at Friedrichshütte, Germany. (our vol. viii, p. 490 and 546). It combines heating from the top by means of arcs from vertical electrodes with heating from the bottom by means of

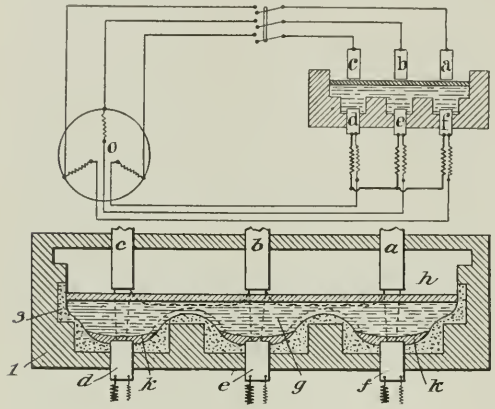


FIG. 1. ELECTRIC STEEL FURNACE.

metallic electrodes embedded in the bottom and separated from the furnace charge by a socalled conductor of the second class, as in the Roechling-Rodenhauser furnace. The object is to heat from the top when the refining depends on a very high temperature of the slag and to heat from the bottom at other periods when this may seem desirable. To increase the heating effect from the bottom, the arrangement shown in Fig. 1 is employed. With three-phase currents, as shown in the upper diagram, this is accomplished "by dissolving the nodal point of the machine and connecting the three inner ends which otherwise form the neutral point, to the bottom electrodes, while the outer ends are connected in the usual manner to the three surface electrodes. By means of this connection the point is transferred into the metal bath itself."

"This heating action in the metal bath, or the increase of heating action due to resistance heating in the metal bath or in the hearth role, by increasing the current, can also be advantageously obtained by connecting in front of the bottom electrodes a transformer which transforms the current from the main furnace source of current to a very high strength of current with a correspondingly small pressure, so that in addition to the primary current a current of any desired strength is supplied by the bottom electrodes direct to the metal bath or to the hearth sole." The lower diagram shows a lining *k* formed of a conductor of the second class disposed around and over the inner extremities of the bottom electrodes *d e f*. The upper electrodes are *a b c*, the hearth is *g* and the slag layer *h*. The refractory lining 3, which is rammed between the bottom electrodes and which, when the furnace is in operation, is traversed by the current, consists of ordinary burned dolomite, being mixed with tar as binding agent. This is the so-called "black mass" which serves in a similar manner for the preparation of basic open-hearths.

The transformer is best built directly under the electric furnace as a part of it to permit easy bottom connection. (1,000,838, August 15, 1911.)

Gold and Silver.

Improved Method of Forming a Halogen Cyanide.—In cyanidation it has been found advisable in some cases to make use of bromocyanide, or similar halogen compound. The usual method of forming this compound has been by mixing sulphuric acid, potassium cyanide and an alkali bromide and bromate. The mixture usually is made only when it is needed, and the procedure is one requiring care. It is the aim of Dr. Hans Foersterling, of the Roessler & Hasslacher Chemical Company, of Perth Amboy, N. J., to provide a better means of forming halogen cyanides by melting an alkali halogen with an alkali cyanide in the proportion of one molecule each and permitting them to solidify. No decomposition takes place in this process and the double salt is used instead of making the mixture above referred to. In order to liberate bromocyanide, for example, from a mixture of sodium bromide and sodium cyanide, an oxidizing agent such as an acid solution of hydrogen peroxide is used. The reaction follows:

$$NaBr + NaCN + H_2O_2 + H_2SO_4 = BrCN + Na_2SO_4 + 2H_2O.$$

By this method it is possible to provide cyanide mills with a correctly proportioned mixture ready for use when dissolved and mixed with an oxidizer. (999,215, Aug. 1, 1911.)

Precipitation of Gold and Silver Prior to Filtration.—The use of aluminium as a precipitant for gold and silver from cyanide solutions has been proposed before, the aluminium being merely a substitute for zinc. Mr. James E. Porter, of Syracuse, N. Y., proposes to use aluminium or its suitable alloys to precipitate gold and silver from cyanide solutions without filtering the solution from the ore pulp.

The ore is finely ground and agitated for a short time with caustic soda to neutralize cyanides and to give the ore a protective alkalinity. Cyanide is then added and the mixture agitated, preferably with aeration and by the use of so-called silica sponge, until all the gold and silver are in solution. If the agitation and aeration are well done the inventor states that sulphides can be effectively treated in the raw state.

Strips of aluminium are then immersed in the plup while the agitation is continued; the immersion being accomplshed in any suitable manner. The dissolved metals will precipitate on the aluminium, which is periodically removed and cleaned by wiping or scraping. As long as any precipitate forms the operation is repeated. The precipitation of gold liberates potassium cyanide and cyanogen; some of the aluminium goes into solution as potassium aluminate with liberation of hydrogen, and the cyanogen and hydrogen combine with the alkali present to form potassium cyanide, thus regenerating the cyanide combined with the precious metals. (1,002,446, Sept. 5, 1911.)

Direct Cyanidation of Sulphides.—In cyaniding sulphide ores containing silver, or gold and silver, with the latter usually predominating, silver sulphide goes into solution in potassium cyanide, forming potassium silver cyanide and potassium sulphide. The solvent action is, however, limited to the state of equilibrium which soon occurs, and after that point has been reached the reaction is reversible and silver again goes into solution. This phase will occur before all the silver has been dissolved. It is the object of an invention of James E. Porter, of Syracuse, N. Y., to provide means whereby the sulphide of potassium formed may be oxidized to thiosulphate, thereby avoiding reprecipitation and aiding in the extraction.

Mr. Porter shows that some form of oxidation is necessary to accomplish this, and proposes to obtain it by energetic and prolonged agitation with air. The conditions of success which he imposes are (1) fine grinding of the ore; (2) suspension of the ore in cyanide solution; (3) thorough permeation of the mass by minute bubbles of air which will remain in the liquid for some time and not rise quickly to its surface, and (4) operation at a temperature of about 180 deg. Fahr. to 190 deg. Fahr. Under these conditions he claims that it will be impossible for soluble sulphides to remain as such, their oxidation following

closely upon their formation, and equilibrium between dissolved silver and dissolved sulphide cannot exist. In this process the inventor prefers to use silica sponge as the medium for atomizing the air. (1,002,447, Sept. 5, 1911.)

Impact Filter.—A filter utilizing centrifugal force in the separation of cyanide solution and ore pulp has been patented by THOMAS H. NEIKIRK, of Midas, Nev. As shown in Fig. 2 the filter consists of a revoluble drum in the shape of a truncated cone, suitably mounted on a hollow shaft and driven from a pulley at 4. Within the drum is a filtering medium against which the incoming pulp is thrown from a number of radial members mounted on the hollow shaft. The filter is designed to work continuously as regards the formation and discharge of cake, and with the minimum of attendance.

The operation of the machine is as follows: Slime, barren solution or wash water are introduced into the filter through the pipes 18, 19 and 20, respectively, each being properly controlled by valves as shown. When the valve on the slime line is open slime enters from pipe 18 to the hollow shaft 3, and is distributed by centrifugal force through connecting radial arms

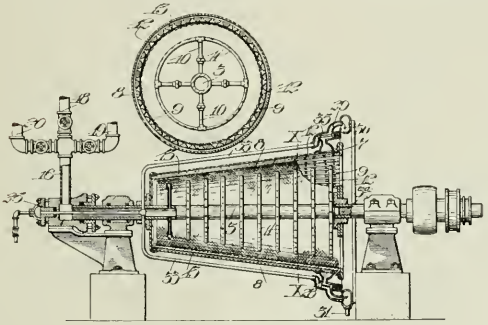


FIG. 2. IMPACT FILTER.

10 to perforated rings 11. The pulp issues from the ring perforations and is forced against the surrounding filtering medium 9, which is supported on a grooved surface 8. The liquid is thus separated from the solids and passes by perforations 12 into an annular channel 13, from which it is eventually discharged into an annular collector 28, which is fastened to the outside of the shell at its larger end. From this collector the solution is finally directed into a drain pipe 31, from which it is conducted to the zinc boxes. The solids accumulate on the filter, and when a cake of sufficient thickness is formed, the valve controlling the incoming slime is closed and barren solution and wash water introduced in a similar manner. The cake having been washed, it is now dislodged from the filter surface by means of a fluid introduced under pressure through the pipes 26 and 33, which connect with the annular channel beneath the filter. Automatic valves 35 prevent the pressure water from finding its way into the collectors which were used for discharging the filtered gold-cyanide solution. The pressure water and filtered pulp thus mix in the cone and are discharged at the lower end of the drum, after which the cycle is repeated.

The impact of the slime pulp against the filtering medium is designed to increase the efficiency and capacity per unit of filtering area. Another important feature is the relative size of the machine to the common filtering apparatus now used. (1,000,540, Aug. 15, 1911.)

Zinc, Lead and Copper

Zinc Furnace.—In the accompanying illustration, Fig. 3, is shown a cross-sectional view of a furnace for the distillation of zinc ores, designed and patented by FRANK LAURENT CLERC, of Boulder, Colo. The novelty of the invention lies in the means used for applying heat to the charge of zinc oxide and reducing substance, and in the proposed manner of introducing the charge. The furnace consists of a retort *a* and heating chamber *b*. The retort has a port *d* for the exit of zinc vapors, and a feeding spout *e* for supplying the charge to be smelted. The bottom of the retort consists of a body of molten metal *f*, such as copper or an alloy of copper and zinc, or a

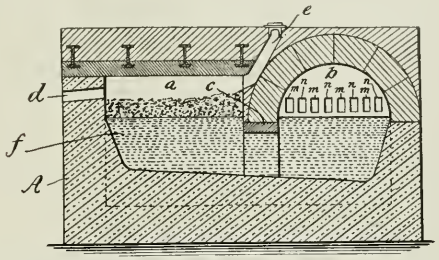


FIG. 3. ZINC FURNACE.

ternary compound which has a melting point higher than the temperature at which zinc is reduced from its oxide by carbon, and which has a high heat conductivity. This body of metal is kept in a molten condition by hot gases from a suitable source of combustion, and the heat is thus conducted through the molten metal to the charge of ore and fuel as shown on the hearth. The reduced zinc vapors pass out through the port *d* and are condensed in a suitable manner. The accompanying carbon monoxide can be collected and burned for any useful purpose. (1,002,037, Aug. 29, 1911.)

Precipitating Copper from Solutions.—The usual manner of recovering copper from dilute solutions such as mine water or water flowing over old copper dumps, is to precipitate the red metal on scrap iron. As is well known, the quantity of iron required is several times that indicated by theory, and the precipitate is not high grade. Mr. E. E. SLAUGHTER, of Clifton, Ariz., has patented an improvement in this method, having assigned the same to W. L. Austin, of Riverside, Calif. Mr. Slaughter prefers to precipitate copper by using a mixture of iron and carbon in the form of small pieces of coke. He thus gains the advantage of an electrolytic action, and states that the precipitated copper is free from basic iron salts and of higher grade than that obtained by precipitation on iron alone. (1,001,466, Aug. 22, 1911.)

Refining and Desilverizing Lead.—The crystallization process used in the refining of lead bullion carrying gold and silver are based on the fact that when such bullion is melted and cooled down to nearly the solidifying point and stirred by hand or otherwise crystals of lead form which are freer from silver and other impurities than the liquid which permeates the mass of crystals. It is customary in this process to draw off the fluid from the crystallized lead, and repeat the process. Usually twelve treatments are required to give commercial lead as one product and a high-grade bullion suitable for cupeling as the other.

Mr. GEORGE P. HULST, of Omaha, Neb., has patented means for handling the semicrystallized mass in a press, by means of which the liquid is squeezed from the crystals, rendering the latter drier and concentrating the impurities in the liquid. By this procedure the number of crystallizations is reduced to six, as compared with twelve as usual. (1,001,525, Aug. 22.)

Cyanidation of Silver Ores at Cobalt.—At the Nova Scotia mill in the Cobalt, Canada, district cyanidation is in use, as noted in the *Mining Magazine* for September, 1911. The ore is crushed in stamps in cyanide solution, and concentrated without classification on Deister No. 2 tables to remove the metallics and rich sulphides. The table tailings are classified, reground in tube mills, thickened in Dorr thickeners, agitated in Trent agitators, and filtered by the Moore process. Precipitation of silver is by the Merrill zinc dust method.

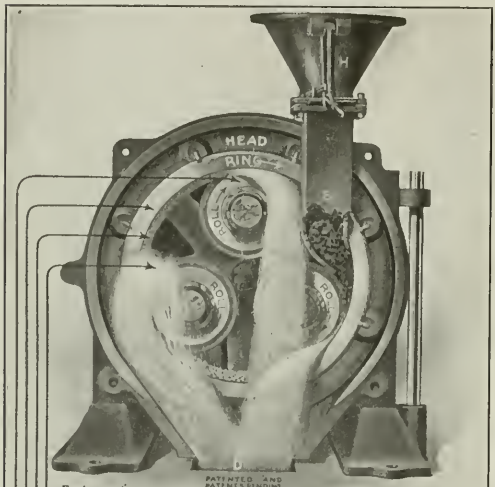
Combination of Ring Roll Mills and Tube Mills in the Grinding of Cement Clinker.

One of the most important problems in many branches of metallurgical and chemical engineering is the proper combination of different grinding machines so as to grind a given material at the lowest cost. Under different conditions the best combination will differ, but in many cases it will be found advantageous to subdivide the whole grinding operations into a series of different ones, carried out in different machines, instead of trying to finish the whole job in one machine.

A case in point is the grinding of cement clinker, and in this respect some tests made by Sturtevant mills are of considerable interest.

The usual practice in reducing Portland cement clinker is to grind the clinker in one ring-roll mill to 20 mesh and finish in three tube mills.

The alternative method, which was compared with that just


HEAD
RING
ROLL
ROLL
ROLL
H
D
PATENTED AND PATENTS PENDING
Feed enters hopper at "H."
Spout "S" delivers it at center of concave revolving ring where it is strongly held by centrifugal force until crushed off by the Rolls, discharging at "D."
Ground rock crushed off of both sides of Ring
Thick layer of centrifugally held unground rock
Rigid revolving Ring drives all three Rolls on 1" layer of centrifugally held material
Spring pressed Rolls driven by centrifugally held material.

RING ROLL MILL.

mentioned, is to grind the clinker in two ring-roll mills to 80 mesh and finish in one tube mill to 83 per cent—200 mesh.

In each case the ground product of the ring-roll mill was raised by means of an elevator to Newaygo screen separators to separate the fine material, which then underwent finishing treatment in the tube mill.

The comparison between the two methods extended to the establishment of comparative figures, firstly, for wear and tear of the machinery, and secondly for power consumption. Of course, the following figures are only comparative, using the same hard clinker.

As to the *cost of maintenance*, the wear and tear of the one ring-roll mill grinding 45 bbl. to 20 mesh per hour represents an expense of 0.076 cent per bbl. The wear and tear of the Newaygo separator, handling 45 bbl. per hour, was 0.022 cent per bbl. The wear and tear of the three tube mills was 0.266 cent per bbl. Hence the total wear and tear of the ring-roll mill, separator and the tube mills (not including elevator) was 0.364 cent per bbl.

On the other hand the wear and tear of the two ring-roll mills, grinding 45 bbls. to 80 mesh per hour, was 0.152 cent per bbl. The wear and tear of the two separator screens was 0.044 cent per bbl. The wear and tear of the one tube mill was 0.089 cent per bbl. Hence the total cost of wear and tear of the two ring-roll mills, separators and the one tube mill was 0.285 cent per bbl. This method, compared with the first one, exhibited a saving of 0.08 cent per bbl. in the maintenance cost of the grinding machinery.

As to the *power consumption*, the one ring-roll mill in the first case requires 0.777 hp per bbl., the Newaygo separator 0.022 hp per bbl. and the three tube mills 5.666 hp per bbl. Hence the total power consumption of ring-roll mill, separator and tube mills is 6.465 hp per bbl.

In the second case the two ring-roll mills require 1.555 hp per bbl., the Newaygo separator 0.044 hp per bbl. and the tube mill 1.888 hp per bbl. Hence the total power consumption of ring-roll mills, separator and tube mill was 3.487 hp per bbl. This represents a saving of 2.987 hp per bbl.

Both with respect to maintenance cost and power consumption the second method turned out to be considerably more economical.

In this comparison the elevator has not been considered since its wear and tear as well as power consumption was taken as the same in both cases.

It is clear from the above comparative figures that the decisive item in the results of the tests was the decidedly high economy of the ring-roll mills and for this reason a description of the Sturtevant ring-roll mill to which the above figures relate should be of interest.

The adjoining illustration shows the construction. A heavy steel anvil ring is secured in a head supported by a horizontal shaft. Against the inner face of this ring hammer rolls are elastically pressed with great force and revolved by frictional contact with material on the ring.

Substances to be ground are fed (up to 1¼ in.) to the inner face of the rotating ring which holds them thereon by centrifugal force to be crushed as drawn under the rolls by ring rotations and discharged on both sides thereof.

The face of the ring is concave and the roll faces convex.

The roll mountings are on the massive door that forms one side of the mill casing, and they are swung away from the ring with the door when it is opened. Thus the whole grinding outfit is made accessible by opening the door. The roll shafts are as large as those of the driving wheels of a locomotive, and crushing pressures are greater. One adjusting screw on the outside of the door regulates the roll forces and gives the rolls an absolutely equal pressure. No rock can withstand these pressures.

When at work, the concave of the revolving ring is always covered with a thick layer of material fed thereto. A naked track is never exposed to the roll faces. They cannot damage one another. Rock is crushed down upon itself (between anvil ring and hammer roll) producing a maximum of fines, with least wear.

As there is a constant feed, while the mill is at work, of coarse and partly reduced material, so there is a constant drop of material crushed off to both sides of the ring by the rolls. This escapes, as in all mills of this class, from the bottom of the case, and is taken to a Newaygo screen to remove the finished product *as soon as made*. The tailings, separated by this screen, are returned to the ring (with fresh feed) for further reduction.

Thus the mill is always breaking down coarse rock *upon* tailings, producing a maximum output and forcing the rocks to crush one another.

It should be emphasized again that in the Sturtevant ring-roll mill the ring drives the rolls. They run like those of an automobile's roller bearing. No roll drives are used nor do the rolls support the ring.

The Absorption of Hydrochloric Acid and Other Gases.

By Percy C. Kingsbury.

Practically all of the muriatic acid made in this country is obtained from the decomposition of common salt by means of sulphuric acid or bisulphate of soda.

The hydrochloric acid gas evolved during the reaction is

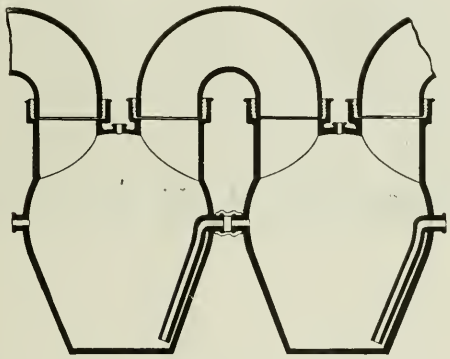


FIG. 1.—STANDARD FORM OF TOURILL.

first cooled and afterward dissolved by water. The cooling of the gas is effected by passing it through large pipes exposed to the air. The pipes are usually of fire clay and on account of the porosity of this material are boiled in tar before being installed.

The plant commonly employed for the absorption consists of a system of stoneware "tourills" connected in series by stoneware piping. A current of water or dilute acid passes through these vessels over the surface of which liquid the cooled gas circulates. One or two small wash towers usually complete the absorption equipment.

For many years little consideration was given to the design of an effective type of tourill for this purpose, chemical manu-

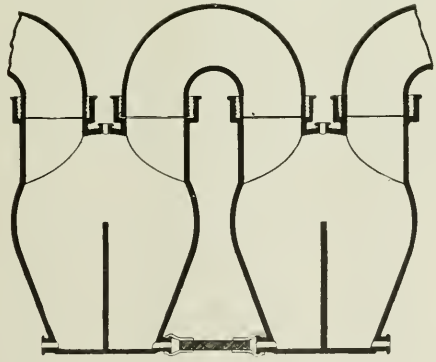


FIG. 2.—ANOTHER STANDARD FORM OF TOURILL.

facturers being content to use the standard form of vessel supplied by the stoneware manufacturers.

This form, illustrated in various modifications in Figs. 1, 2 and 3, has the advantage of being easy to manufacture, and, on account of its symmetrical shape, of resisting the expansion and contraction due to variations in the temperature to which it is subjected.

The principal function of the tourill is to dissipate the heat evolved during absorption as the amount of gas that can be absorbed, or, in other words, the strength of the acid obtainable, decreases very materially with rise in temperature of the absorbing liquid.

An efficient type of tourill should, therefore, offer as large a cooling surface as possible compared with the volume of acid it contains. Those illustrated in Figs. 1, 2 and 3 are for all practical purposes spherical, and therefore from the above point of view of efficiency the least desirable, as of all geometrical forms the sphere has the least surface in relation to its volume.

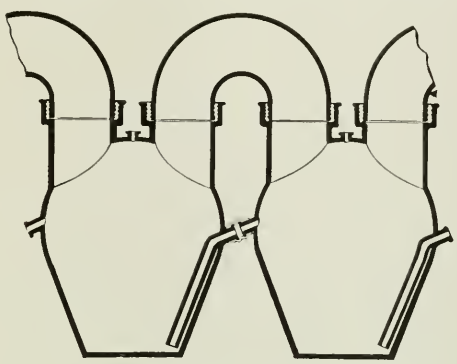


FIG. 3.—ANOTHER STANDARD FORM OF TOURILL.

This tourill has, moreover, several other disadvantages. The acid contained presents a comparatively small surface of contact with the gas to be absorbed, the vessel is difficult to clean out and cannot conveniently be water-cooled. The latter is an important consideration, as artificial cooling is necessary in working for acid of the highest possible strength, especially during the summer months.

These considerations led some acid manufacturers to replace the spherical tourills by vessels of more or less fanciful shapes, most of which proved unsatisfactory on account of the designers not taking into consideration the limitations of the material of which such articles have to be made.

The first practical improvement on the old-fashioned type is the Cellarius tourill illustrated in Fig. 4. Several of the advantages of this vessel are obvious. It can be completely immersed in water, offers a much larger cooling surface in comparison with the volume of acid it contains and also exposes a larger surface of the absorbing liquid to the action of the gas.

By connecting successive tourills in the manner shown in Fig. 5 the vessel can immediately be emptied and flushed out by simply reversing the glass "T" piece. It will be noted that the acid enters the tourill by one of the projecting tubulures and before leaving by the other must flow through twice the entire length of the vessel, passing in a shallow stream over

FIG. 4.—CELLARIUS TOURILL.

the arch at the back of the tourill. This insures an efficient cooling of the liquid and a progressive increase in the strength of the acid as it passes through the vessel.

The tourill withstands variations in temperature very well, as the form of a double arch enables it to take up considerable strain due to expansion and contraction of the material. This form is also advantageous from the point of view of mechanical strength.

One other practical advantage of this vessel is that the acid

connections are under the surface of the liquid, thus preventing the possibility of stoppage of the flow due to the formation of air locks, a matter of frequent annoyance in the old type of vessel.

The many advantages of the Cellarius tourill have received prompt recognition in Europe, where it is rapidly replacing the spherical vessel. Where such substitution has been made it has been found that one Cellarius tourill can replace two or three of the old-fashioned type.

No difficulty is experienced in connecting both kinds of vessel to the same battery. The advantages of the new tourill can, therefore, be realized practically at little expense or disturbance of operation by substituting or adding a few Cellarius tourills at the beginning or end of a series.

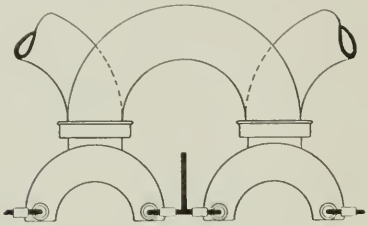


FIG. 5.—CONNECTION OF SUCCESSIVE TOURILLS.

The success of the Cellarius tourill for absorption purposes has led to the design of a modification of this vessel for the cooling and condensation of gases. This pattern is illustrated in Fig. 6 and it will be noted is merely an inversion of the type used for absorption, the advantages of which are retained. It has been employed for the pre-cooling of hydrochloric acid gas before it enters the absorption battery and for the condensation of nitric acid and is a practical and efficient vessel for all purposes where acid gases are to be cooled or condensed to the liquid state.

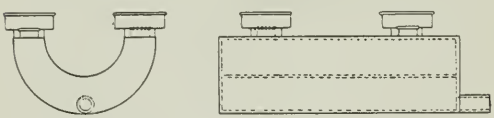


FIG. 6.—MODIFICATION FOR COOLING AND CONDENSATION OF GASES.

The Cellarius tourill was exhibited at the Pittsburgh meeting of the American Institute of Chemical Engineers, December, 1908 (this journal, Vol. 7, page 89), by the Didier-March Company, New York, who are the manufacturers of these vessels in this country.

The Oxygen-Gasoline System of Lead Burning.

While in the old familiar system of "lead burning" hydrogen and air are employed, the new Howard system involves the use of oxygen gas saturated with gasoline. With this new system much more perfect combustion and higher temperatures can be obtained and it is claimed to combine the advantages of superior results with respect to quality of work, with greater economy, greater speed of operation and greater convenience in use.

The adjoining illustration shows the Howard apparatus in operation. It will be seen that there are only two principal parts, namely the gasoline saturator, weighing about 28 lb., and the oxygen cylinder weighing about 68 lb.

To the valve on the oxygen cylinder the oxygen regulator is attached. One outlet of the oxygen regulator is connected by a rubber hose to the blowpipe, while from the other outlet of the oxygen regulator another rubber hose leads to the inlet of the gasoline saturator. The exit gas from the gasoline saturator passes through a hydraulic trap which serves as a safety device in the case of a back-fire, and is then conducted through a rubber hose to the other nipple on the blowpipe. A proper burning mixture is produced at the blowpipe by manipulation of the gas and oxygen cocks.

There is nothing difficult about the operation. In fact, any workman conversant with the hydrogen-air system of lead burning will experience no difficulty in operating the Howard lead burning system.

HOWARD LEAD BURNING APARATUS IN OPERATION.

The advantages of the Howard system are essentially due to the simple apparatus required, as described. As compared with the old hydrogen-air system, the hydrogen generator is dispensed with. The air bellows or pump is dispensed with and consequently the services of a boy are not required. The apparatus is portable and may be carried anywhere, thus doing away with the very troublesome necessity of moving a heavy hydrogen generator and bellows from one job to another.

There is no apparatus to get out of order, involving expense, delays and repairs. No acid or corrosive substance is employed. As neither zinc nor sulphuric acid is used, there is no deleterious matter to be carried through the blowpipe to act injuriously on the lead seam.

The flame produced is absolutely clean and does not blacken nor tarnish lead, brass or copper. This is of special advantage in soldering the two latter metals, and also in repairing old lead work that is thin, as very little shaving is necessary.

A much better, hotter flame is produced than with hydrogen and air and much superior and faster work can be accomplished. The flame is so hot that even heavy lead in wet and cold positions can be burned "in situ" without preheating. A general idea of the comparative speed of lead burning by means of the Howard apparatus and the hydrogen and air method respectively may be gathered from the figures in the following table, in which the approximate foot run burnt per hour is given for various weights of lead sheet.

		Gasoline-Oxygen	Hydrogen-Air
Flat seam	20 lb. lead	20 ft. per hour	15 ft. per hour
" "	12 " "	48 " " "	27 " " "
" "	8 " "	60 " " "	40 " " "
" "	6 " "	108 " " "	90 " " "
Upright "	20 " "	5 " " "	4½ " " "

No gas is generated when the blowpipe is not in use, and consequently there is no waste of gas and no charge to withdraw overnight.

The apparatus can be recharged in about three minutes. This will appeal to lead burners who are familiar with the time taken to recharge the hydrogen apparatus.

The Howard system of lead burning is being introduced by the Linde Air Products Company, of Buffalo, N. Y., which firm has been so successful in the last years in developing new uses for oxygen and especially in popularizing the oxy-acetylene system of welding. It is hardly necessary to add that this company makes the oxygen from liquid air by the Linde process.

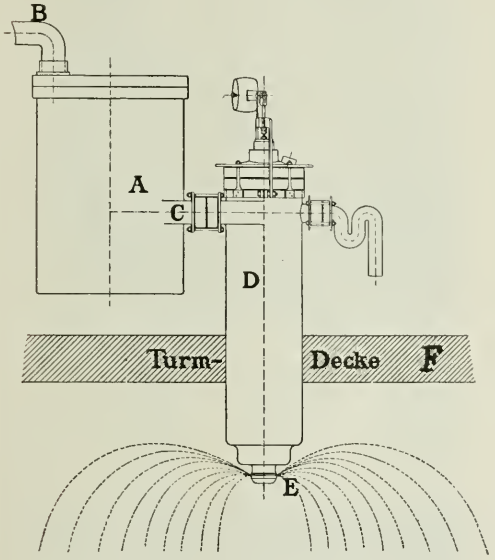
Novel Feeding Device for Glover and Gay Lussac Towers.

The adjoining illustration shows the "effecter distributor" for tower liquid, *i. e.*, a new feeding device for Glover and Gay Lussac towers, built by the Deutsche Steinzeugwaarenfabrik für Canalisation und Chemische Industrie in Friedrichsfeld.

Its chief advantages are that there are no valves nor other movable parts in it, that the liquid is distributed absolutely uniformly over the whole cross-section of the tower, and that only one opening in the top of the tower is required. Neither variations within rather wide limits of the liquid supplied to the tower per hour nor considerable impurities in the liquid are detrimental to the proper operation of the apparatus.

The "effector distributor" is especially suitable for the sulphuric acid industry, since on account of its simplicity and robustness of construction it permits a perfect and uniform distribution of the liquid over the whole tower cross-section in Gay Lussac and Glover towers under all conditions.

The vessel *D* is supplied with the liquid from the filling device



FEEDING DEVICE FOR GLOVER AND GAY LUSSAC TOWERS.

A which operates at periodical intervals. The liquid is gradually given off from *D* to the tower through the spray nozzle *E* in form of a veil of spray of conical form, or rather of umbrella shape. A syphon connected with the interior of the tower conducts any excess of liquid directly into the tower.

The filling device *A* and the spray nozzle *E* are so designed in relation to each other that when the spraying begins the circle which is being sprayed with liquid (or the solid angle of the spray cone) becomes gradually wider while the level of the liquid in the vessel *D* rises. When the liquid in the vessel *D* has reached its highest level and begins to fall again the solid angle of the spray becomes smaller again.

The apparatus is adjustable in such a way that between each period of action there is a shorter or longer interval, or it may be so adjusted that in the moment when the vessel *D* is completely emptied the vessel *A* gives off to *D* the liquid which had accumulated in it in the meantime, so that the spraying process begins anew. In this way the liquid spray changes its conical form continually as though an umbrella was opened and closed successively at regular intervals.

The liquid passes from a reservoir to the filling device *A* through the pipe *B* in a continuous stream. After a certain quantity of liquid has accumulated it is suddenly given off through *C* to the vessel *D*, which then begins to spray through *E* as described before.

This apparatus can be designed for any hourly quantity of liquid and for any tower cross-section, but for the present 5 cu. m. of liquid per hour and a tower diameter of 5 m are the maximum values for which it is built.

Mr. J. W. Sittig, Temple Court Building, New York City, is the sole American agent of this as well as the many other products of the Friedrichsfeld chemical stoneware factory.

Carborundum Refractories.

"Carborundum firesand," one of the by-products of the manufacture of carborundum in the electric furnace, has found very extended use in the last years as a refractory. From a bulletin, entitled "Carborundum Refractories" and issued by the Carborundum Company, of Niagara Falls, N. Y., the following information is extracted.

Chemically carborundum firesand consists of several silicocarbides of compositions ranging from Si_2C_2O to Si_4C_3O, representing the partial reduction of silica by carbon at the intense heat of the electric furnace. This is furnished in the form of finely ground sand and is mixed before using with a binding material such as silicate of soda or fire clay to give it proper cohesion when brought up to heat. A silicate of soda of 52 deg. Baumé is best adapted for this purpose; it must be diluted with a certain quantity of water before being added to the firesand. When clays are used, a crude Jersey fire clay or one of equal purity should be taken. Kaolin may also be used with good results. Prepared fire clay, such as is furnished for laying up fire brick, is generally mixed with ground calcined clay or powdered fire brick, and is not suitable for the purpose.

In estimating the amount of material required for a lining of given dimensions, the weight of the carborundum firesand mixture may be computed as 85 lb. per cubic foot.

Lining of Crucible Brass Furnaces.—For linings of crucible brass furnaces a mixture is used consisting of 70 per cent carborundum firesand, 15 ground fire clay, 8 silicate of soda of 52 deg. Baumé and 7 water.

The silicate of soda is first added to the water and thoroughly dissolved. This solution is then mixed with the clay and firesand until the material acquires the consistency of molding sand. The constituents when thoroughly incorporated form a plastic mixture which is easily molded and has sufficient cohesion when tamped into place to retain its shape until fired. A sheet-iron form is provided of the same diameter as the interior of the furnace, consisting of a sheet of iron bent in cylindrical form, so that after use it can be sprung inward and be easily withdrawn. The center may also be constructed of wood and in such case should be well greased so that it can be drawn without the lining adhering to it.

The mixture is rammed up solidly between the center and the outside shell of the furnace, after which the center may be at once removed and the lining dried out with a slow wood fire. It can then be immediately put in service. In cold weather it is well to warm all the materials before mixing.

It is the practice in some foundries to provide channels or vents to assist in drying out the lining. This is done by placing four or five wooden strips about 2 in. by 1 in., spaced equally around the furnace inside and against the shell. These are withdrawn after tapping in lining.

This same mixture is used for lining ladles for brass, copper or iron.

Linings of Tilting and Rotary Oil-Burning Furnaces.—For tilting and rotary brass furnaces, in which the linings are subjected to the severest possible conditions, a mixture of 86 per cent carborundum firesand, 7 silicate of soda at 52 deg.

Baume and 7 water is recommended. In many cases it is found advantageous to replace about 15 per cent of the firesand with an equivalent amount of kaolin.

This mixture is thoroughly incorporated and tamped between the shell of the furnace and a strong center provided for the purpose. When completed the center is carefully removed and the lining dried out at once with a slow wood fire. It is then gradually brought up to heat by means of the oil flame and can at once be put in service.

Many users of this style of furnace have found it practicable to throw in a small quantity of broken glass when first heating up the lining. This glass, when melted down, can be evenly distributed over the surface of the lining by tilting the furnace, and the interior thus acquires a smoother glazed surface which adds to the durability of the lining.

Patches can easily be made in places where the lining becomes broken or worn. It is best to cut out the broken parts entirely through to the shell, undercutting the old lining, in order that the new portion may be keyed into the old lining. For the purpose of repairs the same mixture is used as when putting in a new lining.

Repair of Firebrick Linings.—A mixture consisting of equal parts of carborundum firesand and fireclay or kaolin is extensively used for patching holes or broken parts of fire brick linings and for filling open joints in furnace walls. It is used to good advantage in ordinary crucible furnaces for brass and steel, also in copper reverberatory furnaces for keeping the side walls in repair. By its careful use it is possible to double the life of any firebrick lining. The mixture is largely used in repairing the Schwartz down-draft furnace, and also as a wash applied daily after the melting. Another efficient mixture for this purpose is the one recommended for crucible brass furnace linings given above.

A mixture of equal parts of firesand and fireclay is used successfully in repairing the brick linings of Tropenas converters, which are subjected to extreme corrosion by the flame.

Carborundum Furnace Paint.—Carborundum firesand may be made up in form of a paint or furnace wash by mixing 65 per cent carborundum firesand with 20 ground fine clay and 15 silicate of soda of 52 deg. Baumé and adding sufficient water to bring it to the consistency of a thick paint, which is applied to the surface of the lining with a broom or stiff brush. Two or three applications gave a proper coating. This makes an excellent protective coating for furnaces subjected to intense flame action and high temperature. It is preferably applied to new brick work, as it does not adhere so efficiently after the linings have become glazed over or covered with slag.

Special applications of carborundum paint are in arch and bridge walls of boiler furnaces which are subjected to severe flame action, in potters' kilns, welding furnaces and malleable iron furnaces. It is of great value in oil furnaces in protecting the brick from the well-known pitting and eating action of the high-pressure oil flame. Carborundum paint is used with good results as a cupola wash in protecting the melting zone of foundry cupolas.

Carborundum mortar is made from a mixture of equal parts of carborundum firesand and fireclay. It is used in laying up firebrick, in patching open joints and holes in brickwork and in general repairs. The life of a firebrick structure is generally limited by the life of the joints. A mortar of carborundum firesand is found to effectively stop the slagging and corroding at the joints.

Carborundum firebrick are furnished in the regular size, 9 in. × 4½ in. × 2½ in., and also in many of the special arch and key shapes. Many special shapes and forms are also made according to specifications and drawings. Carborundum brick have shown great economy in oil furnace work of all descriptions, in connection with the melting of crucible steel in forging and welding furnaces, in electric furnaces and in brass furnaces. They are, moreover, the only brick which will stand the intense heat generated in furnaces using powdered coal fuel.

A New Electric Recording Instrument.

A new type of electric recording instrument, particularly adaptable for use as a recording electric pyrometer, recording voltmeter or ammeter has been designed by the Brown Instrument Company, of Philadelphia.

Recognizing the necessity of a recording instrument which

FIG. 1.—VIEW OF ELECTRIC RECORDING PYROMETER.

is both accurate in operation and simple in construction the Brown Instrument Company has spent much time in designing a recording instrument which can be mounted on the wall or switchboard and used by the most inexperienced workman.

In the common form of recording instruments no means are provided for shifting the recording pen arm out of the way when the chart which receives the record is changed, conse-

FIG. 2.—RECORDER OPEN.

quently there is serious danger of bending the pen in removing the chart. In this respect the new Brown recording instrument is most novel in construction, in that the essential parts of the instruments are mounted on the door instead of in the case. The clock mechanism and chart alone remain in the case when the door is opened. Consequently there is no possibility of

bending the pen because there is no occasion to handle it for any reason whatever. As soon as the door is thrown open the entire voltmeter system and the inking device is swung aside automatically, permitting the old chart to be easily removed and a new one substituted.

Fig. 1 (page 614) illustrates this recorder in operation with the door closed, while Fig. 2 shows the instrument with the door thrown open ready to have a new record chart installed.

The millivoltmeter system which is used in this new recording instrument is a simplified form of the D'Arsonval system which is the only system used in electrical instruments of precision. Also the recording pen comes in contact with the paper momentarily only for the purpose of making a dot, thereby eliminating all friction between the pen and the chart.

An inking pad is placed immediately beside the pen carrying sufficient ink for a week's supply, and this inking pad touches the pen point frequently, keeping the point damp. When the door is closed the arm seen at the left of Fig. 2 and operated by the clock mechanism, comes in contact automatically with the inking device, and every half minute, or quarter minute if preferred, it pushes the inking pad away from the pen, permitting it to swing freely. It then falls, allowing the inking pad by its own weight to press the pen against the paper.

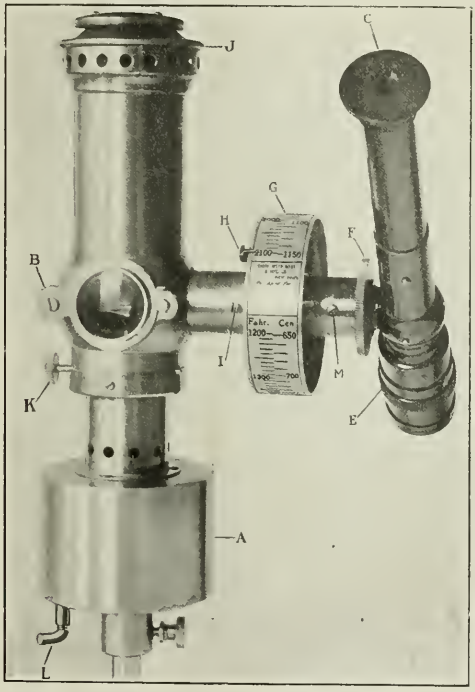


FIG. 1.—PYROSCOPE.

This extraordinary simplicity of construction, the freedom from unnecessary parts, as well as the frictionless operation on the paper, will strongly appeal to those who understand the value or rather necessity of simple design for recording instruments.

Filter Papers.—We have received price list and samples of filter papers in sheets, rolls, circels, and folded-filters, made by Messrs. Carl Schlaicher & Schüll, filter-paper manufacturers of Düren, Rhenish Prussia. Messrs. Eimer & Amend of New York City are the sole agents for the United States.

A New Optical Pyrometer.

In many fields of metallurgy it is now clearly recognized that exact heat treatment is all-important and that the skilled eye of the most experienced attendant is not able to judge temperatures always correctly on account of the considerable variations of the illumination of the room with the weather, etc. This explains the rapid development of the pyrometer industry and the ready introduction of the pyrometer into commercial practice in recent years. Naturally the tendency in the design of various pyrometer types has been to develop robust fool-proof shop instruments from what were originally laboratory instruments requiring careful handling.

FIG. 2.—MEASUREMENT OF THE TEMPERATURE OF A TOOL BY PYROSCOPE.

This same tendency is manifest in the latest design of an optical pyrometer, called the pyroscope and built by the Shore Instrument & Manufacturing Company of 555 West Twenty-second St., New York City. There is no necessity to deal here again with the underlying principle of all optical pyrometers. The chief points of the Shore pyroscope are its simple design, strong and rugged construction and comparative inexpensiveness. It is a little less accurate than a first-class and perfectly calibrated pyrometer, but is stated to give correct readings within 10 deg. Fahr.

The standard lamp used for comparison is a miniature kerosene flame. Ordinary kerosene is filled in the tank *A*, Fig. 1, the wick is lighted as in any ordinary lamp and the flame is adjusted to normal height (about ¾ in.) by the knob *K*. By now looking through eye-piece *C* at the hot tool, etc., which undergoes heat treatment, one gets a clear view as through any telescope. After focusing by turning the knurled ring *E*, the knob *F* is turned, which works the colored diaphragm and

controls the light intensity of the comparison reflector *R* shown in the special diagram in the upper left-hand corner of Fig. 2, which shows the image seen through the pyroscope. Knob *F* is turned until practically no difference can be seen between the redness of the reflector *R* and the tool under treatment. This can best be tested by going a shade higher or lower until balanced. The indicator *H* on the graduated drum *G* then gives directly the temperature reading.

By means of locking by the set screw *M* it is possible to set the instrument at a given temperature so that the work can all be heated up to a certain degree. The telescope can be used in any desired position on its pivot.

Recording Pressure Gauges.

The use of recorders has rapidly increased in recent years in connection with all kinds of measuring instruments, since a continuous record, furnished by a recording instrument, represents the best and most reliable control of the operation of a plant and the work of the attendants.

The Columbia pressure recording gauge, made by the Schaeffer & Budenberg Manufacturing Company, of Brooklyn, N. Y., and shown in the illustration, is noteworthy especially for

RECORDING PRESSURE GAUGE.

some special features of the record chart, namely uniform graduations, large effective record space and the division of the border into a black and a white section, denoting the day and night record respectively.

Columbia recording gauges are made in two sizes, suitable for 8-in. and 12-in. charts respectively and for pressures ranging from 4 in. water pressure up to the highest working pressure met with in practice.

Some of the applications of these gages are the recording of steam, air, gas, water and ammonia pressures. These gauges may be had in dust and air-tight or moisture-proof case, and for temporary installations or field service portable gauges are provided.

The recording chart shows maximum and minimum pressure carried, the rate of rise or fall in pressure, in brief it is a complete log of the operation as indicated by the pressure at any time. A file of charts in the office of the superintendent enables him to compare the work done in the plant hour by hour and day by day and to study in detail how the plant economy is affected by the pressure.

On the other hand the workman likes the recording gauge as it relieves him from the time-consuming and inconvenient writing down of the pressure gauge readings on the report sheet.

On the whole, the installation of pressure recording gauges leads to more conscientious operation and should prove a great incentive to improvements in all chemical works where gases or liquids are carried under pressure.

Titanium Steel.

The American Iron and Steel Association published recently the following statistics of the tonnage of various alloy steels during 1910.

Titanium steel	326,316	gross tons
Nickel steel	106,707	" "
Nickel-chrome steel	52,021	" "
Chrome steel	23,550	" "
Manganese steel	19,360	" "
Vanadium steel	9,049	" "
Other Alloys steel	30,816	" "
Total	567,819	

As is pointed out in a recent circular of the Titanium Alloy Manufacturing Company of Pittsburgh, Pa., these statistics show that the output of titanium steel for 1910 was not only 300 per cent greater than the next largest nickel, but it was greater by 84,813 gross tons than the output of all the other alloy steels combined.

It is, of course, a question whether titanium steel can be considered as an alloy steel in the same sense as, for instance, nickel steel or chrome steel. In the case of the latter two, the nickel and the chromium are alloyed in certain proportions with the steel and give it certain valuable properties.

On the other hand, titanium was originally introduced in the steel industry as a cleaner. Titanium is an excellent deoxidizer and has also a very high affinity for nitrogen. It acts as a purifier of the steel, removing the gases from it, and improves greatly the structure of the steel. When acting in this way the titanium passes into the slag and if no excess of titanium is used, no titanium is contained later in the steel. Such titanium-treated steel is, therefore, not an alloy steel like nickel steel or chrome steel, but the results obtained by the purifying titanium treatment are so remarkable that they alone would explain the enormous increase of the use of ferrotitanium in the steel industry in the last few years.

However, more recently several investigators have found that the beneficial effects of titanium are not restricted to its action as a purifier but that titanium may give valuable properties to steel when the titanium is used in excess over that needed for purification alone. Facts illustrating this action have been cited to us from different investigators working quite independently of each other. It is to be hoped that exact information concerning this most interesting subject will soon become available.

The Non-Ferrous Metal Market.

Prices have fluctuated somewhat since our last report owing to conditions on the Stock Exchange and in foreign markets. The report of the Copper Producers' Association depressed the domestic copper market, as it seemed to indicate a contraction of business abroad. Tin fluctuated considerably as a result of both foreign and domestic conditions. Lead has eased off somewhat, while spelter has advanced.

Copper.—The latest quotations on this metal show a tendency toward lower prices, and there has been a large of-

fering on the part of Lake producers who wish to dispose of stocks before navigation closes. The quotation for Lake copper is 12⅛ to 12⅜ cents, with electrolytic at 12.05 to 12.10 cents.

Tin.—At the close of September tin could be bought at about 35⅝ cents in New York, but as a result of the activity of the Syndicate abroad prices rose, and at present October tin is quoted at 41 cents.

Lead.—As a result of two successive reductions by the principal seller, this metal has fallen off about ¼ cent per pound, and there is little demand in view of the unsettled conditions. New York lead is quoted at 4.20 to 4.25 cents; St. Louis at 4.05 to 4.10 cents.

Spelter.—Prices have steadily advanced in view of large exportations and shortage in the domestic supply. Business has been on a heavier scale and consumers are buying in such quantities as can be supplied. Latest quotations were firm at 6 to 6.15 cents, New York, and 5.85 to 6 cents, St. Louis.

Other Metals.—Business in aluminium is somewhat improved, but prices are no higher. The metal is quoted at 18.5 to 19 cents, New York. Antimony is weaker and prices are reduced to 7 to 8.25 cents for various grades. Prices have fallen slightly in quicksilver; New York prices being $45 per flask of 75 lb.; San Francisco, $44.50.

Notes.

The **Kestner Evaporator Company,** which for a number of years has been located at 333 Walnut Street, Philadelphia, has recently removed its business to the Commonwealth Trust Building, Twelfth and Walnut Streets, in the same city.

Monel Metal Casting.—An interesting feat in the casting of Monel metal was recently accomplished in the foundry of the Bayonne Casting Company, Bayonne, N. J., where a four-blade propeller wheel, 16 feet in diameter, was cast in somewhat less than an hour. Fully 18,000 pounds of the metal were required to allow for gates and finish, the melting temperature being about 2500 Fahr. and the finished weight about 14,000 pounds. The wheel, because of its greater strength and freedom from corrosion, is to take the place of a steel wheel on the S. S. Madison, of the Old Dominion Line.

Mr. **H. M. Hirschberg,** 30 East Twentieth Street, has recently been appointed American representative of the Planiawerke, Ratibor, Silesia, Germany, manufacturers of the well-known carbon electrodes bearing their name. There is such a big demand for products of this factory, that contracts are now being made for the latter part of 1912 and 1913 deliveries, Mr. Hirschberg already having closed for almost the entire output of the factory available for this market. These electrodes are being made in sizes up to 600 millimeters (2 feet) square cross section for rectangular electrodes or 600 millimeters (2 feet) diameter for round electrodes and in lengths up to 2000 millimeters (6 feet 7 inches). It is especially noteworthy that these electrodes are provided with a screw end so that new length can be added when required, thus doing away with all waste of carbon ends.

A New Departure in Handling Gritty Water.—For many years the pulsometer has found favor for dredging work, coal mining, sand suction, etc., because it has no rubbing surfaces so that no difficulty is experienced in handling gritty water. To overcome the terrific cutting action of high velocity water containing very hard flinty matter, the Pulsometer Steam Pump Company, of 17 Battery Place, New York City, announces that pulsometers are now being cast by a special patented process wherein the shell, while of the usual cast iron construction to give ample rigidity, is completely lined on the inside with a surface of carborundum. Sand blast and other abrasive tests would seem to indicate that this lining will stand almost indefinitely the wear of water moving at high velocity and containing anywhere from 40 to 60 per cent of hardest material.

The International Institute of Technography, an association of patent experts and technical bibliographers, has been organized to make comprehensive prior art examinations of patents and literature by a carefully organized system. A careful compilation of any single prior art from all the records, by an individual, requires weeks or months. Such laborious and expensive compilations have already been made, however, in comparatively every art, covering each to at least a comparatively recent period. The association has made it its business to know of, to compile, and to classify all such compilations, to ascertain where they are to be found, and to have a qualified expert at hand to examine them. The offices of the Institute are at 50 Church street, New York City.

The **Aero Pulverizer Company** announce that they have made an arrangement with the Cockburn Company, with works at Jersey City and offices at 126 Liberty Street, New York, to take over the manufacture and sale of the Aero pulverizer. The Cockburn Company has always manufactured these machines for the Aero Pulverizer Company, so that there will be no change in the product or workmanship. Mr. Wm. A. Evans will have supervision of manufacture and sales.

The **Copper Extraction Company,** Anaconda, Mont., has placed with the Ruggles-Coles Engineering Company, of New York, orders for two class "A 2," one class "B 10" and one class "F 8" dryers, for drying copper tailings.

Thermometers and Other Instruments for Breweries.—That chemical engineering plays an important part in brewing was well illustrated at the American Exposition of Brewing Machinery, Materials and Products, which will be held at the Chicago Coliseum, Oct. 12 to 22. Prominent among the exhibitors was the Schaeffer & Budenberg Manufacturing Company, of Brooklyn, N. Y., showing a complete line of its high-grade engineering instruments, among them its new recording thermometers, its special features being the mercury-filled steel capillary tube and the night and day chart, both new departures in thermometer construction. A full line of its brewers' thermometers, together with steam and ammonia gages for indicating and recording purposes, was shown, also its new steam engine and ammonia compressor indicators. Messrs. W. H Shenton and T. J. Mentin officiated at their booth.

Two varieties of natural graphite are generally recognized in the trade—crystalline, or that possessing a lamellar, flaky or fibrous structure, and amorphous, including all other forms. Domestic flake graphite is replacing to some extent the Ceylon graphite used in the manufacture of crucibles for the metal industries, and the increased demand is being met by increased domestic production rather than by increased importation.

Obituary.

Horace H. Emrich, manager of the copper refinery of the Kyshtim Mining Works Company, of Kyshtim, Russia, was killed, according to newspaper cable dispatches, on October 17th, while standing at the window of his residence, by a gun shot from the outside. The Russian government has ordered an investigation. Mr. Emrich is a well-known member of the American Electrochemical Society and has here many friends who will be greatly shocked by the news of his sudden death. He was formerly superintendent of the copper refinery of the American Smelting and Refining Company at Maurer, N. J. In a letter which we received from him this summer, he wrote that he had just finished the design of and was erecting a modern steam generating plant for his refinery. The concluding words of his letter were: "Please give my regards to my friends and tell them that I do not find Russia such a fierce place to live in as we Americans are led to believe. In fact my wife and I find Kyshtim living, people, and climate all that could be desired, and we have spent a very pleasant year here."

Personal

Mr. Philip Argall, manager of Stratton's Independence, Ltd., sailed from New York on Oct. 18 to attend the annual meeting of his company. Mr. Argall will spend some time traveling abroad before his return to this country.

Mr. C. A. Dart has resigned his position as manager of the Rambler Copper & Platinum Company, Holmes, Wyo., and is now in Denver.

Mr. R. R. Foster, late superintendent of the metallurgical plants of the La Tula Mining Company, at La Luz, Guanajuato, Mexico, resigned his position to enter consulting practice. He will have headquarters at Torreon, Coahuila, Mexico.

Dr. Charles H. Fulton is now in charge of the mining and metallurgical department of Case School of Applied Science at Cleveland, Ohio. Dr. Fulton recently resigned his position as president of the South Dakota School of Mines, at Rapid City, in order to relieve himself of executive duties and devote more time to professional work.

Mr. Charles W. Goodale, of Butte, Mont., attended the meeting of the American Institute of Mining Engineers at San Francisco last month.

Mr. Franklin Guiterman, Colorado manager for the American Smelting & Refining Company, returned to Denver Oct. 1, after spending the summer in Europe studying the possibilities of electric zinc smelting.

Mr. Victor G. Hills, of Denver, Colo., who has been in Nova Scotia for some months developing a scheelite property, returned to Colorado for a short time last month. He was called as an expert in a mining suit at Telluride.

Mr. E. M. Johnson is now superintendent of the Altoona Zinc Works, at Altoona, Kan., having lately resigned his position at Depue, Ill.

Mr. Henry P. Lowe, managing director of the Frontenac Mines, Ltd., Central City district, Colo., has gone to England to attend the annual meeting of his company.

Mr. W. J. Quigley, manager of the Chihuahua Mining Company, Chihuahua, Mexico, spent last month in Denver.

Mr. P. E. Van Saun, of the Mill & Smelter Engineering Company, New York, is in New Mexico on professional business for his company.

Mr. Charles T. Van Winkle has resigned his position as superintendent of the Magna plant of the Utah Copper Company, Garfield, Utah, and has opened an office as consulting engineer in Salt Lake City.

Mr. A. W. Warwick, who has been in New York making tests on the Clancy cyanide process, has returned to Colorado, and is at Victor superintending the opening of the Colburn mill in which the Clancy process is to be used.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

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ORE TREATMENT (*Continued*).

473,186. April 19, 1892. Parker C. Choate, Brooklyn, N. Y. Zinc. Treats complex ores containing zinc, iron, sulfur, lead, gold, silver, antimony, arsenic, selenium, cadmium, bismuth, etc. The ore is crushed and, if it contains an excess of sulfur, roasted at a moderate temperature. It is then mixed with coal-screenings, and roasted on a perforated grate beneath which air is admitted. The zinc and other bodies volatilized therewith, e.g., oxides of antimony, arsenic, selenium, cadmium and bismuth, are caught in bags. The condensed fume is taken from the bags and roasted at a temperature of from 300 to 800° F. in a muffle-furnace, to drive off the impurities more volatile than lead or zinc. The zinc oxide is dissolved in sulfuric acid, the lead sulfate settling out. The zinc sulfate solution is electrolyzed between anodes of lead or carbon and cathodes of zinc or copper. The freed acid rises to the surface of the cell and is drawn off through overflow pipes. As it contains considerable zinc, a portion is returned to the lixiviating tank and regenerated by the addition of fresh zinc oxide. Another portion is returned to the cell. Water is added from time to time to replace the loss. The deposited zinc is stripped off the cathodes and melted, or is melted therewith when zinc cathodes are used. The residual sulfates from the bottom of the lixiviating tank are smelted. The tailings from the roaster are also smelted to recover the iron, gold and silver.

484,809. Oct. 25, 1892. George J. Atkins, of London, England. Gold, silver, etc. The apparatus illustrated is substantially the same as that of the patentee's previous patent 473,105. The amalgamating apparatus is connected to the cathode of the electrolytic apparatus or to the positive pole of a dynamo, and a supplemental anode is placed in the exit mouth of the amalgamating apparatus to cause direct deposition of any gold and silver in the solution. The process is claimed as a continuous treatment consisting in passing the ore through an anode compartment while in contact with the anode and then amalgamating it; or, after the anode treatment, passing it slowly through the cathode compartment and thence to the amalgamator.

495,212, Apr. 11, 1893, John Franklin Wiswell, of West Medford, Mass., Assignor to the Electric Gold and Silver Chlorination Company, of Kittery, Maine, and Boston, Mass.

Amalgamates precious metals by adding to the ore in a crusher a solution consisting of mercuric chloride and sodium hypochlorite. The mercuric chloride is prepared by passing a current through cups containing mercury immersed in common salt solution, carbon cathodes being used. The overflow from this tank is electrolysed, carbon electrodes being used, to form sodium hypochlorite. The calomel formed in the cups is heated in a separate receptacle with aqua regia to form soluble mercuric chloride. Iron must be present during amalgamation, and if not present in the ore may be liberated from the iron crusher by the sodium hypochlorite, or may be introduced as iron filings.

BOOK REVIEWS.

Metallurgy of Tin. By HENRY LOUIS, D. Sc. 8vo; 138 pages; 41 illustrations. Price, $2.00. New York: McGraw-Hill Book Company.

A reprint of a monograph by the author in "Mineral Industry," 1896, with such more recent information as was available. The author deplores the non-progressive character of the industry, also the profoundly secretive attitude of tin smelters, and pungently remarks: "To what extent these two facts, the slow advance of the industry and the secretive habits of those engaged in it, are related as effect and cause, may be left to the judgment of the reader."

Hendricks' Commercial Register of the United States for Buyers and Sellers. Twentieth annual edition. 1420 pages. Price, $10. New York: Samuel E. Hendricks Company.

This well-known commercial register for buyers and sellers, which covers all branches of architectural, engineering, electrical, mechanical, and kindred trades and professions, appeared first in 1891. The twentieth edition, just issued, is more complete than any of the former.

The total number of classifications in the volume is now over 45,000, each representing the manufacturers or dealers of some machine, tool, specialty or material required in the above named industries. The whole represents upward of 350,000 names and addresses.

The new edition will, undoubtedly, prove as useful and reliable as the former ones.

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Chlorination versus Cyanidation.

Less than a decade ago the respective merits of the chlorination and cyanide processes of treating gold-bearing ore formed the basis of many a heated controversy which aligned leading metallurgists in two opposing factions. Each process had its champions, but neither held undisputed sway in practice or received the unqualified approval of a majority of the profession. To-day cyanidation appears to have the field to itself, only one prominent chlorination plant remaining in this country, viz., that of the United States Reduction and Refining Company in Colorado.

With apparently such a sweeping victory for cyanidation it might be surmised that the disparity between the two processes is greater than it really is. An excellent opportunity has been afforded in Colorado for a close comparison of the two processes as applied to Cripple Creek ores. This comparison is the more complete because both chlorine made by electrolysis and chlorine generated from bleaching powder and sulphuric acid have been used for years in two competing custom plants. It is a significant fact that the one using electrolytic chlorine continues to treat ore by chlorination, while the last to use bleaching powder and acid has abandoned the method in favor of cyanidation.

In the matter of initial cost of installation of plant the chlorination process is more expensive than cyanidation, due largely to the lead-lined solution tanks, etc., required in the former process. If electrolytic chlorine is used, another item also is added to plant cost. Again, the chlorination process is restricted in its application to non-argentiferous gold ores and to those containing little or no lime. In the latter respect chlorination has been losing ground in Colorado, as lime has been an increasing constituent of Cripple Creek ores, requiring the addition of as much as 40 lb. of sulphuric acid per ton of ore treated.

The cost for chemicals in the two processes will be about the same per ton of ore treated, provided sufficiently cheap electrolytic chlorine is available for chlorination. If chlorine can be made at a cost of four cents per pound for salt and power, the latter costing about one cent per kw-hour, it is likely that chlorination can compete with cyanidation in cost of treating a favorable ore. Under the conditions given the cost of chlorine would range from 18 to 30 cents per ton on sulpho-telluride ores such as are now being mined in the Cripple Creek district.

In refining costs chlorination has an advantage over cyanidation, as it also has in the possibility of making monthly cleanups, thereby avoiding the necessity of carrying a large amount of gold in solution to be estimated in the monthly inventory. This is considered quite an important point in favor of chlorination for a custom plant.

Obviously the chlorination process has lost ground for good economical reasons. But it is a fact that in none of the chlorination plants of the past or present the latest advances in electrolytic chlorine processes (which have been significant in more than one way) have been employed, simply because they were not yet available when those plants were built. With electrolytic chlorine now constantly looking for new outlets, it is not impossible that the pendulum may swing back and chlorination may get another and better chance in the metallurgy of certain particularly suitable ores.

Spelter and Natural Gas

One of the most striking examples of the attractiveness of cheap fuel in metallurgical enterprises, is to be found in the Kansas-Oklahoma natural gas fields and the large zinc smelters located there. Statistics on the Western production of spelter show the transference of the zinc smelting industry from the vicinity of coal fields in Illinois and Missouri to the newer and more attractive gas fields of Kansas and Oklahoma. But no doubt with the gradual exhaustion of natural gas we shall witness a migration of the smelters back to Illinois.

In the first eight years of the decade from 1901 to 1910 the production of spelter in Illinois remained almost stationary at from 45,000 to 50,000 tons per annum. This increased slightly in 1909 to about 67,000 tons, and again in 1910 to 73,000 tons. The Missouri production during this decade held a fairly steady average at about 11,500 tons for the first eight years, but fell to 7000 tons in 1909 and 5500 tons in 1910. During the same period, however, the spelter output in Kansas increased from about 75,000 tons in 1901 to 105,000 tons in 1910. And beginning with 1907, which was the first year in which Oklahoma was credited with spelter production, that State has increased in output from 5000 tons to 35,000 tons in 1910.

This change in the reduction center of the zinc industry has been due to the cheap natural gas available in Kansas and Oklahoma, and any future change to Illinois will be due to cheap coal, for the spelter producing States are but minor producers of zinc ore, while the largest ore producing State is a diminishing producer of spelter. When the production of spelter in 1910 was apportioned to States according to source of ore, Kansas was found to have produced but 4.05 per cent of the spelter made in the United States; Illinois, 0.61 per cent; Oklahoma, 0.91 per cent, and Missouri, 55.71 per cent. But when apportioned to locality in which the ore was smelted, regardless of its source, Kansas led in spelter production among the Western States, followed by Illinois, Oklahoma, and Missouri in the order named.

The natural gas in Oklahoma and Kansas is nearly pure methane, containing approximately 90 per cent CH_4, from 1 to 2 per cent CO, and the balance oxygen, nitrogen, and carbon dioxide. The heat value of methane is 966 b.t.u. per cu. ft., and of carbon monoxide, 344. Based on the analysis of the natural gas it has a heating value of about 875 b.t.u. This is a high value compared with that of producer gas, which averages 150 to 160 b.t.u. per cu. ft. Natural gas is now bought for smelting in Kansas and Oklahoma for three cents per thousand cubic feet, which is equivalent to producer gas at 0.6 cent when fuel values are considered. Prices for other industrial uses of natural gas in this region are higher than for smelting, the enormous volumes used in the latter work giving the advantage of a lower price. For firing boilers in the Joplin field, to which district a great deal of gas is piped from Oklahoma, gas costs 12.5 cents per thousand cubic feet, while for gas engines the cost is 25 cents. Electric power generated from gas is sold at Joplin at a cost of one cent per hp-hour, and can be generated by individual large plants at a cost of six to seven mills per hp-hour.

In view of the favorable natural resources of the Kansas-Oklahoma-Illinois district it is not surprising that ore and concentrates from Montana, Utah, Colorado and other Western States, as well as from Mexico, are shipped there on favorable freight rates for the long haul. But the prospect of exhaustion of the gas is always in sight, and even now the tendency is to raise the price. Kansas has declined as a producer, but Oklahoma is rapidly coming to the front and promises large volumes of gas. Seven cents per thousand cubic feet is now being asked for some industrial uses, and even higher prices will prevail where the gas has to be piped long distances.

The Electric Furnace and the Scrap-Metal Business

We have pointed out on several occasions in these columns that the electric smelting and refining furnace has several peculiar fields of its own, due to the differential advantages which its own nature gives it. Chief among these fields is the electric refining of a semi-refined steel producing a high-grade tool steel, which has had a considerable degree of commercial success in the past few years. Then the production of the difficultly reducible ferroalloys, such as ferrosilicon, ferrotitanium and ferrochrome, has been an assured commercial success both here and abroad for some time past. At present we see the electric brass furnace and the electric smelting furnace for zinc-lead ores both measurably close to the commercial stage. In all these three are certain innate points of superiority, or, to be concise, certain points of differential advantage in the application of electric heat to the work to be done.

These same advantages repeat themselves in the case of the melting down, reducing and refining of metal refuse or waste products from cities and manufacturing plants. The first point of superiority that electrothermic treatment possesses over the older forms of treatment is high degree of control over temperature. This is apparent even to the layman, and it is both true and humorous that many engineers are more or less laymen when it comes to electrometallurgy. Granted once the proper reciprocal relation between the generation of heat and the absorption of heat we have as a logical conclusion perfect control over the temperature of the reacting zone and over the operations therein. As a matter of practical fact, in any successful electric furnace we have an approximation to this ideal condition.

Next in apparent importance is the control over the chemical properties of the fluxes and slags and the reducing properties of the atmosphere. This is particularly true of the electric zinc furnace. Other advantages are absence of dust and heat, the low labor charge and the freedom from exacting care by the furnace men. It is really surprising to see an electric furnace in operation with pointers on the ammeter and voltmeter staying at their positions "as if they were glued there." Such is the great regularity of electric smelting when it is properly

understood. As the thermal efficiency of the electric furnace is always high, provided it is over 150 kw capacity, all these factors combine together to make a low working charge with a high degree of metallurgical economy.

The greatest advantage to our way of thinking is a commercial one—the low capital cost provided power is bought from the outside and only the transformers and switchboards are owned. As the energy transferred per unit of working volume is large—3 kw to 10 kw per cubic foot—the furnace is not expensive.

So we see that every item of reasoning points directly to a wide potential field for the electric furnace in the smelting and refining of scrap metal and waste products that our foundries and manufacturing centers produce. The commercial reasons are both simple and obvious. In any large city many tons of such stuff as brass-founders skimming, skeleton brass, steel borings, etc., are produced each year. In any large city there is always a surplus of electrical power that can be sold reasonably on the 24-hour basis for nine months and on the 20-hour plan during the months of December, January and February. Join these two industrial things by a successful electric-furnace process and we have the raw material which is produced locally turned into the finished product locally. Four handlings and two railroad hauls are avoided. Besides there is created a market for some of the surplus power of the central station. With such a powerful metallurgical instrument at hand, the losses of metallic values would be necessarily low if this same agent were intelligently applied. In general there seems to be no room for doubt that the electric furnace will find a permanent useful place as a means for working up waste material into available material. In this century elimination of waste equals efficiency.

Of the various specific instances that seem apparent, the reduction of skimmings from brass or bronze foundries strikes us as the most promising. These analyze from 40 per cent to 70 per cent zinc plus tin plus copper. Consider a "government or navy" bronze of the composition copper 88 per cent, tin 10 per cent, zinc 2 per cent, which sells in the ingot form for 15 cents or 16 cents. The dross from this will contain a greater proportion of the more easily oxidized tin and zinc. It is sold for its copper contents alone at from 3 cents to 4 cents a pound. Experiments of which we have definite knowledge show that it is not a difficult task to reduce this in an electric furnace and with the mixture of copper to produce an ingot substantially the same as the original alloy. How much better it would be to have this done than to have the skimmings bought by a copper smelter where the copper alone is recovered and that, too, in the debased form of copper-matte. To repeat, the place to do the converting is at the point of production. Naturally there are obvious difficulties. Chief of these is the chance that there would be contamination of skimmings by iron or other foreign substances. It may also be argued that there will be a displacement of the crucible by the electric alloy-furnace which makes no dross and that hence the supply of skimming will rapidly diminish. But practically this should not carry too much weight because human inertia will see to it that the crucible furnace will be only gradually displaced by the electric alloy-furnace. The electric lamp has not supplanted the gas lamp, nor has the latter kerosene lighting. Indeed some wiseacres say that the consumption of candles per capita is as large as ever.

The next apparent fissure in the existing commercial structure into which the electric furnace can insert its wedge is the melting down of steel borings, chips and turnings. The same remark can be made of similar materials of cast iron. Their bulky nature makes them expensive to transport and smelt. The electric furnace which has no blast to oxidize the great surface exposed or to blow them out of the stack has a marked natural advantage over the open-hearth furnace in the case of steel borings, while it has a slight though marked advantage over the cupola furnace in the case of cast-iron borings. In each of these instances the proper and logical procedure is to take the borings, chips and turnings which are commercially unavailable, practically speaking, and to turn them electrically into steel castings or iron castings which are commercially much desired and of great worth. A fact that is apparent and of importance is that it is simple to attain in the electric furnace a casting of good physical properties, *i.e.*, freedom from slag, blow-holes and dissolved oxides. Interdependent with this is the fact that good physical properties conduce far more to a sound casting than do good chemical properties. Naturally, there are many essential details of engineering and business that must be learned and mastered in order to make this a success. But the manufacture of the finest and most difficult steel castings or iron castings out of the cheapest kind of scrap is not only theoretically possible but actually has been done. How attractive it is commercially can be seen by the fact that the raw materials sell at present for from 25 cents to 40 cents per hundred pounds and the finished material sells for as high as 6 cents a pound or over.

Other specific possible examples can be adduced, such as the smelting of tin dross, which indeed is being accomplished commercially, or of Babbitt metal dross. The melting of aluminium dross is possible, but here it is not possible to reduce the oxide, but only to save the entrained prills of metal. The reduction and distilling of galvanizer's skimmings and the galvanizer's "dross" is also open to the electric furnace. Unfortunately, the problem here is complicated by the presence of sal ammoniac in case of the skimmings and the necessity for a practical condenser in both cases. The refining of plumbiferous refuse products is also feasible. But here the older types of furnaces are securely intrenched. But the distinguishing characteristic in many of the above instances is that no great inventive ability and no great business enterprise is needed. A combination of average metallurgical experience and of average business knowledge and leavened by a grain of originality is enough.

A certain amount of commercial inertia, of course, acts like a brake on this and on all material changes in our industrial structure. Indeed it is very proper that it should be so, even though the adventurous minded think it unnecessary. But we are firmly of the opinion that in every manufacturing center good opportunities exist for the "little fellow" in electrothermal treatment of waste metal material, for the capital cost of a five-ton electric smelting and refining plant should not exceed $20,000. Broadly speaking, both specific and general observation indicate that the "trusts" leave plenty of just such opportunities for the "able little fellow."

American Institute of Chemical Engineers.

The fourth annual meeting of the American Institute of Chemical Engineers will be held in Washington, D. C., from December 20 to 22. The headquarters will be at the New Willard Hotel, where all the professional sessions will be held.

The first session will be held on the morning of December 20 (Wednesday), when the following papers will be read and discussed:

Clarence Hall, "Advances in Testing Explosives."

Richard X. Meade, "Distribution of Power in Portland Cement Manufacture."

J. T. Baker, "Manufacture of C. P. Acids."

L. S. Hughes, "Combustion of Pulverized Coal."

Ludwig Thiele, "Manufacture of Gelatin."

S. F. Peckham, "The Natural Bituminous Rocks of the United States."

In the evening the retiring president, Dr. F. W. Frerichs, will present his address on some problems in chemical engineering practice (manufacture of chloroform from alcohol; construction of laboratory apparatus; manufacture and testing of shipping cylinders for liquid ammonia gas).

This will be followed by a paper by John C. Minor, Jr., on the manufacture and testing of carbonic acid cylinders.

On the morning of December 22 (Friday), there will be a symposium on the U. S. patents, papers to be presented by Messrs. E. B. Moore, Edw. T. Fenwick, Walter D. Edmonds, and R. N. Kenyon.

In an evening session Mr. F. G. Wheeler will present an illustrated lecture on the adaptation of the centrifugal pump to chemical service, and Dr. J. C. Olsen will read a paper on a course in chemical engineering education.

The social functions include a subscription dinner at the New Willard on Thursday evening and a reception at the White House on Friday afternoon.

The program of visits and excursions includes the Bureau of Standards on Wednesday afternoon, the U. S. Proving Grounds at Indian Head, Md., on Thursday, the Patent Office on Friday afternoon, and the steel plant at Sparrows Point, Baltimore, or the cement plant of the Tidewater Portland Cement Company on Saturday.

Surface Combustion.

A most interesting lecture was presented on October 27, by Prof. William A. Bone, of the University of Leeds, Eng., at the Chemists' Club in New York, before the members of the different chemical societies. Professor Bone gave a review of the results of the work which he has done on this subject for a long series of years, and his discourse, as well as the very successful experiments by which the lecture was illustrated was greatly enjoyed.

The subject is fully covered in a special article by Mr. Kershaw on page 628 in this issue.

The Western Metallurgical Field.

Cyaniding Concentrates.

A fine example of careful experimental work, followed by successful application of the results obtained, is to be found in the cyaniding of concentrate at the Treadwell mines, Alaska. At the time the experimental work was undertaken the concentrate was being shipped to the smelter at Tacoma, and the cost of treatment of 3-oz. gold concentrate was $11.95 per ton. From the experimental work it was estimated that the concentrate could be treated at home by cyaniding at a cost of $3.25 per ton. The details of experimental work and the present operation of a 100-ton plant were presented by Mr. W. P. Lass, superintendent of the plant, at the recent San Francisco meeting of the American Institute of Mining Engineers. We note here the most interesting features of the experimental work, which will show the development of the process now used.

The concentrate from the various mills amounts to about 1.8 per cent of the original ore, and contains: Fe, 40 per cent; S, 1 per cent; SiO_2, 11 per cent; Au, 2.5 to 4 oz. gold, and 0.75 oz. silver per ton. Preliminary tests were made in glass jars, after which 50-lb. composite samples were ground to 200 mesh in a clean-up barrel with iron balls, the pulp passed over amalgamated plates and then agitated in small Pachuca tanks 4 ft. high and 14 in. in diameter. The results showed that 75 per cent of the gold could be recovered by fine grinding and amalgamation, and 96 per cent by amalgamation followed by cyaniding. The next step was the construction of an addition to one of the mills, and the erection of a 4-ft. x 12-ft. Abbé tube mill. Various forms of classifier were tried, the Dorr proving most satisfactory. Callow cones were used for dewatering, and the agitation was carried on in a Pachuca tank 10 ft. in diameter and 22 ft. high.

The results further showed that by grinding in a 1.5-lb. cyanide solution an extraction could be obtained by cyaniding alone equal to that obtained by cyaniding and amalgamation, or 96 per cent. Thus a satisfactory extraction was obtained either by amalgamating and cyaniding or by cyaniding direct. A preliminary agitation with alkali solution was found to shorten the time of cyanide treatment and save 25 per cent in cyanide consumption. Passing the air used for agitation through a receiver filled with caustic soda or milk of lime also decreased the cyanide consumption, presumably by the removal of oil and CO_2 from the air. When grinding in solution of over 1 lb. per ton, followed by amalgamation, it was difficult to keep the plates bright, due to a dull white surface deposit which, if allowed to remain, turned to a dull gray. A muntz-metal plate was substituted for a copper plate, but as all the plates were coated, no variation in the result was noted, and there did not appear to be any essential advantage in its use.

The results obtained from a period of two years' investigation justified the building of a 100-ton plant. The plans provided for the use of amalgamation if desired, and ultimately nearly 5000 tons were treated by each method before finally deciding to abandon the time-honored amalgamated plate.

The following table shows the work of one of the 5-ft. x 22-ft. tube mills when treating an original feed of 88.75 tons per day.

	+ 100 mesh	+ 200 mesh	— 200 mesh
	%	%	%
Feed	48.7	41.5	9.8
Discharge	10.1	26.4	63.5

This product is classified in Dorr classifiers and only the overflow passes to the cyanide tanks. This material screens more than 98% through 200 mesh.

An interesting device is used in the Pachuca tanks to bring settled concentrate into circulation. After the first treatment the pulp is allowed to settle, and it forms a 30-ton packed mass in the bottom of the Pachuca. This is brought into agitation in 15 minutes by means of a device known locally as a spider, which is an adjustable hollow annular casting with radiating fingers, the whole encircling the central agitation column. When the charge is to be put in agitation the spider is lowered by a small hand windlass until it rests on the settled charge. Air is then turned through the fingers, and at the same time the solution for the next treatment is run into the tank. The device rapidly bores its way to the bottom of the Pachuca, clearing a path to the bottom of the central agitation column. As soon as this is opened and air has been admitted to the central pipe the spider is raised from the tank and full agitation of the charge proceeds.

Kelly presses are used for the final filtration of the pulp, and Merrill zinc dust precipitation presses are used for the recovery of the gold precipitate. In connection with zinc dust precipitation considerable annoyance has been occasioned by the clogging of the cloths in the gold presses. Filter cloths of several kinds have been tried, and now the lightest and cheapest muslin is used with results no worse than with the more expensive

grades. Another trouble was experienced due to the accumulation of precipitated gold in the pipes carrying the gold solution and zinc emulsion to the presses. The interior of the pipes gradually becomes smaller in area; in one instance a 6 in. pipe having an area of 28 sq. in. was reduced by incrustation of gold slime to less than 4 sq. in. After three months' use 80 ft. of the pipe was cleaned with a recovery of $25,898.

A change has been made in the Merrill precipitating presses, so that the effluent solution passes through open pipes into a launder on top of the presses instead of through cocks as usual. This was to permit of intermittent operation of the presses without draining the solution each time the press was cut out, and thereby allowing zinc to oxidize and the mass to settle in the frame so that the full efficiency of the zinc could be obtained. This change has resulted in raising the gold value of the precipitate from $15 to $25 per pound.

The cyanide plant has now been in operation for a year, and during the month ended August 15, 2010 tons of concentrate were treated at a cost of $2.8764 per ton and an estimated extraction of 97.025%, as compared with the experimental estimates of $3.25 per ton and 96% extraction.

New Smelter for United Verde.

The United Verde Copper Mining Company, Jerome, Arizona, is to build a new and modern copper smelting works. The United Verde has long been a prominent copper producer, largely owned and controlled by W. A. Clark. Owing to the policy pursued by the owners, little has been known about the operations of the company, either financially or technically, and this lack of knowledge has caused many rumors to spread regarding the company. Metallurgically speaking, the United Verde was for many years in the fortunate position of being able to run and make money without due regard to that careful technical direction which is considered essential by our present large producers. Many interesting stories are told of the perplexities of former managers and superintendents, many of whom were better fitted for many other forms of work than smelting; and whether true or not, they illustrate well the ludicrous situations in which a non-technical man might find himself in conducting the smelting business.

A number of misfortunes have befallen the United Verde, such as mine fires and caving ground, the latter having been specially disastrous because the smelting plant had been built on the mine property and the settling of the ground damaged the buildings and necessitated much reconstruction. It has been rumored repeatedly that on account of this condition the company would erect a new plant, and undoubtedly the present decision has been influenced by the misfortunes of the past. It will be remembered that the United Verde expected to make a trial of the Knudsen converting furnace, as reported in these columns some months ago, but this was rendered impossible by reason of a serious cave which ruined the construction work before the process could be tried out.

The new smelter will be situated about six miles from Jerome, and at a lower elevation than the present plant. The men in charge of construction are C. H. Repath and J. B. McIntosh, who were prominently identified with the designing and construction of the Tooele plant of the International Smelting & Refining Co., in Utah.

Tooele Lead Plant.

Construction work at the lead plant of the International Company has been progressing, and all of the excavation and foundation work is reported to be completed. Some of the minor buildings such as the bag house, extension of power house, flues, etc., have been completed, and a large amount of steel is on hand for the construction of the bins. Erection of the furnaces is to proceed at once, and it is hoped that two of the three furnaces will be ready for operations early in 1912. New ore contracts are being made with Idaho and Utah producers, and apparently there will be a large quantity of ore on hand and contracted for when operations begin.

Nevada Hills Mill.

In October the new mill of the Nevada Hills Company, situated at Fairview, Nevada, commenced operations, and is now running at practically full capacity. The plant should be one of the best in Nevada for its construction and design were in charge of engineers who had been connected with other successful plants. According to preliminary reports the mill has shipped its first consignment of gold and is making a recovery of between 92 and 93%.

Company Reports.

Utah Copper Company report for the third quarter of 1911 shows that 1,273,373 tons of ore were treated at both plants, the average grade of ore being 1.4829% copper. The gross production for the quarter was 25,851,456 lb. copper, an average monthly production of 8,617,152 lb. The average cost of copper per pound, after making allowances for smelter deductions, and without crediting miscellaneous income, was 7.56 cents, the lowest in the history of the company. The entire Magna plant was in operation through the whole quarter, but only five remodeled sections of the Arthur plant were in commission. The Magna plant treated about 12,000 tons per day and the Arthur plant 3000 tons. This indicates a final capacity for the Arthur plant of 8000 tons per day when the thirteen sections are completely remodeled.

The Ray Consolidated Copper Company has issued a report showing operations since the interim report of April 1, 1911. Four sections of the concentrator are in operation, running at a capacity of about 800 tons per day each. The sections were designed to handle 1000 tons each daily, and have demonstrated their ability to do so. The gross production is about 2,500,000 lb. per month from ore averaging 1.85% copper. The mill extraction is averaging 69%.

Non-Ferrous Metal Market.

There has been little or no change in the market for non-ferrous metals since our last report. The demand for all continues good and the prices generally are firm, reflecting the active condition of foreign markets.

Copper.—Domestic consumption has not been more than normal, but the demand for export has continued to be heavy. As a result there was a large business at advancing quotations. At our last report Lake copper is quoted at 12¾ to 12⅝ cents, and electrolytic at 12.30 to 12.35 cents.

Lead.—The market is unchanged at 4.10 to 4.12½, St. Louis, and 4.20 to 4.25, New York.

Tin.—A large business has been done in this metal, mostly for future delivery. Stocks are so low that a premium on spot tin may come to pass. November tin was quoted at 42¼ cents per pound.

Spelter.—The scarcity of November spelter was more pronounced than in previous months, but there has been no change in the prices. Business has not been more than normal for immediate needs. St. Louis spelter was quoted at 6 to 6.10 cents and New York at 6.15 to 6.25 cents.

The Iron and Steel Market.

November in the iron and steel market opened with gloom universal and closed with sentiment uniformly optimistic, buying heavier and prices showing a tendency toward firmness.

Sentiment had been uniformly pessimistic for three months when on Oct. 26 the United States Government brought suit against the United States Steel Corporation for its complete dissolution, not simply a dissolution of the Steel Corporation as a holding company, but a dissolution also of the component elements, the consolidations of 1898, 1899 and 1900, which in 1901 were brought under the ownership of the corporation. The immediate effect of the bringing of this suit was a fresh accretion of pessimism, but this made no material

impress upon the market, which was depending upon actual conditions and not upon sentiment for its strength. So thoroughly had the previous pessimism liquidated the situation that the market rather tended to improve.

Suddenly the railroads became rather free buyers. Several large purchases of rails for 1912 delivery were made, and although rails were the only commodity which had suffered no decline, the $28 price established early in 1901 having been maintained since, no question seems to have been raised, and the orders booked, aggregating 200,000 tons or more, appear to have been placed at this price, although perhaps the sales were made with a guarantee against decline. It has long been known that the railroads are much more interested in raising the standard of quality than in depressing the price.

In the closing days of October roads comprising the New York Central system began placing orders for cars, and other car orders soon appeared. The total bookings for November cannot be stated at this time, and the movement is not yet over. In brief, it may be said that in the first half of this year car orders amounted to about 50,000 and in the next four months to about 20,000, while November and December are likely to see a total of between 50,000 and 100,000. When early orders in this movement amounted to 20,000 the large plate mills estimated that the business would be sufficient to give them as heavy operations through December as they had had in September and October; later orders guarantee them heavier operations than formerly.

An incentive for the purchase of cars at this time is their cheapness, prices being lower than at any time since the steel car came into vogue. The railroads, however, are believed actually to need the cars. They have never been known to make large purchases in cars or any other line merely because prices were low, even though it has frequently been urged that in certain cases they would eventually save money by doing so.

Unfilled orders of the United States Steel Corporation increased during June, July and August, decreased 84,668 tons in September and increased 83,011 tons in October. Estimates are that in November a considerably larger increase occurred.

Production of pig iron in November was at a slightly lower rate than in October, when the rate was approximately 25,000,000 tons a year, but there had been a continued increase since July.

Operations in the steel industry as a whole continued at about 75 per cent of full capacity. It should be observed, however, that at the present price level no more than 90 per cent could be expected to be operative, so that the addition of business equal to 15 per cent of capacity would in all probability effect some slight advance in prices.

The general level of finished steel prices, averaging the various commodities approximately according to their tonnage importance, declined about $1 per ton during November, as was the case in both October and September, while the months of July and August together witnessed a decline of about $1 a ton.

Pig iron production in 1911 may be forecasted at about 23,700,000 tons, to fulfill which prediction would require about 370,000 tons more to be made in the second half than the 11,665,706 tons officially reported for the first half.

Pig Iron.

The pig iron market has shown remarkable quietness. Almost as a unit buyers have refrained from making any purchases for the new year and are endeavoring to clean up their yards completely by the end of this year. Purchases have been very light. On the whole, the statistical position of the furnaces is good, as in the main they are reducing stocks. The present situation is fraught with some danger, in that buyers may wait too long and then be forced to come into the market in a body, with the result of prices advancing and an unwarranted increase in production occurring. The most important transaction in November was the purchase of 25,000 tons of Bessemer iron by the Republic Iron & Steel Company from the Ohio Iron & Steel Company, operating a merchant stack at Lowellville, near the company's Bessemer steel plant at Youngstown, Ohio. The price paid was $14.25 and delivery was to be completed in a few weeks, the iron being shipped from stock. Brokers had sold a few odd lots of Bessemer at $14, a cut price, and the Republic purchase served to stiffen the market. The market stands quotable as follows: Birmingham, $10 for No. 2 foundry; delivered Philadelphia, $15 for No. 2X, $14.50 for basic, $20 for standard low phosphorus; f. o. b. Chicago district furnaces, $14 for No. 2; $14 f. o. b. valley furnaces (90 cents higher delivered Pittsburgh); Bessemer, $14.25; basic, $12.50; No. 2 foundry, $13.25; forge, $12.75; malleable, $12.50 to $12.75.

Steel.

The market in billets and sheet bars has been a decidedly open one, with special prices made on the few transactions which have occurred. The general level may be quoted at $20, Pittsburgh, for Bessemer and open-hearth billets and sheet bars. Rods are quotable at $26, Pittsburgh.

Finished Material.

Open quotations on bars and plates declined $1 a ton during the month, and on wire products and structural shapes $2 a ton. At the present level the market is firmer all along the line than it has been at any time for months. There may be some further declines, but in general the present level is likely to hold, and some slight advances may occur before the end of the year. This is the first time since April that an advance in any steel commodity is a distinct possibility.

We quote regular open market prices as follows, these prices being occasionally shaded:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill, except Colorado.

Plates, tank quality, 1.10 to 1.15 cents, Pittsburgh.

Shapes, 1.10 to 1.15 cents, Pittsburgh, for beams and channels, 3 to 15-in., inclusive, zees and angles 3 x 3-in. and larger.

Steel bars, base, 1.05 to 1.10 cents, base, Pittsburgh, for prompt delivery, 1.10 to 1.15 cents on contracts for first quarter; 1.20 to 1.23 cents, Chicago, 1.10 to 1.15 cents, Buffalo.

Iron bars, 1.25 cents, Pittsburgh; 1.20 cents, Philadelphia; 1.15 to 1.20 cents, Chicago.

Wire nails, $1.55, base; plain wire, 1.35 cents, base.

Black sheets, 28 gage, 1.85 cents; galvanized, 2.85 cents; blue annealed, 1.40 cents; painted corrugated roofing, $1.35 per square; galvanized, $2.50, f. o. b. Mahoning valley mills.

Tin plates, $3.40, Pittsburgh, for 100-lb. cokes.

CORRESPONDENCE.

Manufacture of Pig Iron in the Electric Furnace.

To the Editor of Metallurgical and Chemical Engineering:

Sir: In the paper of Mr. E. Odelberg in your October issue on electric pig iron and the manufacture of steel therefrom much stress is laid on the fact that when a low-silicon, low-carbon iron is produced which is filled with blow holes the inner surfaces of these blow holes are white, showing entire freedom from oxidation (page 509).

It is pointed out that an entirely different condition prevails as to the surfaces of blow holes in similar iron made in the blast furnace, this showing all the colors of the rainbow; the oxide colors familiar in hardening and tempering steel. This difference is held to prove the entire absence of an oxidizing condition and of oxides in the hearth of the electric furnace, and their presence in that of the blast furnace.

This would be interesting and important if it were true, but, as a matter of fact, the writer has made many tons of white iron full of blow holes (which is known under the trade name of "Spongy Six") in a charcoal blast furnace, and of the many thousand blow-hole surfaces examined, none ever showed any

color except silvery white just as described by Mr. Odelberg.

The real truth of this matter is that these blow holes are produced by the action of imperfectly reduced ore in the bath reacting on the carbon (presumably that dissolved in the iron) and forming thereby carbon monoxide. This, of course, is unable to exert any oxidizing influence under the conditions prevailing in the bath, and the interior of the blow holes which it forms are necessarily free from oxide and perfectly white in consequence. The carbon monoxide does not escape from the iron because the temperature prevailing in the hearth under such conditions is extremely low and the iron is too pasty to permit the bubbles of gas to rise through it.

The electric furnace will come into its own in time, probably in Sweden and on our Pacific Coast sooner than elsewhere on account of the presence in those regions of iron ore and water power in abundance, and the scarcity of mineral fuel. But it is as well to let its growth be a natural one, unhindered by extravagant claims which tend to make the claimants, and sometimes the process, ridiculous.

As the ores of iron are all oxides and gradually reduced as they descend through the furnace, it is always possible for cold or irregular operation to introduce into the hearth imperfectly reduced material containing oxygen, quite irrespective of whether the source of heat for the reduction process be the combustion of carbon or an electric arc. This is so rudimentary that it should not require statement, but apparently it does.

JOHN J. SMITH.

Abandoning the Terms Carbonic Oxide and Acid.

To the Editor of METALLURGICAL AND CHEMICAL ENGINEERING:

Sir: Some time in the middle of the last century chemical nomenclature was reformed by progressive chemists, so as to make terms correspond more nearly with the facts; the formula for water, for instance, was thus changed from HO to H_2O.

It was, I believe, at that time that the consistent applications of the terminations -ous and -ic were introduced for the lower and higher oxides respectively, in order that the names should correspond more nearly with the compounds. According to this nomenclature the lower or monoxide of carbon, CO, should for consistency sake be called carbonous oxide, and the higher or dioxide, CO_2, be called carbonic oxide, even though there may still be some uncertain theories about the valency of carbon.

Notwithstanding these very desirable reforms, and the length of time since they were introduced (more than a generation ago), one still finds the term carbonic oxide used by some for the *lower* oxide CO, the higher one being called by them "carbonic acid," although it is in no sense an acid, any more than the oxide of sulphur, SO_3, is sulphuric acid. These were the older terms used before the reform of the progressive chemists. A prominent metallurgist even went so far as to say that he had never heard or seen the term carbonic acid used for anything but CO. I believe it has been the custom of careful writers for many years to abandon the older nomenclature.

At present the state of affairs is that when one hears or reads the term carbonic oxide one is at a loss to know what the author meant, unless the context determines it. In view of the fact that in metallurgical reactions it generally makes a very great difference whether CO or CO_2 is meant, this confusion is most unfortunate, and as it seems to take more than a generation to bring about a complete reform in terms, and that it therefore seems hopeless to do so at the present time, it would, in the opinion of the writer, be better to abandon the term carbonic oxide entirely (and of course also the positively incorrect term carbonic acid for CO_2) and use only the terms carbon monoxide and carbon dioxide, about which there can be no doubt; this I believe is what is taught in modern up-to-date colleges. It would be better still to use the symbols CO and CO_2.

There is no objection to using the term carbonous oxide for CO, as there should be no confusion. After several generations have passed by the term carbonic oxide might perhaps again be introduced so that the nomenclature concerning carbon will be consistent with that used for other elements, like cuprous and cupric, ferrous and ferric, stannous and stannic, etc.

Philadelphia, Pa. CARL HERING.

Induction Furnace Design.

To the Editor of Metallurgical and Chemical Engineering:

Sir: In the report of the Toronto meeting of the American Electrochemical Society, in your October issue, page 536, Mr. FitzGerald is reported to have said, in reply to an inquiry of Dr. Waterhouse, that the Roechling-Rodenhauser furnace for such sizes would be found greatly superior to other forms of induction furnaces.

As I and certainly many other readers of your journal would be greatly interested to know *in which respects* a Roechling-Rodenhauser furnace of this size would be greatly superior to mine and other induction furnaces, I ask you to put this inquiry to Mr. FitzGerald into your columns.

Kristiania, Norway. ALBERT HIORTH.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir: Unfortunately I have not got a copy of the stenographer's report of the discussion following Mr. Hiorth's paper at the Toronto meeting of the A. E. S. My remembrance is that Dr. Waterhouse asked for someone's opinion, specifically mentioning my name. I think my reply was substantially as reported, but it must, of course, be treated as an opinion based on my own experience, *so far as it goes.* This is a different thing than an opinion based on an investigation made for the purpose of comparing all types of induction furnaces. I think it is only fair to Mr. Hiorth and his furnace to explain this.

My reply to Dr. Waterhouse was based on the experience I have had in the construction and running of various induction furnaces, on what I have seen of the running of an 8-ton Roechling-Rodenhauser furnace and on what I have read of other induction furnaces, such as that of Mr. Hiorth. There can be no doubt at all as to the superiority of the R-R furnace over various induction furnaces I have had personal experience of. I have seen the R-R furnace running in a most satisfactory way on cold scrap and hot metal from Bessemer converter and turning out high-quality steel.

As to the Hiorth furnace—so far as I know the largest one that is regularly working has a capacity of 5 tons, and according to Richards' report last year[1] was melting high-grade materials used in the manufacture of metal of crucible-steel quality. According to the same authority[2]: "At Mr. Hiorth's plant the induction furnace is used for *melting down high-quality metal* and making a high-quality steel, *but he also uses an electrode furnace for working up scrap.*"

The italics are mine. So far as my information goes the R-R furnace seems up to the present to be the best tool, working by induction, for the manufacture of steel.

FRANCIS A. J. FITZGERALD.

FitzGerald and Bennie Laboratories, Niagara Falls, New York.

An Instructive Experiment on the Flow of Heat.

To the Editor of Metallurgical & Chemical Engineering:

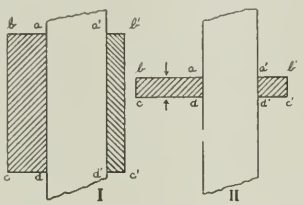
SIR.—In the communication from Mr. Carl Hering in the November issue, entitled "An Instructive Experiment with the Flow of Heat," some conclusions are drawn which rest on a misconception of the conditions involved.

Without going into the mathematics of the conduction and

[1] Transactions of A. E. S., vol. 18, p. 200.
[2] *Metallurgical and Chemical Engineering*, Oct., 1911, p. 545.

radiation of heat, it should be pointed out that Mr. Hering is not warranted in arguing from the fact that more heat is lost through the long than through the short end of a long metallic *rod* held in a flame, that, therefore, more heat will be lost through a thick wall than through a thin one.

Now the cooling in the former case takes place not merely at the ends of the rod, but also—and here to a far greater extent—at the cylindrical surface; while in the case of a furnace or a "tank containing a red hot liquid," the *wall* may be considered as an *extremely short* thick rod of which the cooling surface consists to all intents and purposes of the end only, the remaining surface being comparatively small. In other words, the radiation conditions in the wall are exactly the reverse of those in the rod.



CONDITIONS OF WALLS AND ROD.

In Fig. I, which represents the wall conditions, the total radiation surface is practically the same with the thick wall as with the thin one, being composed chiefly of the end surfaces of which *b c* and *b′ c′* are the diameters. Consequently the amount of heat lost will depend on the rapidity of the *conduction* in the walls, which will evidently be lower for the thick one.

In Fig. II, which represents the rod, on the other hand, the cylindrical surfaces of rotation about *a b* and *a′ b′* make up practically all of the total—and in this case it depends on the relative magnitudes of the radiation and of the conduction coefficients of the material whether more heat will be lost through the long or the short rod. Thus, with a good conductor like copper, Mr. Hering finds the long rod loses more heat; with a poorly conducting substance, like pipe clay, the reverse seems to be true.

The important practical conclusion, however, which needs to be emphasized here, is that noted above, namely, with any material, a thick wall will insulate better than a thin one. Thus, Mr. Hering's experiment does not contradict furnace practice.

B. S. Lacy.

Perth Amboy, N. J.

* * *

To the Editor of Metallurgical and Chemical Engineering:

Sir: Mr. Lacy would do less injustice to others if he would read over more carefully what he criticises with such rather strong terms as "misconception," "not warranted," etc., before publishing such criticisms.

If he will read over my communication again, he will find that the chief point in it (which he seems to have missed) was to call attention to the surprisingly high virtual thermal resistance which exists at a red-hot surface exposed to the air. In this there are no "misconceptions of the conditions involved"; at least he has not been able to point out any.

In his second paragraph he misrepresents me if he is trying to make it appear in the last clause as though I had concluded that the loss through thick furnace walls in general was greater than through thin ones. The term copper is not mentioned by him. What I did say applied only to copper walls, and I distinctly stated that with clay, and therefore probably with other refractories also, the results would very likely be reversed.

I did and still do maintain, and Mr. Lacy will probably not deny it, that with such a monstrosity as a furnace with relatively very thick walls of copper, the leakage per sq. ft. would be greater through the walls than through the top surface of the charge, if liquid. This would certainly be true when the cross sections and exposed surfaces had the same relations as in the experiment, and this is all that I claimed. Such a monstrosity was used by me only as an illustration in an argument, not as a practical case. When the walls are relatively thin, I should want to try an actual experiment before drawing conclusions. The mathematical relations are complicated and the final result depends on several relations combined (see this journal April, 1911, p. 189). Until Mr. Lacy can show proof to the contrary, I therefore deny that I am "not warranted" in drawing the conclusion which I did.

Mr. Lacy's diagrams are not very clear, but as I understand them he has omitted the flare of the walls, which, of course, is one of the chief features implied by the term relatively thick walls. A furnace with relatively thick copper walls is, of course, a practical absurdity, and was used by me merely as an extreme case, in order to bring out the point more clearly.

In his last paragraph I differ with him when he says "any" material, as with relatively thick walls of copper his conclusions are not correct. In this paragraph he again misrepresents me, if he intends to imply by it that I had claimed that my experiment contradicted furnace practice. My original article clearly states that with clay the results would probably have been the opposite to those with copper, and therefore would confirm furnace practice. In fact, it was to try to satisfy myself that thick walls of insulating material were justified, that I made the experiment.

I desire to add here, however, that I believe that increasing the thickness of walls may easily be overdone, and my experiment tends to strengthen this view. Our data of the necessary physical constants is too meager to determine this mathematically. A furnace which I recently designed remained positively cold on the outside, and it is therefore a question whether the extra thickness was warranted. I am inclined to believe that the thermal resistance of fire-brick work is higher than we now suppose, probably owing to the joints, particularly when the bricks are set parallel to the walls so as to make as many joints as possible, as was the case in this furnace.

Philadelphia, Pa. Carl Hering.

Presidency of the American Electrochemical Society.

To the Editor of Metallurgical and Chemical Engineering:

Sir: As the Eighth International Congress of Applied Chemistry will be held in this country in the year 1912, on which occasion many foreigners are expected to visit this country, and as electrochemistry and metallurgy are represented as sections of this congress, the American Electrochemical Society should, in our opinion, take this into consideration in the selection of its president for the coming year, as he will be expected to represent this society at the official and social functions.

Dr. Charles A. Doremus is in our opinion especially well qualified for the presidency for that year, and would do credit to the society. We hereby recommend to the members of this society that they nominate him in the ballots which they will soon receive. He has an international reputation as a chemist, is a graduate of Heidelberg University, under Bunsen, is versed in the duties of international functions, is acquainted with many of the foreign visitors, having been Official Delegate from the State Department to the Second International Congress, and made chairman of the American Committee of Organization by Moissan, for the Fourth Congress, is conversant with the three principal languages, is able to devote the necessary time to the duties which would be imposed upon him, has been a Director and Vice-President of this society, a member since its organization, was one of the eleven original organizers taking a very active part in starting the society, and in the work of the New York Section, of which he has been Chairman.

Charles S. Bradley, Henry S. Carhart, Carl Hering, P. Heroult, Alois von Isakovics, W. R. Whitney.

Sizing, Diametric and Volumetric, with and without Screens.

By B. F. RICE.

In a large number of the manufacturing industries and in the treatment of ores it very frequently becomes necessary to grade dry and granular material according to size or bulk of particle. This constitutes one of the most serious problems with which engineers and mill men have had to deal.

Up to the present time machines working on two separate and distinct principles have been used for this work. I refer, first, to the different forms of classifiers which classify the material according to its weight, putting a large particle of low specific gravity with a smaller particle of high specific gravity. This is accomplished by the introduction of an hydraulic or pneumatic floating medium. A grading of this kind can be made also on dry material without the use of a floating medium. A machine for work of this character was developed by us, but was discarded for want of a promising field.

Second, different forms of screens, such as the impact, shaking, belt, inclined, trommel and gyratory screens, which have met with only a limited degree of success. This is probably due to a number of different causes, but more particularly to the high cost of maintenance, together with a rather low efficiency.

It has been our aim in the development of the McKesson screenless sizer[1] to provide a cheaper and more efficient method of sizing dry granular material than is afforded by any screening device. This was no small undertaking, and as we were the originators of the idea, as is clearly shown by the United States patent records, we have not only developed a new machine but an entirely new principle, which appears to the uninitiated to be contradictory to the known laws of gravitation. But like a great many other radical departures from old established laws, it is very simple when once understood.

Our policy has been to bring this machine to such a high state of development that its commercial success would be assured before it was presented to the public. We have passed our greatest expectations and I am safe in making the assertion that the sizer is well within the commercial limits of 90 per cent of the industries requiring closely sized material. There are, of course, special cases in this field as in others, that will require especially designed machines to successfully meet their requirements, but the fundamental principle of this machine is such that there is no question but that its development will eventually reach a point far beyond anything ever considered by the advocates of close sizing.

Comparing the work of this sizer to that of a screen, we find, to begin with, that the principle of the sizer is as far removed from that of a screen as is the principle of a classifier from that of a screen, but in the opposite direction. In other words, a screen is intermediate between the two principles. A classifier grades according to weight regardless of size, volume or bulk of particle. A screen grades according to the diameter of a particle and takes cognizance of two dimensions only. The McKesson sizer grades according to the bulk or volume of particle, regardless of its shape, and is affected by specific gravity only to a very limited degree, and that directly opposite to the effect of specific gravity in a classifier. This will be shown by the chemical analysis of the screen oversize and undersize in the products tested.

It is obvious that if a product, graded according to bulk or volume of particles, be tested on screens, there will naturally be a wide range between the two extremes in this product. That is, the difference, between a scale and a cube as measured by a screen which takes cognizance of two dimensions only will cover a relatively wide range, while in reality the volume of the particles may be identical.

[1]See October issue of this journal, p. 553.

The reverse is true of a screened product, for, while the product may be between screens of only a few points difference, the ratio of volume of largest to smallest particle may be several times the ratio of the cubes of the apertures of the screens limiting the product.

Quoting Dietz & Keedy, METALLURGICAL AND CHEMICAL ENGINEERING, November issue, page 578: "In the practical application of close and accurate screening we have adopted as a standard a series of thirteen screens between 20 and 200 mesh conforming to a harmonic series, thus producing fourteen products including oversize and fines. The average volumetric ratio is 2:1, while the efficiency, when measured by the screen actually limiting the grades, averages better than 90 per cent, the only drop under 90 per cent being in the —150 + 200 mesh grade, which, with the screen area usually exposed, varies between 80 per cent and 85 per cent. The series, as developed by us and used as a standard, is the following, in which screen cloths have been selected of apertures conforming to the law of cubes."

Mesh.	Apertures, Inches.	Ratio of Volumes of Largest to Smallest Particles.
20	.0410	1.722:1
24	.0342	1.722:1
28	.0282	1.783:1
34	.0229	1.867:1
42	.0183	1.959:1
50	.0145	2.012:1
62	.0116	1.953:1
74	.0089	2.214:1
86	.0068	2.242:1
109	.0054	1.996:1
125	.0041	2.284:1
150	.0032	2.103:1
200	.0026	1.865:1
Average		2.000:1

To show the fallacy of this system let us take the product between the screens 20 and 24 mesh, and assume a particle in the shape of a flat scale that will just correspond to the square of the aperture of the 24-mesh screen, but having a third dimension or thickness of, say, 0.0026 in., which is the aperture of the 200-mesh screen. It is obvious that this particle is bound to report between the screens 20 and 24 mesh, and will represent the smallest particle in that product with a volume of 0.000003041064 cu. in. Now take a cube that will correspond to the cube of the aperture opening of the 20-mesh screen, which is 0.000068921 cu. in. Here we have a volumetric ratio of largest to smallest particle of 22.66:1 instead of 1.722:1 as claimed. Take the next two screens, 24 and 28 mesh, and figuring on the same basis we get a ratio of 19.37:1 instead of 1.783:1. It is quite true that in some cases an ore is found that will have a fracture favorable to the application of the law of cubes, but unfortunately these cases are very much in the minority.

From the foregoing it is evident that in a product graded by screens it is a physical impossibility to get anything like the ratios claimed in all but a few rare cases where all the material will break in the form of cubes, and it is equally impossible to test by the use of screens with any degree of accuracy the efficiency of a machine which grades according to volume or bulk of particle.

Dietz and Keedy have taken the results of a test on the McKesson sizer, and have calculated the volume ratio of largest to smallest particles according to the aforementioned law of cubes. Taking, for example, their calculation for product No. 7 which apparently shows the largest ratio, their figures are as follows:

Product No.	Screen Mesh.	Efficiency per Cent.	Volume Ratio of Largest to Smallest Particle.
7	—50 + 82	90.63	6.20:1

A very careful test was made on this product to determine the actual, not theoretical, ratio of largest to smallest particle in this product. It was screened to give two products, viz., —50 + 66 which is the largest grade, and —74 + 82 which is the first grade in product No. 7. Equal volumes of these two grades were then taken by displacing 0.02 cc. of water, and the number of grains in each grade was accurately counted. There were 2200 particles in the coarse grade and 4105 particles in the fine grade. Now we have something definite to work upon and we find that there is an actual volume ratio of largest to smallest particle of 1.893:1 instead of 6.20:1 as calculated from the law of cubes. This accounts for the lapping of products so seriously objected to and shown graphically by Dietz and Keedy on page 578. A careful examination of the different products under a powerful microscope reveals the same condition throughout all the products.

Taking the case of product No. 15:

Screen.	Per Cent.	Efficiency.
—82	.....	
+97	6.25	
+116	18.75	
+139	43.75	68.75
+172	25.00	
—172	6.25	

We have here a product ranging from +97 to —172 which, when examined under the microscope reveals the scaly nature of the grades +97 and +116. In this case a scale corresponding to the square of the aperture of screen 97, which is 0.0058, and having a third dimension or thickness of one-seventh that amount, will give a particle +97 mesh which belongs in a 200-mesh product.

Now to take up the effect of specific gravity: From the analysis given herewith it is evident that whatever effect specific gravity has, it manifests itself in such a way as to remove all semblance of a doubt as to reversal of the accepted fundamental principles governing agitated material.

The attempt to show that the McKesson sizer is a dry classifier has led Dietz and Keedy into the error of predicting certain things regarding the nature of the products of this machine. They say, speaking of classified products, "In each grade it will be found that the lightest mineral reports to the greatest extent in the coarsest grade, while the heaviest mineral will be found to the greatest extent among the finer particles of the particular product under consideration. This same action is to be expected in a machine of the McKesson type."

In other words, they believe that if one of the McKesson products be subjected to a screen analysis the coarsest portion will contain the most gangue and light mineral, while the finest portion will carry the most mineral of high specific gravity, and the intermediate sizes will contain proportionate quantities of the minerals according to their specific gravities. They suggest a chemical analysis of the products from a mixed sulphide ore as being the best way to prove their contention.

With this object in view I have taken a product from such an ore and have had its several screen grades analyzed. The results are exactly opposite to what Dietz and Keedy predict, and show that in no sense is the product a classified material. In fact the analysis shows conclusively that the product is a sized material.

Screen Analysis on Size No. 8 of a Zinc-Iron Crude Ore.

Screen.	Per Cent.
—40	.....
+50	12.50
+66	75.00
+74	9.36
—74	3.12

According to Dietz and Keedy's anticipation that the product is classified material we should find a preponderance of gangue and lighter minerals in the coarse grade, and a similar excess of iron and zinc in the fine grade. If their contention is correct we ought to find the highest percentage of insoluble in the +50 grade, with diminishing percentages in the finer grades. Similarly, iron and zinc as the heavier minerals should appear in the different grades in increasing proportions. But such is not the case—indeed the facts are the reverse of Dietz and Keedy's assumptions, and disprove their idea that the McKesson machine is a dry classifier. Its different products are sized, and this has been proved many times by treating the most complex mixtures, the components of which were widely separated in specific gravities.

Denver, Col.

Chemical Analysis of Size No. 8 and Its Several Screen Products.

	Zinc, per Cent.	Iron, per Cent.	Insoluble, per Cent.
Size No. 8.....	29.20	12.00	15.56
Products of Size No. 8.			
+50	25.10	10.05	2.68
+66	32.90	13.30	8.90
+74	25.80	7.80	28.64
—74	14.80	5.40	48.08

Flameless or Surface Combustion.

A New Method of Using Gaseous Fuel for Heating and Steam-Raising Purposes.

By John B. C. Kershaw.

Sir Humphrey Davy, during his remarkable researches upon flame in the year 1816, discovered that under certain conditions a mixture of a combustible gas and air (in proper proportions) when forced into contact with a heated porous and refractory material or metal could *be burned without the production of any flame*, yet with the maintenance of a very high temperature. For various reasons this method of employing gas has made little progress, as regards its industrial application for heating or steam-raising purposes, and it is only to-day, nearly one hundred years after the first discovery of the method, that flameless or surface combustion has been developed into a practical and valuable industrial process.

Professor William A. Bone (F.R.S.), professor of applied chemistry in the Leeds (England) University, and Mr. C. D. McCourt have for some time past been engaged in the study of the theoretical and practical side of the subject, and have recently patented a process and apparatus, by means of which this method of combustion can be applied either for heating crucibles, furnaces or steam-boilers, with efficiencies that approach 95 per cent. of the actual heat value of the fuel gas used. The practical importance of this new method of using gaseous fuel for heating purposes *depends altogether upon this question of efficiency*. If, as stated by Bone and McCourt, thermal efficiencies of 90 per cent to 95 per cent can be attained in practice with all varieties of fuel-gas, whether for steam-raising or for furnace operations, the future of the new process would seem to be assured.

The one drawback attending the gasification of solid fuel in the past has been that the gas when employed for heating purposes (owing to its low radiant heat power) only yielded from 50 to 70 per cent of its heat to the object that required it, and that the direct thermal loss (apart altogether from the heat lost during gasification) amounted, therefore, to from 30 per cent to 50 per cent of the total heat value of the fuel.

The gasification of solid fuel and the employment of the gas for heating purposes have therefore only been adopted in the past under special conditions, when a very high temperature was required (as, for instance, in the glass and steel industries, where regenerative gas furnaces are employed), and it has been most unusual to find any steam-boiler or ordinary furnace heated by fuel gas, on account of the low thermal efficiencies obtained with this method of heating.

Bone and McCourt by rendering it possible to produce the greater portion of the heat as radiant heat at a very high temperature, and to attain heat transfer efficiencies of 95 per cent with gaseous fuel, have, however, brought this method of heating within the domain of practical application in the manufacturing industries. In the future one may expect to see a gradual extension of the plan of gasifying the solid fuel at the pit's mouth, recovering the ammonia and more valuable organic by-products of the destructive distillation of the coal in adjacent chemical works—and then selling the residual gas to manufacturers, either for power generation or for heating purposes. This method of utilizing fuel is undoubtedly the most scientific and the most promising, since it makes the best use of all the constituents of the fuel, involves the least waste and indirectly helps to solve the smoke problem.

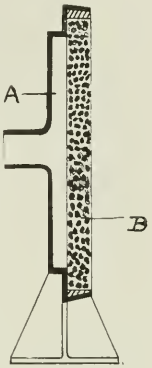


FIG. 1—DIAPHRAGM FOR SURFACE COMBUSTION.

The readers of this journal may therefore be assumed to have a special interest in it. In the following notes the writer will give some details of the apparatus used for the trial results obtained with the new method.

Fig. 1 shows in sectional elevation the simplest form of apparatus for applying surface combustion to heating purposes. *A* is the feed-mixing chamber through which the gas and air pass; *B* is the diaphragm at the surface of which combustion occurs. The amount of air used is in slight excess of that required for *complete* combustion of the gas.

The air and gas are forced into the mixing chamber, and

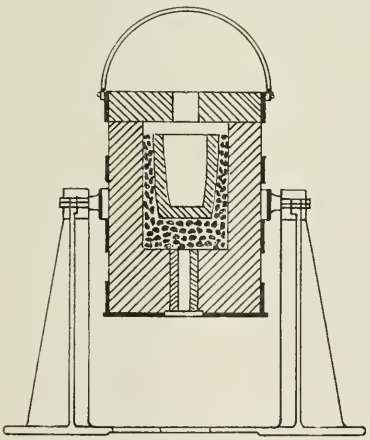


FIG. 2—APPLICATION OF SURFACE COMBUSTION TO A CRUCIBLE FURNACE.

therefore through the diaphragm, at a speed greater than that mixture; otherwise what is known as "backfiring" will occur. The prevention of "backfiring" and the choice of a suitable refractory material are in fact the two difficulties which have hindered the successful operation of this method of burning gas.

The gas-mixture burns without flame within a layer ⅛ in. (3 mm) in thickness, at the surface of the diaphragm, and maintains it in a high state of incandescence. The other parts of the apparatus are cold enough to touch with the naked hand, and the slightest variation in the pressure of the gas at which the explosion wave is propagated through the gas-supply causes an instant change in the degree of incandescence of the diaphragm, showing that there is no reserve of heat in the body of the same, and that the temperature attained by this method is under instant control. An incandescent surface of this type can be used for concentrating heavy liquids similar to solutions of sodium silicate and sulphuric acid by surface evaporation.

Fig. 2 shows the new method applied to the heating of a

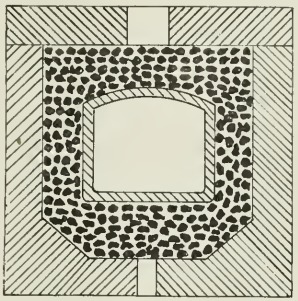


FIG. 3—APPLICATION OF SURFACE COMBUSTION TO A MUFFLE FURNACE.

crucible tipping furnace and Fig. 3 to a muffle furnace. In these cases a bed of well-graded pieces of the refactory material surrounds the crucible or muffle and the explosive gas-mixture is blown into this bed through the opening shown in the diagrams at the bottom of each furnace. As the gas-mixture impinges upon this incandescent bed, combustion is completed almost instantaneously, and the products of combustion as they rise through the upper portion of the bed rapidly impart their heat to the material composing it.

A charge of cast iron can be melted in one of these crucibles in ten minutes, starting with cold metal, and the only limit at present as regards temperature is that imposed by the melting or decomposing point of the refractory materials employed. Carborundum (silicon carbide), which has been widely used for the purposes of a furnace lining and is considered highly refractory, is decomposed quickly into CO_2 and SiO_2 in the

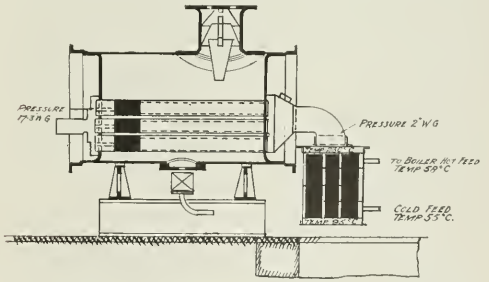


FIG. 4—APPLICATION OF SURFACE COMBUSTION TO STEAM RAISING.

interior of these furnaces, indicating that a temperature of 2000° C. has been approached.

With mixtures of coal gas and air, or water gas and air, in theoretical proportion, injected into the furnaces at a maximum speed, temperatures have been attained which no material hitherto used for furnace or muffle construction can long withstand. Gases of lower calorific value, such as coke-oven and blast-furnace waste gases, of course yield lower temperatures. No unburnt gas escapes by this method of combustion when once the correct proportions of gas and air have been properly adjusted, and, therefore, the maximum

temperature possible to the gas is always obtained unless, as already explained, an excess of air is purposely admitted to reduce the temperature.

Fig. 4 indicates how this principle of combustion can be applied to steam-raising purposes. The experimental boiler shown in diagrammatic longitudinal section in this illustration is provided with ten fire-tubes, each 36 in. in length and 3 in. in diameter. Each tube is packed for ⅞ of its length with the refractory material, broken and graded into pieces of suitable size.

At the front end of each tube is a fireclay plug provided with a central hole ¾ in. (18 mm) in diameter. This plug serves the double purpose of keeping the front end of the boiler cool and of providing a suitable aperture through which the explosive gaseous mixture can be injected at a speed very much higher than the speed of "backfiring."

Attached to the front plate of the boiler is a mixing-chamber of special design. The economizer at the rear end of the boiler is provided with nine tubes, each 3 in. in diameter and 12 in. in length, and packed as before with the refractory material, suitably graded.

Under working conditions this experimental boiler showed *complete combustion* of the gaseous mixture before it had traversed half the length of the fire-tubes. The central portion of the refractory core of each tube was maintained at a very high temperature; this was rapidly transferred to the water in the boiler by radiation.

As regards the rate of combustion, about 1000 cu. ft. of coal gas have been burned per hour, in this small boiler. The temperature of the exit gases is from 60° to 70° C. higher than that of the water in the boiler, but the economizer intercepts the larger portion of this heat, and reduces the temperature of the exit-gases to less than 100° C.

Sixty-five per cent of the total evaporation of the boiler occurs over the first foot length of the fire-tubes. This causes violent ebullition and rapid circulation, the latter being an important factor in obtaining rapid heat transmission.

In an ordinary Lancashire boiler the rate of heat transmission is equivalent to the evaporation of 6 to 7 lb. of water per square foot of heating surface per hour; in the locomotive boiler this is increased to 10 lb.; whereas in the boiler just described it rises to 20 lb.

The following are figures obtained during a test run of the boiler:

The amount of coal-gas fed into the boiler, expressed as dry gas at 0° C. and 760 mm, was 0.96 cu. ft. per hour. The net calorific value of the gas was 562 b.t.u. per cubic foot—the total heat supplied to the boiler in terms of the net calorific value of the gas was 550,800 b.t.u. per hour.

The actual amount of water evaporated was 450.3 lb. per hour, which in terms of water evaporated from and at 100° C. would be 550 lb. per hour. The actual heat transmitted to the water was $450.3 \times 1172 = 527{,}800$ b.t.u. per hour. The ratio between the heat transmitted to the water and heat supplied was, therefore, $\frac{527{,}800}{559{,}800} = 0.943$.

The pressure under which the gas-mixture is blown into the boiler is equal to 17.3 in. (water gage), while the exit gases escape to the economizer at a pressure of 2 in. The resistance to the flow of gas, offered by the fire-clay plugs and the packing of refractory material in the fire-tubes, explains this steep pressure gradient. According to the inventors of this system, the actual power expended in drawing or forcing the gases through the boiler and economizer is only 4 per cent of the total power generated. This reduces the net efficiency of the method to 90 per cent, which is still considerably higher than that obtained with coal or gas-fired boilers.

Another advantage of this system is the rapidity with which steam can be raised. Starting with the ten-tube boiler all cold it is possible to obtain steam at 150 lb. per square inch pressure in about twenty minutes, and to continue the high rate of evaporation indicated—viz., 20 lb. of water from and at 100° C. per square foot of heating surface per hour—so long as the gas supply is properly maintained.

If, in order to meet an "emergency" demand for steam, there is occasion to force the boiler so as to obtain an evaporation of say 30 lb. of water (from and at 100° C.) per square foot of heating surface per hour, all that is necessary is to force more of the gaseous mixture through the boiler tubes, and to sacrifice 5 per cent in efficiency.

A final advantage of the system, as applied to steam-raising purposes, is that in a large boiler the tubes could be arranged in groups, and that the gas mixture could be cut off, or put on to, any group of tubes, according as the demand for steam was diminished or increased.

This elasticity of the system is claimed by the inventors as of great value and advantage in those manufacturing industries where a widely fluctuating load has to be carried by the boilers. A large boiler fired on this system is now being erected at Leeds by the Radiant Heat Syndicate, which has taken over the industrial development of the Bone and McCourt patents, and the test results obtained with this boiler will be awaited with great interest by all power engineers. As already stated, this novel method of combustion seems to open up a new era of progress in the application of gaseous fuel to manufacturing and industrial purposes.

Direct Smelting of Argentiferous Copper Ore at Rio Blanco, Chile.

By Ignacio Diaz Ossa.

The ore deposit known as Cristobal Colon occurs almost on the summit of the chain of mountains forming the left side of the Rio Blanco valley, and is situated 5 km from a station on the Transandino Railroad. The deposit consists of a narrow vein of high-grade copper ore, carrying high silver values, crossing three thick beds of mineralized volcanic turf.

The vein was worked by a day-level until the first bed was cut; a shaft was then sunk at the junction of the vein with the bed, when two other beds were discovered. These have been attacked along the face. During the year the output amounted to 1800 tons.

The ore is raised by winches to the level of the adit, which is about 25 m above the lowest bed. Near the mouth of the adit the ore is trammed along the edge of a precipice for about 200 m to the upper station of a ropeway connecting the mine and smelter, the latter being erected on the banks of the river. The ropeway is in two stages, and usually remains idle from 1 to 3 p. m. daily, owing to the high winds which prevail at that time.

The beds, the total thickness of which amounts to about 15 m, contain on an average 5 per cent copper. Rapid hand sorting raises this percentage to 8, and the value of the silver to about 75 pesos per ton. (One peso = $0.23 gold.)

Smelting Operations.

The blast furnace, which is of the cylindrical type, is defective and not built on modern lines. Its daily capacity is 20 tons. The dimensions are as follows: Height from charging floor to tuyeres, 2.17 m; diameter of cylinder at charging floor and tuyeres, 1.10 m; number of tuyeres, 12; diameter of tuyeres, 0.17 m; length of water jacket, 1.53 m; upper part, covered with brick, 0.87 m; height from tuyeres to crucible, 0.23 m. The blast is provided by a Root blower operated by two Pelton wheels producing 25 hp. A short channel conducts water from the Blanco to the wheel house.

On taking charge of the smelter I did away with the old heavy forehearth on rails, and made an inner crucible 1.40 m deep, with a diameter at the floor of 0.58 m; and another 1.10 m below the level of the tuyeres. I also dispensed with the

lip for the discharge of slag, making an arrangement with firebrick which gave much better results.

The following figures show the percentage composition of the ore: Silica, 45 to 50 per cent; lime, 8 to 12; iron, 7 to 11; sulphur, 4 to 6; alumina, 7 to 10; lead, 1.5 to 3; copper, 8.9 to 12; arsenic, 0.1; silver, 700 to 800 grams per ton.

As the only flux obtainable was a dolomite limestone containing 8 to 10 per cent heavy spar, old reverberatory slags had to be used as well. Analyses showed them to have the following percentage composition: Silica, 45 per cent; iron, 31; lime, 12; alumina, 8; copper, 2.5; silver, 150 grams per ton.

The ore, flux and slag were mixed in such proportions as to form a bisilicate of lime. The resulting slag analyzed as follows: Silica, 50 to 52 per cent; lime, 26 to 30; iron, 10 to 12; alumina, 6 to 9; copper, 0.15; silver and lead, traces only. The slag ran well at a temperature exceeding 1250 deg. C. Both iron and sulphur oxidized easily, the final products being matte containing 75 per cent copper, and copper in bars. About 25 per cent of the copper was obtained in bars of 90 per cent metal. The copper matte contained from 7 to 9 kg of silver per ton.

In ten days we smelted 54,530 kg of ore carrying 12 per cent copper and having silver to the value of 60 pesos per ton; 12,910 kg of ore of 7 per cent copper and 40 pesos silver per ton; 38,200 kg of ore and slag of 2.5 per cent copper and up to 10 pesos silver per ton; 176,000 kg of flux, and 30,200 kg of coke. The total costs amounted to 8576.8 pesos, and the value of the product was 14,718 pesos, giving a profit of 6141.2 pesos. With proper fluxes, which are now being accumulated, the profit should be about double the above.

Cooling in the crucible was reduced by increasing the blast. In order to prevent as much as possible firing at the level of the charging floor, and consequently black tuyeres, charges of 450 kg were used, the coke and charge being moistened with water. The coarse ore and fuel were charged around the circumference of the furnace, and the fine material in the center. This prevented small pieces passing down the vertical sides of the furnace without being properly calcined.

No doubt many ores can be smelted to a matte containing a high percentage of copper, and even to metallic copper, provided pyrites is not added to the furnace charges, but only calcareous or ferruginous oxidized fluxes, such as basic slags, hematite, etc. Loss in the slag is generally said to be the result of producing mattes of high copper content, but in the Rio Blanco smelter, with a high crucible, the slag never rises above 0.16 per cent copper, and the average is only 0.12 per cent.

Santiago de Chile, S. A., Casilla 962.

Tests on Monel Metal Castings.

The Bayonne Casting Company, which has succeeded in making castings of monel metal up to 25,000 lb. in weight, has made some interesting tests of such castings under U. S. Navy inspection.

This new "natural alloy" bids fair to displace steel and bronze in a large number of cases as it has been found to be as strong as steel and is less corrodible than bronze, while it takes a finish like pure nickel. Below are the results of the tests and the U. S. Navy specifications for monel metal castings.

Bureau of Steam Engineering, United States Navy specifications for monel metal castings are as follows:

Yield point (minimum).....	32,500 lb. per sq. in.
Tensile strength (minimum)...	65,000 lb. per sq. in.
Elongation in 2 in. (minimum)	25%

An average of tests of monel metal from 25 heats made under U. S. Navy inspection gave:

Yield point	35,197 lb. per sq. in.
Tensile strength	72,843 lb. per sq. in.
Elongation in 2 in.....	28.2%
Reduction of area.....	35.6%

The Electric Reduction of Iron Ores.

With Special Reference to Results Obtained in the Electro-Metals Furnace at Trollhättan, Sweden, and in the Noble Furnace at Heroult, California.

BY OTTO FRICK.

The publication of Mr. J. A. Leffler's report* on the Trollhättan furnace, presented at the annual meeting of "Järnkontoret" in Stockholm on May 31, 1911, offers an opportunity to discuss the possibilities of the electric reduction of iron ores.

The writer has not had an opportunity to personally inspect the Trollhättan furnace, but Mr. Leffler's report gives a very detailed account, both of the construction and of the working of this furnace. The information with regard to the Noble furnace has mainly been collected during a visit at Heroult, Shasta County, Cal., last year.

Type and General Design of Furnaces.

The Trollhättan furnace and the Noble furnace belong to the same type, and are on most essential points alike. This type has been developed from experiments carried out on different lines and without any mutual understanding or communication.

Both furnaces consist of a wide melting chamber superposed by a shaft of smaller diameter so as to allow the introduction of electrodes through the roof above the crucible—a type of furnace which has been proposed by Keller already in 1901.

The main advantage of this type of furnace is the possibility of using a high shaft and introducing the electrodes in the lower part of the furnace without exposing them to too great a pressure from the charge and the possibility of producing a hot zone on the outside of the charge column at some distance from the brick work.

The shape of the shaft is in both cases of a similar form as used in an ordinary blast furnace, that is to say conical towards the melting zone as well as towards the furnace top. This shape is preferable to a cylindrical shaft, as it allows to increase the volume of the shaft without interfering with the dimensions of the melting chamber, which in an ordinary blast furnace has to be narrower than the shaft to allow the blast to reach the center, and in the case of the electric furnace has to be wider than the shaft to allow the introduction of the electrodes through the roof of the said chamber.

In the first furnace of the Electro-Metal type, built at Domnarfvet, Sweden, the shaft of the furnace was supported on iron pillars. In the Trollhättan furnace the shaft is hung on strong beams, in a similar manner as used in the Noble furnace, erected one year before the Trollhättan furnace, which arrangement is preferable to the previous one as it secures free space all around the melting crucible.

It has been claimed that this shape in the case of an electric furnace should offer special advantages not present in an ordinary blast furnace, in so far as it should prevent the charge in the melting chamber from being pressed against the roof, an effect pretended to occur with a shaft of cylindrical shape. This is, however, not the case, as the shape of the shaft has no influence on the behavior of the charge in the melting chamber.

Whether the lower part of the shaft is conical or cylindrical, there will be the same pressure at the neck of the melting chamber, and if the temperature of the charge at this point is below the one at which the material gets pasty, the charge will in both cases take a conical shape in the melting chamber, the surface of which will form an angle of about 35° to the horizontal. Only if the temperature at the neck were high enough to make the charge enter into the melting chamber in a semi-fluid state there would be a danger of the charge spreading against the roof.

As it is under no circumstances desirable, and, in fact, next to impossible to work the furnace in such a manner, no such danger needs to be feared. In later runs of the Noble furnace

*See this journal, July issue, p. 368, September, p. 459, October, p. 505.

the shaft has obtained a cylindrical shape without altering the behavior of the charge in this respect.

Size and Power Consumption	Trollhattan Furnace	Noble Furnace
Maximum power consumption	2,500 hp 1,830 kw	2,040 hp 1,500 kw
Total height of furnace	14,700 mm	8,300 mm
Outside diameter of crucible	5,650 mm	3,960 mm
Outside diameter of shaft	3,200 mm	2,790 mm
Volume of shaft	25.5 cu. m	7.9 cu. m
Radiation surface	189 sq. m	105.5 sq. m

From these figures it will be seen that the dimensions of the Trollhättan furnace are considerably bigger than that of the Noble furnace.

The diameter of the crucible is mainly governed by the size of the electrodes which have to be introduced through the roof. By the use of graphite electrodes, and also by using the greatest possible number of electrodes, a saving in the diameter of the crucible is obtained.

The size of the shaft in the Trollhättan furnace has been calculated for an average ouput of 23 tons of pig in 24 hours, and on the assumption that the proportion:

$$\frac{\text{volume of charge in 24 hours}}{\text{volume of shaft}} = 1.55.$$

This figure is evidently based upon the average practice of the Swedish charcoal blast furnaces, in which it varies between 1.21 and 1.91.

The author of the Trollhättan report expresses the opinion that the correctness of this calculation has been substantiated by the results obtained, but it appears as if these results in reality show that the shaft is unnecessarily big.

In a paper read by the present writer at Järnkontoret in Stockholm in May, 1905, it was shown that the electric reduction furnace theoretically only needs to be 1/12 of the size of that of an ordinary blast furnace.

The main object of a shaft on any furnace is to recuperate the heat of the gases from the melting zone, and if possible to give the carbonic oxide an opportunity to reduce part of the ore.

In an ordinary blast furnace the weight of the gases exceeds the weight of the ore, limestone, and coke or charcoal by 30 per cent to 50 per cent.

In an electric reduction furnace the gases evolved amount only to about 40 per cent of the weight of the charge.

By the use of a circulating gas introduced in the lower part of the furnace, it is, however, possible to increase the total weight of the gases passing through the shaft so much as to make it exceed the weight of the charge in a similar proportion as in an ordinary blast furnace, but even then a considerable difference between the blast furnace and the electric furnace remains in so far as the temperature of the gas entering the lower part of the shaft cannot be as high in the electric furnace as in the blast furnace unless the circulating gas is heated outside the furnace by the use of hot blast stoves.

In the Trollhättan furnace such an arrangement has not been provided. The heat, carried by the circulating gas to the shaft, has thus to be absorbed from the melting zone, and is necessarily restricted by the surface and the temperature of the charge through which the gas is passed.

The result of the Trollhättan furnace demonstrates to full evidence that this surface and this temperature just mentioned are not sufficient to allow the gases to reach more than 500° C.

From the records collected the highest temperature ever observed at the bottom of the shaft has been 682° C, and the lowest has been 241° C, while the average temperature at this point has been 510° C.

The corresponding temperature in a blast furnace is considerably higher and probably a great deal above 1000° C. The total heat stored in gases at 1000° C. is nearly 2.5 times bigger than that at 500° C. It is, therefore, evident that with the same amount of gas per ton of pig the volume of the shaft of an electric reduction furnace can be made 2.5 times smaller than the corresponding shaft of an ordinary blast furnace with an equally satisfactory transfer of the heat from the gases of the charge. Whether it is consistent with economical working to use so much circulating gas will be discussed later on.

The volume of the shaft of the Trollhättan furnace is 25.55 cu. m.

The corresponding volume in the case of the Noble furnace has been only 7.9 cu. m., and this has proved to be more than ample. In later runs the volume of the shaft has been further reduced by being made cylindrical, apparently without interfering with the economy of the process.

The temperature of the gases leaving the shaft of the Trollhättan furnace has varied between 205° C. and 29° C. and has on an average been 76° C.

These figures compare well with the temperature generally held by the gases from a blast furnace, which as a rule is above 200° C. It has, however, to be kept in mind that the total heat contained in the gases in a blast furnace, say at 1100° C., exceeds the heat which can be absorbed by the charge, whereas the heat of the produced gases in an electric furnace is very much less than that required by the charge.

By varying the amount of circulating gas in the electric furnace, the total heat brought into the shaft by the gases can be made to equal or to exceed or to become less than the heat which can be absorbed by the charge.

Electric Arrangements.

The Trollhättan furnace is supplied by two single-phase transformers, each of 1100 kilovolt amperes, on the primary side connected to a three-phase system of 10,000 volts, 25 periods, according to the Scott method of transformation from a three-phase to a two-phase system. From the transformers a two-phase current is delivered to the four electrodes of the furnace.

The Noble furnace has three single-phase transformers each of 750 kilovolt amperes connected to a three-phase system of 2200 volts and 60 periods.

The low-tension current from the transformers is supplied to the six graphite electrodes.

In both furnaces the regulation of the current is effected by a variation of the number of turns of the primary coils of the transformer. The priority of this device belongs to the Noble furnace for which it was designed by the General Electric Company in the early part of 1900.

The arrangement is well known for other purposes and it is only natural that it has been adopted for electric reduction furnaces as it affords a simple and effective means of maintaining a steady power consumption, and is the only possible way if the furnace has not an individual dynamo but is connected to a great power distribution system.

At the Noble furnace it has also been found useful to use the regulating coils alone, and thus to increase the secondary voltage for the purpose of starting the furnace. It was found, for instance, that with cold charcoal and the normal voltage of 70 volts no current would pass through the charge, and only by raising the voltage to about 100 volts in the above explained manner it became possible to make the current start.

This difficulty will evidently not arise if the charge is preheated to a sufficiently high temperature by blowing air into the crucible.

Electrodes and Method of Heating.

The Trollhättan furnace and Noble furnace differ essentially with regard to the method of heating and to electrodes.

The former furnace has four ordinary carbon electrodes of 660 × 660 mm, which enter into the charge only slightly, as it is endeavored to keep them "free burning"; that is to say, to produce a small space between the electrodes and the charge across which electric arcs are formed.

The latter furnace has six graphite electrodes of 8½ in. diameter—216 mm—which enter into the charge as far as

possible so as to prevent them from becoming "free burning" and so as to insure best possible contact between the electrodes and shaft.

In the report on the Trollhättan furnace it is said that "free burning" is an essential feature, as otherwise the voltage becomes unnaturally low and the dimensions of the electrodes and leads must become excessive and that finally too great variations of the power consumption would result.

It is noteworthy that several of the electrodes of the Trollhättan furnace must have entered into the charge quite considerably, as demonstrated by the sketches given in the report. The electrode III, for instance, appears to have entered into the charge with about 90 cm. This electrode shows also the smallest net electrode consumption per ton of steel.

In using "free burning" electrodes the main resistance in the path of the current between the electrodes, and between the electrodes and the bottom, if this is made to form one pole of the circuit, is concentrated in the arcs near the electrodes, so that nearly the whole of the electric energy is commuted into heat at this point. To convey this heat from the electrodes to the different parts of the charge it is necessary that the temperature near the electrodes be essentially higher than that required for the reduction of the ore and for the melting of the pig and the slag.

A further inconvenience of "free burning" electrodes is that the hottest zone is generated near the surface of the charge and thereby causes difficulty in protecting the roof from the heat. This difficulty has, however, in the Trollhättan furnace been proved to be overcome by introducing a suitable amount of cooling gas around the electrodes.

The introduction of the electrodes into the charge to such an extent as to prevent their "free burning" reduces the resistance at the contact between the charge and the electrode, and thereby the amount of heat produced by the passage of the current at this point and increases the heat generated by the current in passing through the charge. Less heat has thus to be transferred from the zone around the electrodes and a lower temperature would thus be required. As the further end of the electrode is at a considerable distance from the surface of the charge, there is no danger of over-heating the roof and no gas circulating for the purpose of cooling the roof will be necessary.

The difficulty which this arrangement causes is undoubtedly more serious in the case of ordinary carbon electrodes than in the case of graphite electrodes.

There is no doubt that it is not the extra cost of cables and electrodes which would prevent the use of the method applied in the Noble furnace.

The essential difficulty is really to be sought in obtaining electrodes of sufficient length and strength.

The great size of ordinary carbon electrodes makes it difficult to manufacture them in great length or to provide them with a reliable joining device.

Also the jointing of graphite electrodes offers certain difficulties. At Heroult the one end of the electrode is shaped into a conical screw fitting into the upper part of another piece of electrode.

The cutting of this screw joint is very easy on account of the softness of the material, and the joint makes a perfect electrical connection, but the strength of the electrode is considerably reduced and has proved not to withstand the pressure exercised by the charge.

In the Noble furnace the pressure of the charge on the electrodes is very nearly equal to their breaking strength, so that the additional force arising from a sudden descent of the charge easily causes the breaking of the electrodes at the joint. By proper design the electrodes can, however, be made strong enough.

In the case of the Noble furnace the patrice part of the joint was essentially weaker than the matrice part, and only by giving them equal strength, the total strength of the electrodes can be increased by as much as nearly one hundred per cent.

The strain on the electrode, caused by the charge, also depends on the inclination given to the electrodes. This inclination is 35° to the vertical, and by reducing the same to 20° the strain on the electrode can be approximately reduced by about 30 per cent.

With graphite electrodes of bigger diameter than 8½ in. still less trouble from this source needs to be experienced.

The method of introducing the electrodes into the charge and thus working the furnace as pure Resistance Furnace requires a bigger current and a lower voltage than in a furnace with "free burning" electrodes.

With increasing amperes it is of great importance that provisions be made to avoid inductive losses along the path of the current. In this respect, the care taken at the Trollhättan furnace does not seem to equal that at the Noble furnace, in so far as the inductive voltage drop per 1000 amperes in the former furnace appears to be 2.5 volts with a periodicity of 25 as against 2.8 volts in the Noble furnace with 60 periods, so that the inductance in the former case is more than double that in the Noble furnace.

Consumption of Electrodes.

The consumption of electrodes in an electric Reduction Furnace is of no small influence on the economy of the process.

The average consumption during the run of the Trollhättan furnace has amounted to 10.28 kg, costing about 75 cents per ton of pig produced. The actual consumption from wear has, however, only been 5.27 kg, so that a waste of nearly 50 per cent has been caused by no arrangement for jointing the electrodes having been provided.

It is, however, to be noted that the total electrode consumption during the four periods to which the report refers has dropped from 11.24 to 7.45 kg. The cause of this reduction in the electrode consumption is probably to a large extent due to the high average content of CO_2 in gases during the first period, but also other causes may have been of influence.

With reference to the consumption of graphite electrodes due to the wear and tear, no reliable figures are available as the breakage has been very big from causes mentioned above.

From the inspection of such broken electrodes which had been in use for some days the actual wear does not seem to have been appreciable.

As there is no arc between the electrodes and the charge, no particles from the electrodes will be carried away by arcing and the erosion of the electrodes can only be caused by the oxygen in the charge or in the circulating gas.

Taking into consideration the much smaller surfaces of contact as against an ordinary carbon electrode, and also the greater resistivity against oxidation, it is not unlikely to be expected that the actual wear of graphite electrodes will be less than that of ordinary carbon electrodes.

At the present price difference, and disregarding the advantages of graphite electrodes in other respects, the wear of these electrodes should only be 40 per cent of that of carbon electrodes to make this item of cost equal for both kinds of electrodes.

Charcoal Consumption.

The average coal consumption of the Trollhättan furnace during the whole run to which the report refers has amounted to 418 kg of charcoal per ton of pig besides the carbon derived from the electrodes and from the coke and electrode pieces charged. As this figure appears to be rather high it has been thought of interest to investigate this question more closely.

For this purpose it has been necessary to figure out the average analysis for all the materials which have been charged into, respectively delivered from the furnace.

In Appendix I these analyses are recorded. In this appendix are also noted the total amounts of material per ton of pig.

Appendix II contains a complete investigation of the different

purposes for which the carbon has been used during the different periods of operation, with the view of finding out the causes of the differences in coal consumption during different periods.

At first the total weight of gases per ton of pig are calculated from the average gas analyses and the total weight of C charged into the furnace. The total weight of gases produced per one ton of pig varies between 810 kg and 905 kg, and averages 855 kg.

The next calculation aims to find out the value of:

$$\frac{\text{Volume CO}}{\text{Volume CO}_2} = \frac{\text{Weight of C burnt to CO}}{\text{Weight of C burnt to CO}_2} = K$$

To arrive at this value the CO and CO_2 contained in the charcoal and limestone have been subtracted from the total amounts contained in the gases.

The value of K which may serve as an indication for the economy of the process varies between 2.57 and 5.58 and averages 3.71. This figure does not differ very much from the figure obtained in an ordinary blast furnace.

Next come the calculations made to find out the amount of oxygen to be reduced from water and ore, and of oxygen introduced into the furnace by air.

The weight of oxygen from water is found with the aid of the hydrogen in the gases reduced by the amount of hydrogen in the charcoal.

The weight of oxygen from the air is calculated on the assumption that the nitrogen in the gases is derived from air introduced into the furnace or into the piping connected with the circulating fan.

The amount of oxygen from ore is calculated on the basis of the analysis of the pig, and the average analysis of the oxygen connected to Fe in the ore, so arrived at, varies from 434 and 460 kg, and averages 445.7 kg.

These figures are very closely in accordance with those previously found from the analysis of the gases.

The amount of C necessary for the reduction of the oxygen is calculated from the formula:

$$C = 0.75 \frac{K + 1}{K + 2} O$$

and should vary between 0.586 and 0.651 kg C for one kg O, the average value for the whole run being 0.619. The calculated weight of C varies between 285.58 and 332.22 kg per ton of pig and differs only by a few per cent from the actual weight of C supplied.

The result is most instructive, as it furnishes full information about the manner in which the carbon has been consumed, and thereby indicates by what means it would be possible to reduce the coal consumption.

From a study of the figures it will be seen that quite an appreciable part of the carbon, 10 per cent to 15 per cent, is consumed in the reduction of the water introduced by the circulating gas.

This part of the carbon is saved in a furnace without any gas circulation and can also be saved in the case of a furnace with circulating gas if provisions are made to take the gas from the furnace at a level below that at which the water is evaporated and by preventing the gas from absorbing any moisture in cleaning the same from the dust.

The only way to diminish the amount of carbon used for the reduction of the ore is by striving at a lower value of "K." In this respect it is noteworthy that the best value of "K" obtained during the second period coincides with the smallest amount of limestone and that during the third period "K" reaches the high value of 5.58, indicating a rather uneconomical working at the same time as the weight of limestone per ton of pig is highest or 336 kg per ton.

This should indicate that the coal consumption in an electric furnace will be reduced by using burnt lime instead of limestone, and as the burning of the limestone can economically be done by using part of the escaping gases for the purpose it thus will save current as well. It should always be made a rule to burn the limestone outside of an electric furnace.

During the period 2 practically all of the Fe_2O_3 has been reduced to FeO by CO. It is scarcely to be expected that any FeO can be reduced by CO under the conditions at present prevailing in a furnace of the Trollhättan type, and it is therefore not probable that the amount of charcoal for the reduction of the ore can be brought down further than during the said period 2 unless some radical alterations of the process are introduced.

Power Consumption.

The average actual power consumption for one ton of pig has been 2391 kw-hours. This corresponds to 3.66 tons per kw-year or 2.692 tons per hp-year if the power were fully utilized during the 8760 hours of the year.

It has become the general practice to express the output of a furnace in tons per kw-year or in tons per hp-year, but it ought to be pointed out that these figures may in fact be rather deceptive, as they do not take into consideration stoppages and loss in output by the furnace not always working at full capacity.

Only in very rare cases electric reduction furnaces will obtain the electric current on the basis of a meter tariff, thus only paying for the actual amount of electricity consumed. As a rule the electrical energy will have to be paid for at a fixed price per year for a certain amount of maximum available power.

In Table 1 the production per kw-year, according to these two different modes of expression, is given for the different periods of the run. From this it will be seen that the tons produced per hp-year to be paid for have varied between 1.315 and 2.7, with an average of 1.88 tons, whereas the production per hp-year, according to the report, has varied between 2.43 and 3.0, with an average of 2.69 tons per hp-year.

It is evident that it is most misleading to people not thoroughly conversant with the question to state the production per hp-year actually consumed, instead of hp-year to be paid for, and it may correctly be suggested that if the production is stated in the former figure it should be accompanied by a statement of the load factor.

As it has been deemed of interest to investigate how the electricity has been spent, an approximate calculation of the power consumption during the different periods has been made in the Appendix III.

The heat required for the reduction has been calculated from the analyses of the pig produced and with due regard to the heat developed by the combination of the oxygen in the ore with the carbon.

The heat necessary for the reduction of the water carried into the furnace by the circulating gas has been found in a similar way.

The heat absorbed by the calcination of the limestone can easily be found from the analyses of the limestone and the weight thereof charged.

In the calculation of the heat necessary for the evaporation of water no consideration has been taken of the water which escapes from the furnace, the amount of which could only be accurately ascertained by knowing exactly the amount of circulating gas. As it seems probable that this has been three to four times bigger than the amount of escaping gas the evaporation of the water in the escaping gases will require one-third to one-fourth of the heat calculated for the evaporation of the water in the circulating gas and would then amount to 10,000 to 15,000 calories per ton of pig.

This undervaluation of the heat necessary for the evaporation has in some way been counterbalanced in calculating the heat carried away by the gases.

As no figures with regard to the temperature of the pig and slag are given, the heat carried away by them can only be approximated and it has been assumed that each kilogram of

TABLE I.—POWER CONSUMPTION.

Period.	1.	2.	3.	4.	Total.
Number of charges	1–10	11	12–17	18–29	1–29
Tons of pig produced	1,154.5	145.5	248.0	332.1	1,882.5
Duration of mn, in hours	2,115.6	189.5	660.25	528.75	3,501.9
Duration of mn, in year	0.242	0.0216	0.0755	0.0603	0.4
Total kw-hours used	2,651,029	312,601	650,480	877,706	4,500,596
Total kw-year used	303	35.7	74.3	100.8	514.8
Total kw-year, to be paid for	445	39.7	139	111	737.0
Load factors	68.1%	89.7%	53.6%	90.3%	60.9%
Kw-hours per ton of pig	2,296	2,149	2,623	2,643	2,391
Kw-year per ton of pig used	0.262	0.246	0.300	0.302	0.273
Kw-year per ton of pig, to be paid	0.335	0.273	0.561	0.335	0.392
Tons of pig per kw-year used	3.82	4.08	3.34	3.31	3.66
Tons of pig per hp year used	2.81	3.0	2.46	2.43	2.69
Tons of pig per kw-year, to be paid	2.98	3.66	1.78	2.98	2.55
Tons of pig per hp-year, to be paid	2.19	2.70	1.315	2.19	1.87

pig has contained 283 calories and each kilogram of slag 500 calories, figures which should not be far from accurate.

In adding up the total power consumption the same has been expressed in kw-hours.

The total kw-hours theoretically required varies according to this calculation between 1536.3 and 2153.6 kw-hours per ton and averages 1721.4 kw-hours per ton. The actual energy consumption has varied between 2149 and 2710 and has averaged 2391 kw-hours per ton.

To this consumption of electric energy should be added the heat generated by air entering into the furnace with the circulating gas. The amount of this heat has varied between 13.95 and 35.5 kw-hours per ton, making the total actual heat consumption to vary between 2184.5 and 2735.8 and to average 2416.3 kw-hours per ton.

The difference between this actual heat consumption and the theoretically calculated energy consumption should correspond to heat losses from radiation from furnace walls, electrodes, leads and transformers, including the heat carried away by the cooling water used at the neck of the furnace and round the electrodes.

Calculations of this kind must always be more or less approximate and cannot be expected to give an absolutely correct image of what is actually taking place, but the information with regard to the distribution of the heat on different purposes will nevertheless be sufficiently accurate.

On studying the different items on which the heat has been expended it is found that a reduction of the energy consumption can be obtained by the same means which will secure a reduction of the coal consumed; that is, by preventing water from being introduced into the crucible with the circulating gas and by using burnt lime instead of limestone in the charge.

A further economy can be secured by drying the charcoal outside the furnace to get rid of the water.

By these precautions the theoretical energy consumption can be reduced by 136 to 232.8, or on an average 208 kw-hours per ton. The actual economy would vary between 195 and 358 and average 294 kw-hours per ton, the economy so secured would amount to from 9.1 to 15.2 and average 12.3 per cent. of the total power consumption.

A further saving will ensue from the use of burnt lime instead of limestone in so far as a lower value of K can be reached.

The average calculated efficiency of the furnace has been 71 per cent. This figure is hardly satisfactory for a furnace of this size, but is to a great extent due to the average low load factor.

In Appendix III it has been attempted to arrive at a distribution of the losses as close as possible corresponding with the actual conditions. The losses in the iron of the transformers and in the copper can be fairly accurately calculated from the information given in the report about the efficiency of the transformer and the size of the leads.

The heat absorbed by the cooling water has been measured at several opportunities and found to vary between 5.3 per cent and 8.2 per cent and average 6.56 per cent of the load.

The report also contains observations of the surface temperature of the furnace, which observations enable one to calculate the radiation losses to be about 200 kw, to which figure has to be added the radiation from the upper part of the electrodes with about 20 kw.

Comparing the losses calculated in this manner with the figures arrived at by the subtraction of the theoretical power consumption from the actual, it will be found that with the exception of the periods 2 and 3 the agreement is very fair.

Taken over the whole run, the two figures are exactly alike, which will indicate that the calculated distribution of losses is not far from being correct.

An appreciable reduction of the losses can only he gained by diminishing the dimensions of the furnace and by suitable precautions to prevent too much heat being carried away by the cooling of the electrodes.

The possible efficiency of the furnace if working at full load can be calculated from the above figures to be

$$\frac{1840-240-0.104\times1840}{1840}=\frac{1408.5}{1840}=0.7655=76.55\%$$

When a more satisfactory load factor can be secured after having overcome the troubles caused by the transformers there should be no difficulty in maintaining this efficiency, which is sufficiently high to make the working of the process satisfactory, although further progress also in this direction may be expected.

As to the power consumption of the Noble furnace, no accurate figures are at hand, but from the manager of the plant it has been stated that the power consumption has averaged 1940 kw-hours per ton. This figure has never been reached in the Trollhättan furnace.*

The cause of this lower power consumption is certainly partly due to the high-grade ore used, but the fact that gas circulation has only been used incidentally and also that the dimensions of the furnace are considerably smaller than those of the Trolltättan furnace have also contributed to the greater economy obtained.

Gas Circulation.

The idea of using circulating gas in an electric reduction furnace was originated by Harmet in 1901.

The effect of the circulating gas is a double one, namely, on one side the cooling of the charge in the neighborhood of the point of introduction of the gas and on the other side the conveying of the heat absorbed in the lower part of the furnace to the charge in the shaft.

The cooling effect becomes of paramount importance in furnaces which are worked along the line of the Trollhättan furnace and in which most of the heat is generated near the free surface of the charge in the melting chamber.

The cooling effect depends upon the amount of heat absorbed by the decomposition of H_2O and CO_2 when brought into contact with incandescent carbon, respectively, in raising the temperature of the gas.

*See, however, the very latest results obtained at Trollhättan, as recorded in our October issue, p. 482. During the week of September 3 to 9, 1911, the figure of 1736 kw-hours per ton was reached.—Editor.

To obtain an idea of the amount of heat that in this way can be absorbed, Table 2 has been computed from the average results of the Trollhättan furnace.

TABLE II. EFFECT OF CIRCULATING GAS

Ratio of Circulating Gas to Produced Gas.	1.		2.		3.	
Temperature to which gas is raised	500°	800°	500°	800°	500°	800°
Heat in gases in kw hours	205	290	410	580	615	870
Heat absorbed in decomposition of CO_2, kw hours.	354	354	708	708	1,062	1,062
Heat absorbed in decomposition of H_2O, kw-hours.	110	110	146	146	165	165
Total heat absorbed by circulating gas kw hours per ton of pig	669	754	1,264	1,434	1,792	2,097

These figures should show that a very effective cooling can be obtained for the protection of the roof, but from the great amount of heat that would be absorbed by a complete decomposition of the CO_2 it appears probable that only a comparatively small part is actually decomposed, considering that the total power consumption has only been 2391 kw-hours per ton.

For the effective protection of the cover against the heat radiation from the arcs it appears as if it should be sufficient to use a circulating gas of not more than twice the amount of produced gas.

In the case of a furnace of the Noble type, in which the electrodes enter so far into the charge as to prevent arcing, it has been found unnecessary to use any circulating gas at all as long as the electrodes remain unbroken.

It has been previously stated that the weight of the gases produced are only 35 per cent to 40 per cent of the weight of the charge, and as the specific heat is very nearly the same for the gas as for the charge the heat absorbed by the produced gas is not sufficient to heat the charge to any high temperature. In fact gases of 1000° can only heat the charge to about 350°.

This has already been recognized in 1901 by Harmet, who has in several patents proposed to increase the volume of gas by establishing a current of circulating gas taken from the top of the shaft and introduced into the melting zone by means of a blower.

The quantity of heat that can be absorbed by the circulating gas is, however, restricted by the temperature of the charge and the surface of contact of the charge through which the gas has to pass.

Without arrangements by which the circulating gas is forced to pass through the space between the electrodes and the charge, and thus through the electric arc, it is not possible for the gases to acquire any high temperature. The observations made at the Trollhättan furnace tend to show that the circulating gas in fact only serves to cool the charge and has not been able to acquire a temperature exceeding 690° C. at the neck of the melting chamber, the average value having been only 510° C.

The manner in which the circulating gas is introduced into the furnace at Trollhättan can thus not effect an essential extension of the zone within which a reduction by CO takes place.

To make the gas effective in this respect arrangements must be provided by which the gases find an opportunity to acquire a higher temperature mainly by passing the gas through the hottest part of the charge

Load Factor.

The average power consumption of the Trollhättan furnace constructed for 1840 kw has been about 1285 kw over the whole time of operation from Nov. 15, 1910, to April 9, 1911. The average load factor has thus been only about 70 per cent.

As the load factor is of the greatest importance for the economical working of the process, it is of interest to study the variation of the same during the different periods. This is done in Table 3.

TABLE III. LOAD FACTOR

	Period. From	To	Total Hours	Total Kw hours	Average Kw.	Load Factor.
1	Nov. 16	Feb. 11	2,115.5	22,651,029	1,252	68.1%
2	Feb. 11	Feb. 19	189.5	312,601	1,650	89.7%
3	Feb. 19	Mar. 19	660.2	650,480	986	53.6%
4	Mar. 19	April 9	528.5	877,706	1,660	90.3%
	Nov. 16th	April 9th	3,502	1,500,596	1,285	69.9%

The low load factor during the first period is probably partly to be ascribed to inexperience with the manipulation of the furnace, partly to the defectiveness and the long break down of Transformer II.

The poor load factor during period 3 seems to have been due exclusively to the break down of the Transformer II. The load factors of the periods 2 and 4 approach 90 per cent, a figure which must be considered very satisfactory.

As there should be no difficulty to avoid break downs of the transformers, the experience at Trollhättan shows that an average load factor of 90 per cent can be maintained during the period of the working of the furnace, thus without taking into consideration the time for repairing and heating up.

It has already been shown that the temperature in the shaft does on an average not exceed 500° C. With suitable bricks it ought, therefore, to stand for years without any repair.

The melting chamber, which naturally is exposed to heavier wear and tear, will have to be relined from time to time. The duration of the first crucible, being about six months, must be considered as most satisfactory. The loss of time for exchange of the lining can be greatly reduced by arranging the melting chamber in such a way that it can be removed altogether and a spare crucible, ready lined, can be put in place.

With such an arrangement the total interruption from the exchange of the lining should not need to exceed a week, and can probably be reduced to two or three days. The load factor for a whole year should, therefore, not need to fall below 88.5 per cent, which means that the furnace could be worked at full capacity during 7750 hours out of a total of 8760 hours per year.

Conclusions.

The results of the Trollhättan furnace prove that the electric reduction of iron ore can be carried out in a technically satisfactory way, although the process is open to improvements by which the cost of production can be reduced.

By eliminating the water from the circulating gas the coal consumption can be reduced by about 14 per cent. The use of burnt lime instead of limestone will secure a further economy as there will be less CO_2 in the gas, which thereby will become more strongly reducing.

The average economy in power consumption which may be gained is:

By eliminating the water in the circulating gas 139 kw hours per ton of pig, or 5.76 per cent.

By using burnt lime instead of limestone 92 kw-hours per ton of pig, or 3.18 per cent.

By heating the charge outside the furnace so as to evaporate the water 62 kw-hours per ton of pig, or 2.58 per cent.

The total average saving thus possible amounts to 293 kw hours per ton of pig, or 12.15 per cent.

Further savings in power are possible by reducing the dimensions of the furnace as much as possible and by always running the furnace at full load.

A small saving may also be secured by preventing the air from entering into the furnace. The average amount of air per ton of pig has been approximately 45 kg, burning 6 kg of charcoal and producing 25 kw hours. If the price of the current is 0.16 cent per kw hour, corresponding with $10 per hp

year, the value of the heat generated is 4.4 cents, whereas with a price of the charcoal of $10 per ton the value of the charcoal burnt is 6 cents.

If provisions were made to obtain all the savings possible, the following result would have been obtained. The average coal consumption would have been 360 kg instead of 418 kg per ton of pig, and probably still lower, as a greater part of the ore would most likely have been reduced by CO on account of the stronger reducing action of the gases. The average power consumption would have been:

$$\frac{1,721\ 4-208\ 1\ 513\ 4}{0.765} = \frac{}{0.765} = 1,985 \text{ kw-hours,}$$

instead of 2391 kw-hours per ton.

With an average load factor of 88.5 per cent the output would have been:

3.0 tons per kw year, paid for
2.87 " " hp year " "

Even still better results may be anticipated because of the saving in power consumption through the stronger reducing capacity of the gases. As it is not possible to predict how much the reduction of the ore by CO may be increased, the saving in power-consumption from this source has not been taken into consideration.

Beckenham, Kent,
England.

Appendix I.—Analyses of Raw Materials and Total Amounts per Ton of Pig.

AVERAGE ANALYSIS OF ORE.

1.

Fe_2O_3	6.51%	Fe	62.5%
Fe_3O_4	77.92%	Si	3.82%
FeO	0.55%	Mn	0.45%
MnO	0.59%	S	0.0095%
MgO	2.66%	P	0.0115%
CaO	1.59%		
Al_2O_3	0.84%		
TiO_2	0.24%		
SiO_2	8.12%		
P_2O_5	0.0285%		
S	0.0095%		
Loss on ignition	0.942%		
	100.00%		

AVERAGE ANALYSIS OF CHARCOAL.

2.

H_2O	14.5%
Gases	10.26%
Ashes	3.06%
C	72.10%
	100.00%

AVERAGE ANALYSIS OF CHARCOAL GASES.

3.

	Per Cent. of Gases.	Per Cent. of Charcoal.
CO_2	38.6%	3.96%
CO	41.65%	4.272%
CH_4	19.04%	1.955%
H	0.71%	0.073%
	100.00%	10.260%

AVERAGE ANALYSIS OF CHARCOAL ASHES.

4.

	Per Cent. of Ashes.	Per Cent. of Charcoal.
SiO_2	37.64%	1.1523%
K_2O, Na_2O	6.04%	0.1840%
CaO	19.33%	0.5920%
MgO	5.53%	0.1690%
Fe_2O_3	22.48%	0.6880%
Mn_2O_3	3.85%	0.1178%
Mn_3O_4	1.87%	0.0572%
P_2O_5	1.00%	0.0306%
SO_3	0.92%	0.0281%
CO_2	1.34%	0.0410%
	100.00%	3.06%

AVERAGE ANALYSIS OF SLAG.

5.

SiO_2	43.06%	Fe =	2.25%
Al_2O_3	5.50%	Si =	20.1%
TiO_2	2.45%	Mn =	1.48%
FeO	2.89%	S =	0.03%
MnO	1.91%	P =	0.028%
CaO	30.13%		
MgO	13.93%		
CaS	0.0674%		
P_2O_5	0.0597%		
	99.9971%		

AVERAGE ANALYSIS OF PIG IRON.

6.

Fe	95.918%
C	3.36%
Si	0.32%
Mn	0.37%
S	0.015%
P	0.017%
	100.00%

AVERAGE ANALYSIS OF DRY GASES BY WEIGHT.

7.

		C.	O.	H	N.
CO_2	33.16%	9.06%	24.1%		
O	0.12%		0.12%		
CO	60.45%	25.9%	34.55%		
H	0.83%			0.83%	
CH_4	1.45%	1.05%		0.35%	
N	4.04%				4.04%
	100.00%	36.01%	58.77%	1.18%	4.04%

MATERIALS FOR ONE TON OF PIG.

8.

a. *Ore:*

Fe_2O_3	105.62 kg
Fe_3O_4	1,266.29 kg
FeO	8.97 kg
MnO	9.62 kg
MgO	43.23 kg
CaO	25.85 kg
Al_2O_3	13.64 kg
TiO_2	3.90 kg
SiO_2	131.95 kg
P_2O_5	0.46 kg
S	0.15 kg
Loss on ignition	15.30 kg
Total	1,625.00 kg

b. *Charcoal.*

H_2O	60.9 kg
Gases	42.9 kg
Ashes	12.8 kg
C	301.4 kg
Total	418.0 kg

Gases from Charcoal:

CO_2	16.55 kg
CO	17.8
CH_4	8.16
H	0.3
Total	42.90

c. *Limestone:*

Fe_2O_3	0.31 kg
MnO	0.45 kg
MgO	0.40 kg
CaO	70.21 kg
SiO_2	2.18 kg
P_2O_5	
S	0.0013 kg
CO_2	55.65 kg
Total	129.2 kg

d. *Pig Iron:*

Fe	959.18 kg
C	35.60 kg
Si	3.20 kg
Mn	3.70 kg
S	0.15 kg
P	0.17 kg
	1,000.00 kg

e. *Gases—dry:*

CO_2	281.86 kg	C	106. kg
O	1.02 kg	O	199.6 kg
CO	513.82 kg	H	10.0 kg
H	7.05 kg	N	34.4 kg
CH_4	11.91 kg		
N	34.34 kg	Total	850.0 kg
Total	850.00 kg		

Appendix II. on the coal consumption during different periods and Appendix III. on power consumption will be found on pages 638 and 639 respectively.

Electrolytic silver may be removed easily from cathode sheets by painting the sheets with a mixture of silver nitrate, copper nitrate and hydrochloric acid. After drying these sheets can be used for two or three months before deposits will begin to stick. At the San Francisco mint electrolytic refining is done with a direct current at 15 volts, and a current density of 8.3 amperes per square foot of cathode surface.

Concrete mortar blocks may be constructed according to the following method, as outlined in the *Cement Age*. The proportions of material are: 1 part by volume of portland cement, 3 parts by volume of freshly screened sharp sand, 5 parts by volume of freshly broken rock of any character suitable for concrete work, and passing a 3-in. ring. This proportion is continued to a point within 2 ft. of the upper surface of the mortar block, when the proportions may be changed as follows: 1 part by volume of portland cement, 2 parts by volume of freshly screened sand, 4 parts by volume of freshly broken rock passing a 1.5-in. ring. The upper layer of 6 in. from the top of the block should be constructed with the following proportions: 1 part by volume of portland cement and 1 part by volume of freshly screened sand. In some cases these blocks have been built for each 10-stamp battery, or in others a solid block has been built under the entire 100 stamps in the mill, without any expansion joints; and after several years' service the block appears to be in excellent condition.

Appendix II.—Investigation of the Coal Consumption During Different Periods.

Number of Charges.		1–10.	11.	12–17.	18–29.	1–29
1. Average gas analysis by weight in per cent	CO_2	33.53	37.60	33.25	28.70	33.16
	O	0.186	0.05	..	..	0.12
	CO	59.45	55.10	61.60	66.80	60.45
	H	0.792	0.782	0 84	1.04	0 83
	CH_4	1.62	0.92	0.97	1.23	1.40
	N	4.475	5.56	3.34	2.01	4.04
Total	C%	35.87	34.54	36.20	37.40	36.01
Total	HC	1.397	1.012	1.082	1.347	1 18
2 Average weight of charcoal per 1 ton pig, in kg	Water	69.1	50.8	59.8	40.2	60 9
	Gases	41.7	36.9	49.3	43.1	42.9
	Ashes	11.8	11.0	13.2	17.2	12 8
	Coke	293.1	277.6	323.4	325.7	301.4
Total		415.7	376.3	445.7	426 2	418.0
3. Average weight of limestone per 1 ton pig, kg		85.2	61.4	336.0	158.0	130 2
Average weight of CO_2 in limestone per 1 ton pig, kg		36.5	26.3	144.0	67.8	56.6
Average weight of C in limestone per 1 ton pig, kg		9.95	7.18	39.3	18.5	15.26
4 Average weight of C charged in	kg					
C from limestone	kg	9.95	7.18	39.3	18.5	15.26
C from electrodes at 95.7%	kg	5.58	5.02	4.32	3.7	5.05
C from electrodes, pieces	kg	0.80	0.66	1.51	0.29	1.04
C from coke at 80%	kg	0.36	....	....	....	0.48
C from charcoal—coke	kg	293.10	277.60	323.40	325.70	301.40
C from charcoal—gas	kg	17.80	15.75	21.00	18.40	18.30
Total C charged	kg	327.59	306.21	389.53	366.59	341.53
5. Weight of C in pig	kg	35.20	31.70	28.80	33.00	33 60
Weight of C in gases leaving furnace	kg	292.39	274.51	360.73	333.59	307.93
6 Total weight of gases per 1 ton of pig	kg	816	795	995	891	855
7. Weight of CO_2 from limestone	kg	36.5	26.3	144.0	67.8	56.0
Weight of CO_2 from charcoal at 38.6 %	kg	16.1	14.25	19.0	16.6	15,56
Total	kg	52.6	40.55	163.0	84.4	72.56
Weight of CO_2 in gases	kg	273.0	298.55	330.5	255.4	283.56
Weight of CO_2 from oxidation of C	kg	220.4	258.0	166.5	171.0	211.0
8. Weight of CO in gases	kg	485.0	437.5	613.0	594.0	516
Weight of CO from charcoal at 41.65%	kg	17.3	15.4	20.5	17.9	17.8
Weight of CO from oxidation	kg	467.7	422.1	592.5	576.1	498.2
9. CO/CO_2 from oxidation, by weight		2.12	1.635	3.55	3.365	3.36
CO/CO_2 by volume = K		3.33	2.57	5.58	5.28	3.71
10. Weight of O in CO_2 from oxidation	kg	160.2	187.5	121.0	124.3	153.3
Weight of O in CO from oxidation	kg	267.2	241.5	338.5	329.7	284.5
Total weight of O from oxidation	kg	427.4	429.0	459.5	454.0	437.8
11. Weight of H in gases	kg	11.38	8.05	10 75	11.97	10.1
Weight of H in charcoal—gases at 5.27%	kg	2.19	1.94	2.59	2.26	2.26
Weight of H from water	kg	9.19	6.11	8.16	9.71	7.84
12. Weight of O in water, decomposed	kg	73.52	48.88	65.28	77.68	62.72
Weight of H_2O in charge, decomposed	kg	82.71	54.99	73.44	87.39	70.56
13. Weight of N in gases, from air	kg	36.40	44.1	33.2	17.9	34.5
Weight of O from air	kg	11.25	13.6	10.24	5.52	10.65
Weight of free O in gases	kg	1.52	0.4	.... ..	.... ..	1.03
Weight of O from air, burnt by C	kg	9.73	13.2	10.24	5.52	9.62
14. Average analysis of pig in per cent	Fe	95.69	96.27	96.78	96.93	95.92
	C	3.52	3 17	2 88	3.3	3.36
	Si	0.313	0.267	0.196	0.43	0.32
	Mn	0.453	0.270	0.101	0.316	0.37
15. Weight of O from ore at 0 3825 kg per kg	Fe	366	368	370	371	366 5
Weight of O from ore at 1.142 kg per kg	Si	3.6	3.1	2.26	4.9	3.65
Weight of O from ore at 0.291 kg per kg	Mn	1 3	0.8	0.3	0.9	1.08
Total weight of O from ore	kg	370.9	371 9	372 56	376.8	371.23
16. Weight of O from ore	kg	370.9	371.9	372.56	376.8	371.23
Weight of O from air	kg	9.73	13.2	10.24	5.52	9.62
Weight of O from H_2O	kg	73.52	48 88	65.28	77 68	62.72
Total weight of O to be reduced	kg	454.15	433.98	448.08	460.00	443.57
17. Weight of C in gases as CH_4	kg	9.95	5.58	7.25	8.20	8 97
Weight of C in charcoal gases as CH_4	kg	6 10	5.51	6.54	6 25	6 13
Weight of C for the formation of CH_4	kg	3.85	....	0.71	1.95	2.84
18. Kg C per kg $O = 0.75 \frac{k+1}{k+2}$		0.609	0.586	0.651	0.647	0.619
19. Weight of C for different purposes:						
Weight of C to reduce ore	kg	225.5	217.5	242	243 5	229.6
Weight of C to reduce H_2O	kg	44.75	28.65	42.5	50.2	38.80
Weight of C to burn O from air	kg	5.92	7.73	6.67	3.57	5.95
Weight of C to form CH_4	kg	3.85	....	0.71	1.95	2.84
Weight of C in pig	kg	35.20	31.70	28.80	33.0	33.60
Total weight of C required	kg	315.22	285.58	320.68	332 22	310.79
20. Weight of C actually supplied:						
Weight of C from charcoal	kg	293.10	277.60	323.40	325.70	301.40
Weight of C from electrodes	kg	5.58	5.02	4.32	3.70	5.05
Weight of C from electrodes—pieces	kg	0.80	0.66	1.51	0.29	1.04
Weight of C from coke	kg	0.36	.....	....	.....	0.48
	kg	299.84	283 28	329 23	329 69	307.97
21. Weight of charcoal, which can be saved if no H decomposed, thus no H_2O in circulating gas	kg	69	39.0	59 6	68.2	58
Per cent of actually used		16.6	10.4	13.35	17.0	13.85

Appendix III.—Power Consumption.

Number of Charges	1–10.	11	12–17	18–29	1–29.
Calculation of power consumption for 1 ton of pig:					
1. Reduction of O from Fe at 4,330 calories	1,583,000	1,593,000	1,600,000	1,605,000	1,586,000
Reduction of O from Si at 6,850 calories	24,700	21,200	15,500	33,600	25,000
Reduction of O from Mn at 5,950 calories	7,700	4,700	1,800	5,300	6,400
Reduction of O from P about calories	1,000	1,000	1,000	1,000	1,000
Total calories for reduction of ore	1,616,400	1,619,900	1,618,300	1,644,900	1,618,400
2. Heat generated by O and C per 1 kg O $= 1850 \times \frac{k + 3.262}{k + 2}$ calories	2,287	2,360	2,155	3,170	2,260
3. Total calories from O in ore + C	849,000	878,000	804,000	818,000	839,000
Calories to be supplied by electricity for the reduction of the ore	767,400	741,900	814,300	826,900	779,400
4. Reduction of O from H_2O at 3,600 calories	264,700	176,000	235,000	279,000	226,000
Heat generated by O in H_2O + C calories	168,000	115,300	140,800	168,500	141,700
Calories to be supplied by electricity for the reduction of H_2O	96,700	60,700	94,200	110,500	84,300
5. Calories to be supplied by electricity to expel CO_2 from limestone	36,500	26,300	144,000	67,800	56,000
6. Calories to be supplied by electricity for the evaporation of water	44,400	29,450	39,300	46,800	37,800
7. Calories for heating generated gases to 100° C. CO_2 at 22.7	6,200	6,770	7,500	5,800	6,450
O at 20.7	32	8	...	...	21
CO at 23.9	11,590	10,430	14,620	14,250	12,320
CH_4 at 62.5	825	457	603	686	750
H at 332	3,790	2,670	3,575	3,980	3,355
N at 23.9	870	1,052	795	418	826
Calories to be supplied by electricity for the heating of produced gases	23,307	21,387	27,093	25,134	23,722
8. Calories for heating circulating gases, assumed to be about	46,600	42,800	54,000	50,000	47,500
9. Calories to melt and superheat pig	283,000	283,000	283,000	283,000	283,000
10. Calories to melt and superheat slag at 500	102,500	112,000	390,000	229,000	163,500
11. Heat developed by O from air + C calories	22,200	30,400	22,100	11,950	21,700
Heat developed by O from air in kw-hours	25.9	35.5	25.8	13.95	25.3
12. Actual power-consumption in kw-hours	2,296	2,149	2,710	2,581	2,391
13. Power consumption in kw-hours for 1 ton of pig:					
For reduction of ore kw-hours	896.0	865.0	950.0	965.0	910.0
For reduction of H_2O kw-hours	112.8	70.9	109.9	129.0	98.5
For expelling of CO_2 kw-hours	42.5	30.7	168.0	79.1	65.3
For evaporating of H_2O kw-hours	51.8	34.4	45.9	54.7	44.2
For heating generated gases kw-hours	27.2	24.9	31.6	29.3	27.6
For heating circulated gases kw-hours	54.4	49.8	32.2	58.6	55.2
For melting pig kw-hours	330.0	330.0	330.0	330.0	330.0
For melting slag kw-hours	119.5	130.6	455.0	267.0	190.6
Total kw-hours, theoretically	1,634.2	1,536.3	2,153.6	1,912.7	1,721.4
14. Actual power consumption in kw-hours	2,296	2,149	2,710	2,581	2,391
Heat from O in air in kw-hours	25.9	35.5	25.8	13.95	25.3
Total kw-hours actually consumed	2,321.9	2,184.5	2,735.8	2,599.95	2,416.3
15. Efficiency of furnace in per cent. of electrical energy supplied	70.1%	69.9%	78.5%	73.4%	71%
16. Losses through radiation, etc. kw-hours	687.7	648.2	582.2	687.25	694.9
17. Hours per 1 ton of pig	1.83	1.297	2.66	1.59	1.86
18. Losses per hour kw	376	500	219	432	373
19. Average power-consumption kw	1,252	1,650	986	1,660	1,285
20. Theoretical saving in power-consumption:					
If no H_2O is reduced kw-hours	112.8	70.9	109.9	129	98.5
If burnt lime is used instead of limestone kw-hours	42.5	30.7	168.0	79.1	65.3
If no H_2O to be evaporated kw-hours	51.8	34.4	45.9	54.7	44.2
Total saving per ton of pig kw-hours	207.1	136.0	323.8	262.8	208.0
21 Practical saving possible in kw-hours by using burnt lime and dry gas	296	195	412.5	358	294
22. Practical saving possible in per cent of total power-consumption	12.9%	9.1%	15.2%	13.86%	12.3%
23 Distribution of losses:					
Loss in transformer iron at 20 kw kw-hours	36.6	25.84	53.2	31.8	37.2
Loss by radiation at 220 kw kw-hours	402.5	285.—	585.0	350.0	409.0
Loss in water cooling at 6.6% kw-hours	151.0	141.7	178.6	170.2	157.5
Loss in copper of transformer and leads at 3.8% kw-hours	87.2	81.7	103.0	98.0	91.0
	677.3	534.24	919.8	650.0	694.7
24. Practical saving in kw-hours					
If no H_2O is reduced	161	101.2	140	175.8	138.8
If burnt lime is used instead of limestone	60.6	43.9	214	107.4	92
If no H_2O is evaporated	74	49.2	58.5	74.5	62.3
Total	295.6	194.3	412.5	357.7	293.1
25. Practical saving in per cent of total power-consumption:					
If no H_2O is reduced	7	4.7	5.16	6.8	5.26
If burnt lime is used instead of limestone	2.64	2.04	7.90	4.15	3.81
If no H_2O is evaporated	3.33	2.29	2.16	2.88	2.08
Total	12.97	9.03	15.22	13.83	12.15

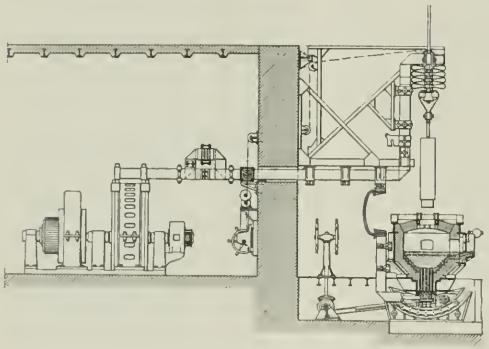
Remelting of Ferromanganese in the Electric Furnace and the Use of Molten Ferromanganese for Deoxidation.

An article by Bronn and Schemmann on the remelting of ferromanganese and the use of molten ferromanganese as a deoxidizer was abstracted in the November issue of this journal, page 575.

Another very interesting article dealing with the same subject was published in *Stahl und Eisen* of Sept. 7, 1911, page 1467, the author being Dipl. Engineer Felix Schroeder, of Livet (Isere), France.

The waste of ferromanganese when charged cold, especially during the time when slag is being removed, caused the abandonment of this process. Other disadvantages were delays due to the time required for the absorption of the solid ferromanganese, a cooling down of the bath and an irregular distribution of the manganese in the steel.

The best method for melting ferromanganese was found to be the use of the electric furnace, and the results obtained with one furnace of the Keller system* at the Burbacherhutte in Saarbrücken are described. It was adopted there after previous tests at the works of Keller-Leleux Company in Livet had shown the possibility of melting 80 per cent ferromanganese in the electric furnace without any losses of manganese.



ELECTRIC FURNACE FOR MELTING FERROMANGANESE.

The furnace seen in Fig. 1 was designed for the manufacture of steel and was run alternately on steel and ferromanganese during the last nine months. The use of molten ferromanganese was begun at once on a large scale. More than 1000 tons of ferromanganese have been remelted in the electric furnace, to be used as molten addition to all basic converter metal in the ladle for several months.

The continuous operation of the converter plant during the six weekdays required an uninterrupted running of the electric furnace in order to have hot ferromanganese ready whenever needed. On Sunday the metal usually remained in the furnace and was maintained fluid by the electric current, the consumption of electric current for this purpose being "very low." When no electric power was available the furnace was emptied and restarted on Sunday night. During the week the furnace was never emptied. Small repairs were done on Sunday, the hot metal meanwhile being poured into a ladle.

Changes of electrodes were very seldom necessary and required only a short time. The electrode was raised, the arm turned aside, and the new one, already hanging on another arm, lowered into the furnace.

Special attention was paid to the lining of the furnace, in order to avoid any interruption of the furnace operation due to repairs of the lining. The longest period during which the furnace was used for melting ferromanganese was three months. During this time the roof of acid brick, which had already been used before during a period of steel refining, was neither repaired nor renewed.

The dolomite lining was not renewed, and only a few small repairs were made from time to time. After these three months of ferromanganese melting the furnace was started to run on steel again, and the lining was still in fair condition.

The roof and lining of the furnace when used for melting ferromanganese, therefore, last at least three months, and the author claims that a lining of magnesite would stand from six to nine months in continuous operation.

The inclined lower lining between door and hearth and also the lining of the hearth itself were not changed for nine months, requiring only a few repairs.

The above facts, therefore, disprove the fear sometimes expressed that the hearth of the Keller furnace, owing to its complicated structure, would have only a short life. It has also been suggested that the iron rods of the hearth would likely melt or be converted into ferromanganese, but neither did happen, as it could be avoided very easily*

The furnace is started by lowering the upper electrode to the charge of solid ferromanganese. It is not necessary nor desirable to have previously melted iron in the furnace.

The operation of the furnace is very simple, one melter and a boy being sufficient. The best place for the furnace is the charging floor of the converter.

The molten ferromanganese is poured into a small ladle and brought to the different converters. The addition of it into the steel ladle is carried out in the same way as the addition of spiegeleisen in the production of rail steel. The furnace may be dipped by hydraulic pressure like the converters.

From the results obtained at the Burbacherhutte, the following conclusions may be drawn:

1. The remelting of ferromanganese in the Keller furnace occurs without any losses of manganese by volatilization.

For example, 10 average samples of solid ferromanganese had the following analyses before charged into the electric furnace:

80.44	81.60	82.32
80.12	79.81	79.18
79.80	79.52	79.71
80.32	Average = 80.20 per cent Mn.	

Thirty-seven successive samples of molten ferromanganese taken from the electric furnace had the following analyses:

81.35	79.82	79.18	80.32
81.06	81.06	79.18	81.06
81.38	80.12	81.06	81.38
80.44	80.44	80.75	81.06
80.12	80.12	81.06	80.44
79.81	80.44	79.50	79.81
79.20	79.50	80.12	79.50
80.12	80.32	79.91	80.32
80.11	79.32	79.51	79.91
79.32	Average = 80.22 per cent Mn.		

The remelting, therefore, causes practically no loss of manganese. There is no smoke nuisance at the furnace.

2. The net saving due to the use of molten ferromanganese amounts to 35 per cent. of the amount of solid ferromanganese which would be necessary for the production of steel of the same quality and under the same conditions of operation. Another saving results from the fact that all the fine pieces of ferromanganese, which are usually lost in ordinary practice, can be used up.

In this comparison it is assumed that the same amount of manganese shall be in the finished steel as in usual practice. As a matter of fact, however, it is possible, if the addition of

*For a detailed description of the Keller furnace see our Vol. VII, page 255 (1909).

*This would seem to contradict the experience of Bronn and Schemmann with the steel electrodes in the bottom of the Girod furnace (see page 575 of our November issue).—Editor.

ferromanganese is made in molten form, to reduce slightly the manganese content in the finished steel without affecting the rolling quality.

3. Costs of melting based upon a price of half a cent per kw-hour were approximately $5 per ton of ferromanganese, including electric power consumption, electrode consumption, wages, linings, and repairs. These are not the costs of special experiments, but a monthly average, and therefore include costs of heating and starting the furnace, delays, interruptions due to electrode changes, Sunday work, etc., or, briefly, all the expenses of using the process in a commercial basic converter plant.

Assuming that the melting-cost per ton of ferromanganese is $5, the price of one ton of ferromanganese $40 only (abroad), and that 5 kg. of ferromanganese are to be used per ton of steel, we obtain the results given in Table 1.

TABLE I.

Yearly production of steel in tons.	Ferromanganese remelted in tons.	Total melting cost in dollars.	Saving of ferromanganese in tons, if 1 kg. of ferromanganese is saved per ton of steel.	Price of ferromanganese saved in dollars.	Saving of ferromanganese in tons, if 2 kg. of ferromanganese is saved per ton of steel.	Price of ferromanganese saved in dollars.
100,000	500	2,500	100	4,000	200	8,000
200,000	1,000	5,000	200	8,000	400	16,000
300,000	1,500	7,500	300	12,000	600	24,000
400,000	2,000	10,000	400	16,000	800	32,000
500,000	2,500	12,500	500	20,000	1,000	40,000
600,000	3,000	15,000	600	24,000	1,200	48,000
800,000	4,000	20,000	800	32,000	1,600	64,000

It may be said that, on account of the smaller ferromanganese consumption, the weight of steel so produced is also less. The slag is also less. The useful phosphorus in the slag increases and there is no reduction of phosphorus back into the metal. All these facts have a very small influence on the final results.

The price of ferromanganese at present is very low and may still decline. The above consumption of 5 kg. ferromanganese per ton of steel is also an assumed figure; it will in reality vary according to the quality of steel produced at different plants, therefore the above calculation should only be considered as an example.

It shows, however, that in no case in melting costs exceed the saving in ferromanganese even if only 1 kg. of ferromanganese per ton of steel should be saved. But the situation is improved considerably if 2 kg. or more of ferromanganese can be saved.

Besides this, the use of ferromanganese offers many other advantages which should not be overlooked. The desired quality of the steel is far more easily obtained than is possible with the use of solid ferromanganese. All of the molten alloy is completely absorbed by the hot metal. With some experience in estimating the manganese contained in a sample of converter metal taken after the blow, it becomes easy to determine accurately the amount of ferromanganese to be added. This is of great value for high-carbon steels.

Using solid ferromanganese on cold heats in the converter, one always runs the risk of its not being absorbed in the bath.

The distribution of molten ferromanganese in the ladle is uniform, the deoxidation quick and more effective, and rephosphorization impossible.

Not only is the quality of the steel improved, but the capacity of the converter plant can also be increased since the time required for the resting of the metal to permit the ferromanganese to dissolve is saved. The gain per heat is always a few minutes, which amounts to something when running full capacity.

There is no reason why this process should not be successfully employed in open-hearth practice. R. J. W.

The Production of Oxygen in the Workshop.

I.—The Production of Oxygen from Alkali-Chlorates.

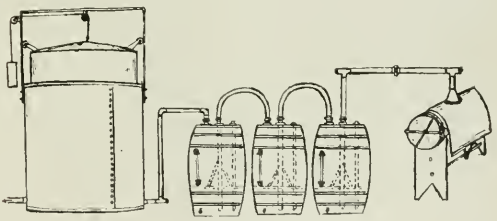
BY AUG. S. NEUMARK.

There are two ways open for the average consumer to obtain oxygen, namely, to purchase the gas compressed in tanks from the manufacturer or to make it himself by chemical means.

Among the various methods used for the production of oxygen, the one generally used (and the most abused, I might add) is the chlorate process. If potassium chlorate is heated to a temperature of 350°-370° Cent., it melts and then liberates all of its oxygen.

The decomposition is an exothermic one, *i. e.*, heat is evolved during reaction, and when it once has set in, the evolution of the gas becomes very violent. This action can be checked by adding a medium which serves to prevent too rapid generation and also lowers the temperature necessary to start the same. For want of a better name such agents are called catalytic agents; the part they play during decomposition is somewhat obscure.

Manganese dioxide (black oxide of manganese) is usually employed for that purpose and is intimately mixed with the chlorate in the proportion of 1:6 or 1:8. The addition of manganese dioxide causes the formation of small quantities of chlorine, and since pure chlorate does not liberate such, it is most likely that the manganese dioxide causes a momentary conversion into potassium manganate, thus setting free chlorine, which on further heating decomposes the manganate again, liberating oxygen and leaving as a residue a mixture of potas-



OXYGEN PLANT (COMPRESSOR TYPE).

sium chloride and manganese dioxide. The reactions may be expressed by the following two equations:

(1) $2KClO_3 + 2MnO_2 = 2KMnO_4 + Cl_2 + O_2$.

(2) $2KMnO_4 + Cl_2 = 2KCl + 2MnO_2 + 2O_2$.

The admixture of the manganese dioxide reduces the temperature necessary for decomposition to 200-210° Cent.

A much better substitute would be ferric oxide, a red powder, used as a material for paints. However, it must be pure and previously have been thoroughly heated. It will reduce the temperature required to 110-120°C.

Other so-called catalytic agents, which have been recommended, are copper oxide, lead peroxide, graphite, dry quartz-sand, and powdered glass.

The addition of common table salt will further check a too violent generation. Mixtures giving the best results consist of 2 parts of chlorate, 2 parts of table salt and 3 parts of ferric oxide; or 12 parts of chlorate, 6 parts of salt and 1 part of manganese dioxide.

According to an English formula, the addition of powdered iron and a small quantity of iron oxalate will still further reduce the temperature.

Large crystals of potassium chlorate must never be used and care should be taken that the chemicals are pure and in the dry state. The presence of any chlorite or hypochlorite in the chlorate is liable to cause an explosion; barium sulphate, which has been used to adulterate the chlorate can also cause explosive decomposition.

One pound of potassium chlorate yields 4¼ cu. feet of oxygen, the cost being about 2½ cents per cubic foot.

Sodium chlorate, having a lower molecular weight, produces from 16 to 18% more gas per unit of weight, thus reducing the cost to about 2 cents per cubic foot. There is no reason to think that sodium chlorate is more dangerous to use than the potassium salt. If the conditions are alike, the decomposition of the former will be as slow and smooth as with the latter.

Violent generation is caused by the melting of the chlorate before giving of its oxygen, and then liberating all of it at once. This, however, can never happen if the materials are pure, if the chlorate is in a perfectly dry state and in form of small pieces or crystals and if intimately mixed with the catalytic substance.

The retorts used for generating oxygen must be of a proper construction. Manganese dioxide or ferric oxide is always carried along by the force of the oxygen current. It is, therefore, essential that the outlet pipe is sufficiently wide. If the outlet is too small, the gas may come off at such rapid rate as to exceed the delivery power of the exit pipe, which, to make matters worse, may be choked up from a previous charge.

Even for the smallest generating outfit, the piping should have a diameter of not less than 1 inch, preferably 1½ inches. It is advisable to have a dome or a cone-shaped exit piece, and that the exit pipe should be short, without any sharp turns, so as to offer the least resistance to the gas. "L" fittings should not be used; "T's" should be substituted, with the open end closed up by a plug. This facilitates cleaning of the pipes.

The oxygen is purified by allowing it to bubble through barrels, partly filled with water and a solution of caustic soda. The delivery pipe reaches nearly to the bottom of the barrel and is provided with some arrangement to break up the bubbles. The barrels should also have a safety pipe which not only prevents any bursting, in case of a too rapid generation, but takes care of the back-pressure when generating ceases.

The last wash-barrel is connected with the gasholder from which the oxygen is forced into storage tanks by means of suitable compressors. The latter are probably the most expensive part of an oxygen plant, although a fairly good compressor, able to produce a pressure of 250 lbs. per sq. inch, can be bought for $125.00. Those having two stages of compression are, of course, more expensive. The compressors must be water-cooled and provided with proper means of lubrication. Oil must never be used as a lubricant; a soapy solution or a diluted solution of glycerine may be used instead.

Oxygen can also be generated under sufficient pressure to force it into storage tanks which would do away with gasholder and compressor and therefore take up very little space. However, such a plant is neither economical nor are the results obtained satisfactory. The waste of gas during recharging reduces the yield of oxygen to 4 cu. ft. per pound. The purification, too, is incomplete and accidents caused by leaving the valve between retort and storage tank (or scrubber) closed are liable to happen. Several of such accidents have come to the writer's notice. Safety valves are of little use; the use of check-valves, made of non-corrosive material, might, however, minimize the possibility of such accidents.

In both the compressor type and pressure type of oxygen plant the gas is liberated by external heating of the retorts. Another type is the internally heated type, where the chlorate mixture is lighted with a match or with the aid of some ignition powder whereupon the retort is sealed. The chlorate which is always in the powdered state is mixed with some combustible substance (charcoal, hydrocarbons, cellulose, powdered metal) and an inert matter (infusorial earth, clay, sand).

Following are a few mixtures which have been used for this purpose:

100 parts chlorate of potash, 40 to 60 manganese oxide, 5 to 20 infusorial earth.

70 to 90 parts chlorate of potash, 0.9 to 8.5 coal dust (lampblack), 0.1 to 21 infusorial earth.

100 parts chlorate of potash, 20 to 40 infusorial earth, 3 to 5 powdered charcoal.

500 parts chlorate of potash, 62 manganese dioxide or ferric oxide, 4 lampblack.

"Oxygenite" belongs to this class of combinations.

The Applications of Electricity in the Metallurgical Industry of Italy.*

By R. Catani

The development of the use of electricity in recent years has greatly benefited the metallurgical industry in many ways, as a means for driving the auxiliary machinery of metallurgical apparatus, blast-furnaces, and mixers, the hoisting and transport of raw and finished materials, and the driving of rolling-mills. The last and most important development, however, is that of furnaces dependent, for their heat, upon the thermal effects of the electric current.

The Italian metallurgical industry has also largely benefited by the introduction of electric energy, but it is proposed to confine this report mainly to an account of the furnaces and electric processes applied in Italy to metallurgical production.

The subject might perhaps have been dealt with from the point of view of the various types of electric furnaces which have been tried or are being worked at present in Italy, but seeing that this would comprise almost every kind of electric furnace, both of the arc and induction types, and that the paper would consist of a series of descriptions of furnaces such as have already lately appeared in many technical publications, it appears more desirable in the present paper to discuss the metallurgical products obtained in Italy by means of the electric furnace.

The author therefore proposes to deal first with the production of electric steel; secondly, with the smelting of pig iron by electricity; and, lastly, with alloys of iron with other metals,

I. Production of Steel in the Electric Furnace.

In Italy steel is manufactured in the electric furnace either direct from ores or from iron products, and, indeed, the first experiments in the direct production of steel from ore in the electric furnace were made at Rome by Stassano in 1898.

A. Steels Produced Direct from the Ore.—The problem of the direct reduction of steel from the ore presents difficulties of such a kind as to deter many metallurgists, however well skilled, from attempting to solve it. The numerous experiments carried out last century in the endeavor to attain this object, using coal as fuel, met with no success on a commercial scale.

On the other hand, the results of a few years' trial with the electric furnace have surpassed expectation, and it may be affirmed with certainty that, in due time, by the aid of the electric furnace, the problem of producing steel by a direct process on a commercial scale will have been overcome.

The experiments of the last century cleared up several essential points which have assisted greatly in the further development of the process. In Stassano's experiments at Rome the charge was prepared in the same manner as in previous attempts to effect the direct reduction of ore with coal as fuel. The material was first crushed, pressed into briquettes with tar and finally treated in the electric furnace.

Recent trials have shown, however, that in the electric furnace it is equally easy to treat the ore whether in a fine state or in lumps, and hence it is possible to dispense with the preparation of the raw material, which forms a not inconsiderable item in the first cost of the steel.

The early experiments of Stassano, as stated by him in 1902, led to no practical results for the reason that he adhered too closely to blast-furnace practice, both in regard to the shape of

*A paper presented at the October meeting of the Iron and Steel Institute in London.

the furnace and the process itself. In fact, the first furnace used at Rome in 1898 to 1899 was a shaft furnace in which, instead of having tuyeres for blowing, electrodes were provided for conducting electric current. As to the method of smelting, Stassano arrested the ore by means of a grating at a distance of about 20 cm above the electric arcs. The experimental furnace afterwards put up at Darfo did not differ very materially from that at Rome, nor were the results much better than those previously obtained.

Stassano finally adopted a furnace of the hearth type; the principal characteristics were that it was roofed over, and long electric arcs played between the mass of the charge and the arch of the furnace. The arcs swept the surface of the slag without passing through it; the heat, therefore, was not imparted direct by the contact with the slag or the molten mass, but passed indirectly into the bath by radiation. The electrodes had a high density, and were held in water-cooled supports. The furnace was completely closed in, and provision was made for collecting and carrying off the gases evolved during the process.

Some Stassano furnaces are fixed, others can be rotated. In some the only rotary movement is that of tilting toward the side of the tapping hole so as to facilitate tapping. Others can be rotated on their vertical axis so as to stir the bath. Single-phase current with one or more pairs of electrodes or three-phase current can be used. Stassano's experiments for the direct production of steel by means of his electric furnace may be divided into two groups, namely, those performed during the years 1898 to 1902, the results of which were published in the *Rivista di Artiglieria e Genio*, March, 1902, and those which were published in 1908 in the *Giornale del Genio Civile*.[1]

In the *first trials* only hæmatite ore of first-class quality, from the Island of Elba, was used. The chemical analyses of the material smelted, of the product obtained, and the proportions of materials used per 100 kgs. of product are given in Table I.

TABLE I.—FIRST EXPERIMENTS OF STASSANO.

Ores per cent.	Charcoal per cent.	Pitch per cent.	Limestone per cent.	Average Product.
Fe_2O_3..... 93.02	C..... 90.42	C..... 59.20	CaO 51.21	Fe....99.7
MnO..... 0.619	Ash ... 3.88	Hydro- } 40.50	CO_2 ....43.43	C..... 0.10
SiO_2..... 3.790	Moisture. 5.70	carbons }	MgO.... 3.11	Mn ... 0.10
S..... 0.058		Ash.. .. 0.27	Al_2O_3 } 0.50	Si... 0.10
P..... 0.056			Fe_2O_3.. }	
CaO+MgO. 0.500			SiO..... 0.90	
Moisture.... 1.720				

QUANTITY USED PER 100 KG. OF STEEL PRODUCED.

Kilog.	Kilog.	Kilog	Kilog.	Kilog.
165.8	26.5	20	19.1	100

Altogether there were produced only 125.6 kg of steel. The loss of iron in transition from the ore to the steel was about 8 per cent. The consumption of energy averaged 4187 kw-hours per ton of product, and the mean efficiency of the furnace was 48 per cent. Stassano omitted to give particulars of the electrode consumption, but, according to an estimate of his, it was stated to amount to 12 kg per ton produced.

These experiments prove, first, the possibility of obtaining fluid steel direct from the ore in the electric furnace; secondly, the advantage that with such a furnace the amount of materials consumed approximates more closely to the theoretical amount of materials required than was the case in the older processes for the direct production of steel from ore.

The industrial importance of the process, however, was insignificant, for the reason that only material of first-class quality could be used, that the total production was negligible, that it was considered necessary to crush the ores and agglomerate them with tar and, finally, that the energy consumption was excessively high. Stassano concluded his account of 1902 with the following estimate of the cost of the direct production of steel by his processs:

[1]See also the extensive article on his work by Major Stassano in this journal, Aug., 1908 (ELECTROCHEM. & MET. ING., vol. VI, p. 315).

	Lire.
1600 kg of ore at 15 lire per 1000 kg	24.0
Crushing the ore at 3 lire per 1000 kg	4.80
200 kg of limestone at 5 lire per 1000 kg	1.0
250 kg of coke at 45 lire per 1000 kg	11.25
Crushing the coke at 2 lire per 1000 kg	0.50
190 kg of pitch at 70 lire per 1000 kg	13.30
Manufacture of briquettes at 3 lire per 1000 kg of the mixture.	6.75
Consumption of electrodes (12 kg at 30 lire)	3.60
Maintenance of the furnaces	12.0
Labor	6.0
Maintenance of ingot molds and apparatus	3.0
4000 hp at 0.0057 lire per hp hour	22.80
General expenses	3.0
	112.0 =$27.62

(NOTE.—1 lira may be taken as equivalent to 19.3 cents.)

Certain of the data in this estimate appear to be inaccurate; for instance, the use of metallurgical coke is referred to, whereas charcoal was used in the experiments. Also, the cost per kilogram of electrode at 0.30 lire (6 cents) is too low, and the electrode consumption of 12 kg per ton of steel is untrustworthy.

Moreover, Stassano subtracted 18 lire from the 112 lire as representing the value of the carbon monoxide and of the volatile hydrocarbons of the pitch, the total amount of gas yielded being 900 cu. m., valued at 2 centesimi (0.4 cent) per cubic meter. But it is to be noted that, if that consideration is reasonable in the case of the electric furnace, it also holds good for the other ordinary processes in which gas is evolved from coke furnaces, blast-furnaces, and steel furnaces. Theoretically speaking, these gases would have a higher value than the gas from an electric furnace.

Hence, in making comparisons of the cost of ordinary steels, the amount of 112 lire, as given above, should be taken instead of 94, the figure obtained by deducting the value of the gas estimated at 18 lire. Further, to the figure of 112 lire ought to be added the charges for cooling water, for depreciation and for the additions in the refining process, though, on the other hand, it might be reduced by the amount of cost of preparing the ore mixture if, as in other electrical processes for the conversion of ore into steel, it should prove possible to treat pulverulent ores by the Stassano process.

In the *second series of his experiments*, Stassano used a different ore, of the following composition:

	Per Cent
Fe_2O_3	68.70 Fe=48.09
Mn_3O_4	3.23
SiO_2	17.15
Al_2O_3	2
CaO	1
MgO	5.67
P	0.15
S	0.12

In this case, also, the ores were crushed and briquetted, but with a 25 per cent solution of silicate of soda instead of with pitch.

In spite of the impurity of the ore, steels of good quality were obtained, the analyses of the product of charges Nos. 2 and 4 being given in Table II:

TABLE II.—STEELS PRODUCED DIRECT FROM THE ORES IN A STASSANO ELECTRIC FURNACE.

Charge No.	COMPOSITION.					MECHANICAL TESTS.		Coefficient of Quality A×R.
	C.	Mn.	Si.	P.	S.	Tensile Strength, Kg./mm². R.	Elongation per Cent. A.	
2	0.26	0.21	0.03	0.010	0.040	55	23	1,265
4	0.80	0.30	0.22	0.015	0.045	86 3	13	1,122

In the case of this second series of experiments, in which only four charges were worked, particulars were not given either of the total production or of the consumption of electrode and of water, or of the composition of the limestone and the charcoal. The furnace was of 200 hp, with three-phase current, and the average consumption of electric energy was 4250 kw-hours.

The charges were as follows:

	Kilograms
Ore	100
Limestone	35
Charcoal	21
Aqueous solution of silicate of soda	8
Carbide of calcium	5

On the basis of these figures and of the percentage of iron in the ore, and assuming that in this second series of experiments the loss of iron was the same as before, namely, 8 per cent, the consumption of material per ton of steel produced would have been:

	Kilograms.
Ore	2,250
Charcoal	540
Limestone	787
Silicate of soda	180
Carbide of calcium	112

In addition, the author has calculated from the particulars furnished by Stassano at the Sixth Congress of Applied Chemistry, that his three-phase electric furnace of 200 hp used in these last experiments consumed, for every ton of metal produced, about 50 cu. m of water for cooling the holders of the electrodes.

B. Steels from Iron Products.—With the electric furnace, however, the general practice is to use pig iron for the production of iron and steel in the same manner as in the ordinary type of furnace, in which case the material can be charged into the furnace in either the liquid or the solid state. The consumption of electric energy diminishes rapidly the greater the amount of heat stored in the materials when charged, and the greater the degree of the purity of these. Table III shows the approximate power consumption in certain cases taken from actual practice.

TABLE III.—KW-HOURS NECESSARY FOR THE PRODUCTION OF A TON OF STEEL. (RODENHAUSER-SCHOENAWA FURNACE).

	Kw-Hours.
From cold pig iron	1,500
From liquid pig iron	1,100 to 1,200
From cold pig iron and cold scrap	900 " 1,300
From liquid pig iron and cold scrap	600 " 1,000
From cold scrap	300 " 900
Refining.	
Steel of crucible quality from liquid iron	200 to 300
Ordinary electric steel from liquid iron	120
Mixer	50

In general, where electric energy is costly, it is convenient to refine only liquid charges in the electric furnace, but steel-works are not always in the neighborhood of electric power stations of sufficient capacity to supply energy at reasonable prices. In that case it is necessary to melt solid charges in the furnace.

1. **Cold Charging.**—The raw materials consist generally of pig iron, scrap, ore, and additions.

Stassano Furnace.—The first installation in Italy was that of Stassano at the Royal Artillery Works at Turin, established by the Italian Ministry of War. That furnace has been in operation since 1903, and is of 200 hp. Subsequently there was formed in Italy the Stassano Electric Furnace Company, which established an electric steel works at Turin. The electrical characteristics and the number of these furnaces were as follows:

Number of Furnaces.	Horse-power.	Kind of Current.	Volts.	Amperes.
2	100	Single-phase	75	1,000
2	200	Three-phase	100	900
2	1,000	"	150	2,700

Three-phase currents at 21,500 volts were supplied by the Societa Anonima Electtricita Alta Italia, and the voltage reduced by special transformers to 75, 100, and 150 volts for the furnaces of 100, 200 and 1000 hp respectively. The single-phase furnaces of 100 hp were shunted on a single-phase circuit.

The current suplied by the above-mentioned company to the furnaces amounted to about 350 kw from December to March, and to 800 kw from March to December, while the total power of the furnaces was 2600 hp = 1910 kw, as may be calculated from the foregoing data.

The Turin works of the Stassano Electric Furnace Company manufactured chiefly parts of railway carriages and wagons, and of automobiles. The Stassano Thermo-Electric Furnace Company went into liquidation in 1909, and some of the electric furnaces of the Turin plant have been transferred to the works of the "Elba" Company at Portoferraio and some to the Milan Steel Works.

The Stassano furnace installed at Portoferraio now works with cold charges, but it is probable that as the company has available liquid pig iron and Bessemer steel it will soon adopt liquid charging for the production of high-quality steel. The installation at Milan is not yet in operation.

The results obtained with the 200-hp furnace at the Royal Artillery Works at Turin and at the Stassano Thermo-Electric Furnace Company, published by Stassano in 1908, are set forth in Tables IV to VII.

The Héroult Furnace.—Héroult furnaces are installed in Italy at the works of the Mannesmann Tube Company, formed by the amalgamation of the Mannesmann Tube Works at Düsseldorf and of the Metallurgical Company of Rome. The works are situated at Dalmine in the district of Sabbro, Province of Bergamo. Three-phase currents are delivered at the central station at 12,000 volts, where they are transformed to 550 volts. There are two furnaces in which open-hearth steel scrap is melted, and both furnaces are in operation at the present time.

The Girod Furnace.—One Girod furnace is in course of construction at the steel works at Cornigliano Ligure of the Giovanni Ansaldo Company. The furnace is three-phase, of 350 kw at 50 cycles, with a capacity of 4 to 5 tons. The material is charged cold, the maximum potential between phases being 70 volts. The furnace will shortly be put into operation, and is intended for the manufacture of special steels only. It can also be operated with liquid charges, in which case the production will be increased by about 40 to 50 per cent.

Liquid Charging.—Electric steel furnaces may be charged either with liquid pig or with steel produced in the converter or open-hearth, and either the acid or basic process may be employed. Where steel from the converter or open-hearth furnace is charged a simple refining operation alone is required, and the improved quality, and particularly the increased resistance to shock, of the electric steel appear to be due to the much lower percentage of occluded gas in that material as compared with the gas contained in ordinary steel, the commercial composition being the same in both cases.

The Kjellin Furnace.—The first furnace which was worked with liquid charges in Italy was the Kjellin furnace installed at the works of the Giovanni Andrea Gregorini Steel & Iron Foundry Company at Castro on Lake Iseo. At first cold scrap was used, but it is now working on liquid steel taken from a 4-ton open-hearth furnace, which supplies to the electric furnace 1400 kg of metal of the following composition at each heat:

Carbon per cent.	Silicon per cent.	Sulphur per cent	Phosphorus per cent.	Manganese per cent
0.08 to 0.10	0.03 to 0.05	0.010 to 0.020	0.008 to 0.015	0.18 to 0.35

The final product contained:

Carbon per cent.	Silicon per cent.	Manganese per cent.
0.60	0.25	0.12

The recarburization was effected by adding pig iron, and since the furnace is tapped every 2½ hours, yielding at each heat 1400 kg of steel, the daily production, reckoning in the added pig iron, amounts to about 15 tons.

The steels produced, whether of a very soft quality or hard, are very malleable notwithstanding when the manganese is

TABLE IV.—200 HORSE-POWER FURNACE AT THE ROYAL ARTILLERY WORKS, TURIN

CHARGES.			COMPOSITION OF STEEL.						MECHANICAL TESTS		Steel Produced, Kilograms	Kilowatt Hours per Ton
No.	Composition.			C per cent.	Mn per cent.	Si per cent.	P per cent.	S per cent.	Tensile Strength, Kg/mm².	Elongation per cent.		
18	Mixed scraps	350 kilograms	Maximum	0.525	1.54	0.20	0.035	0.040	83	20	11749	1250
	Scrap in bundles	200 "	Minimum	0.40	1.224	0.15	0.025	0.027	67	14		
	Steel turnings	100 "	Average	0.451	1.388	0.176	0.030	0.033	73	16		
	Ferro-silicon	2.50 "										
	Ferro-manganese	10.50 "										
	Aluminium	0.20 "										
	Lime	6 "										
	Ore	1 "										

TABLE V.—200 HORSE-POWER ROTATING AND TILTING FURNACE OF THE STASSANO THERMO-ELECTRIC COMPANY (WORKING ON SOFT STEEL CASTINGS)

CHARGES.			COMPOSITION OF STEEL.					MECHANICAL TESTS		Kilowatt Hours, per Ton.
No	Composition.			C per cent.	Si per cent.	S per cent.	P per cent.	Mn per cent.	Tensile Strength, Kg/mm².	Elongation per cent.	
6	Steel scrap	330 kilograms	Maximum	0.260	0.330	0.052	0.038	1.020	45.0	19.0	
	Steel and iron turnings	330 "	Minimum	0.200	0.090	0.040	0.025	0.410	42.0	17.5	
	Steel, very rusty	70 "	Average	0.235	0.195	0.047	0.031	0.546	43.5	18.2	1260
	Haematite pig	18 "									
	Ferro-silicon	2.2 "									
	Ferro-manganese	6 "									
	Lime	30 "									
	Carbide of calcium	4 "									
8	Cast steel scrap	250 kilograms	Maximum	0.270	0.490	0.051	0.031	0.460	43.0	20.5	
	Sheet steel scrap	350 "	Minimum	0.140	0.100	0.040	0.025	0.390	39.8	17.0	
	Turnings	130 "	Average	0.207	0.236	0.043	0.027	0.421	41.2	19.0	
	Turnings, very rusty	70 "									
	Haematite pig	28 "									
	Ferro-silicon	3.43 "									
	Ferro-manganese	3.20 "									
	Lime	30 "									
	Carbide of calcium	4 "									
3	Cast scrap	200 kilograms	Maximum	0.500	0.250	0.058	0.040	0.610	45.0	19.4	
	Sheet steel scrap	400 "	Minimum	0.280	0.080	0.045	0.035	0.360			
	Turnings	150 "	Average	0.360	0.140	0.052	0.036	0.486			
	Turnings, very rusty	70 "									
	Haematite pig	60 "									
	Ferro-silicon	2 "									
	Ferro-manganese	3.60 "									
	Lime	35 "									
	Carbide of calcium	5 "									
9	Cast scrap	400 kilograms	Maximum	0.300	0.300	0.054	0.038	0.680	57.0	19.0	
	Turnings	200 "	Minimum	0.220	0.090	0.040	0.025	0.400	42.0	14.0	
	Turnings, very rusty	150 "	Average	0.240	0.196	0.046	0.037	0.530	48.0	17.0	
	Haematite pig	45 "									
	Ferro-silicon	1 "									
	Ferro-manganese	4.3 "									
	Lime	40 "									
	Carbide of calcium	5 "									

TABLE VI.—1000, HORSE-POWER FURNACE OF THE STASSANO THERMO-ELECTRIC STEEL COMPANY (STEEL FOR PROJECTILES)

CHARGES.			COMPOSITION OF STEEL.						MECHANICAL TESTS.		Kilowatt Hours per Ton.	
No.	Composition.			C per cent.	Si per cent.	S per cent	P per cent.	Mn per cent.	Tensile Strength, Kg/mm².	Elongation per cent.		
4	Steel turnings	2600.0 kilograms	Maximum	0.32	0.380	0.040	0.025	0.950	61	23		Fixed Furnaces.
	Soft scrap	1400.0 "	Minimum	0.300	0.150	0.035	0.022	0.780	57	19.8		
	Ferro-silicon	9.500 "	Average	0.310	0.242	0.037	0.023	0.877	58	21.4	958	
	Ferro-manganese	21.500 "										
	Aluminium	4.0 "										
	Rolling-mill scale	30.0 "										
	Lime	60.0 "										
	Carbide of calcium	15.0 "										
4	Steel turnings	2600 kilograms	Maximum	0.470	0.085	0.041	0.031	1.300	71	18		Rotating Furnaces.
	Soft scrap	1400 "	Minimum	0.420	0.046	0.038	0.027	1.150	69	16.5	918	
	Haematite pig	130 "	Average	0.450	0.061	0.039	0.029	1.211	70	17.1		
	Ferro-silicon	130 "										
	Ferro-manganese	41 "										
	Aluminium	4 "										
	Rolling-mill scale	30 "										
	Lime	60 "										
	Carbide of calcium	15 "										

TABLE VII.—SPECIAL STEELS. ROTATING AND TILTING FURNACES OF 200 HORSE-POWER

No.	Products.	Composition of steel product. C per cent.	Si per cent.	S per cent.	P per cent.	Mn per cent.	Mechanical tests. Tensile Strength, Kg/mm².	Elongation per cent.	Kilowatt Hours per Ton.	
	Nickel steel	0.180	0.123	0.048	0.007	0.780	56	29	1180	Ni = 5.60
		0.165	0.110	0.015	0.007	0.690	54.5	30.1	1200	Ni = 5.56
	High speed steel	1.560	0.062	0.012	0.010	0.979	.....	.....	1150	W = 1.27; Cr = 3.5
	Tungsten steel	0.960	0.123	0.018	0.018	0.600	.....	.....	1195	W = 1.560

quite low. They are even mallable with the following extreme compositions:

	Per cent.
Carbon	0.07 to 0.68
Silicon	0.015 " 0.10
Sulphur	0.018 " 0.007
Phosphorus	0.008 " 0.006
Manganese	0.006 " 0.08

The Kjellin furnace of the Gregorini Company is a simple induction furnace taking 300 kw. It is supplied with single-phase current at 25 cycles. When working on cold charges the production is about 10 tons daily, the energy consumption in that case being 690 kw-hours per ton of steel produced. The patent furnace lining of the Poldihütte is used, which lasts for about 120 heats, during the first 80 of which exceptionally rapid working is possible. High speed steel can be manufactured in this furnace containing, say, 25 per cent of tunsgten and 6 per cent of chromium.

II. Production of Pig Iron in the Electric Furnace.

Experiments made in Italy in the smelting of pig iron in the electric furnace differ somewhat from those made in other countries with the same object. In the trials made at Sault Ste. Marie, on behalf of the Canadian Government, a full account of which was given in Dr. E. Haanel's report in 1907, iron ores were smelted in a Héroult furnace. Dr. Haanel referred in his previous report to the First Canadian Commission in 1904 to the manufacture of electric steel, in which he described his experiments with the Héroult process at La Praz and with the Keller process at Livet. That report deals only with experiments, and not with the manufacture of steel on an industrial scale, such as that which has been attempted by the Swedish Company Aktiebolaget Elektrometall, which, having ample financial resources at its disposal, has produced many tons of pig iron in the furnaces of Lindeblad, Stalhane and Grönwall. Ordinary grades of iron ore were used, both in the experiments described by Dr. Haanel and also in the smelting of pig iron on an industrial scale by the Swedish Company.

In Italy experiments have been carried out by Carcano[2] for the production of pig iron from the residues of the pyrites which abound in this country. The pyrites residues can always be used in the blast-furnace, but thorough purifying and enrichment, as well as agglomeration by special processes, are necessary to prepare them for treatment in the blast-furnace. The chemical composition of the natural pyrites residues varies considerably, but what chiefly differentiates it from ordinary iron ore is its high sulphur percentage, often exceeding 4 per cent.

The pyrites residues treated by Carcano in the electric furnace had the following composition:

	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Silicon	8.0	12.25	9.10	7.9
Iron	56.0	47.0	52.0	60.3
Aluminum	9.30	18.80	12.30	6.9
Sulphur	2.01	2.63	4.25	2.01
			Per Cent.	

The residues were charged in their natural powdery condition, and it was endeavored to work with the most basic slag possible, preferably with a multiple base and with a certain percentage of manganese, so as to ensure that the slag had a sufficiently strong desulphurizing effect. The necessity of highly basic slag has been much discussed when it is desired to obtain a very energetic desulphurizing action, but, generally, manganese is found to have a sufficiently desulphurizing influence. This was also evident in the experiments under consideration, as may be seen from the following chemical determinations made on the raw material and on the pig iron produced under these conditions:

	Per cent.	Per cent.	Per cent.	Per cent.	Per cent.
Manganese	0.558	0.744	1.28	2.14	2.54
Sulphur	0.146	0.124	0.093	0.048	0.026

The production of pig iron amounted to something under 100 tons, and the quality of the product may be judged from the following analyses:

Sulphur.	Phosphorus.	Manganese.	Silicon.
Per cent.	Per cent.	Per cent.	Per cent.
0.046	0.063	2.23	3.83
0.058	0.075	2.17	4.46
0.015	0.073	2.26	6.70
0.026	0.016	2.54	5.70 carbon = 1
	Ferro-Alloys.		
0.015	0.060	11.25	3.30
.....	0.060	13.66	8.90

In sample No. 4 of the foregoing series of analyses the carbon amounted to only 1 per cent, while the manganese and silicon were much higher. The material might be classed as a special kind of steel rather than as a pig iron.

In these trials various types of electric furnace of different powers were used. The first furnace was 180 hp, while that tried in 198, which Carcano found to give the most favorable results, was of a capacity varying from 200 to 300 hp. It was a closed furnace with neutral hearth supplied with single-phase or three-phase current with automatic charging, and with recovery of the carbon monoxide evolved during the reactions. That gas was used for the purpose of preventing deoxidation of the pyrites residues.

The energy consumption per ton of pig produced averaged 2100 kw-hours in a furnace of 180 kw and using pyrites residues containing 50 to 55 per cent of iron in the form of ferrous oxide.

The lower ends of the vertical electrodes rested on the slag, so that the furnace worked as a surface resistance furnace and the slag was of very high temperature, much above that of the bath. By this means it was possible to obtain highly fluid slags sufficiently basic in character, which accounts for the ability to desulphurize sufficiently when using materials containing so high a proportion of sulphur.

As regards the economic aspect of the process, too many data are unfortunately lacking, such as electrode consumption, charges for cooling water, refractory material for lining, and labor, to make it possible to present a practical estimate. There is to be taken into account, besides, the cost of raw material and of transport, the pyrites, at least in Italy, being generally situated at a considerable distance from the regions where water-power is plentiful. On the other hand, the districts abounding in water-power are far from the markets for metallurgical products and from the seaports.

III. Ferro-Alloys.

The chief metallurgical products manufactured in the electric

[2]This journal, April, 1909 (Electrochem. & Met. Ind., vol. VII, p. 155).

furnace in large quantities are ferro-alloys. Some, such as ferro-manganese and ferro-silicon low in silicon, were already known before the introduction of electric smelting, but with the electric furnace it is possible to produce alloys with a carbon content very much lower than that obtainable when smelting these alloys in the blast-furnace. The production of alloys such as high grade ferro-silicon, ferro-chromium, ferro-tungsten is only practicable in the electric furnace, and these are manufactured either from the ores of the respective metals or direct from the metals themselves or from the metals and ores together. The only ferro-alloy electrically manufactured in Italy on an important scale is that of ferro-silicon, of which 628 tons were produced in 1909. It may be useful, nevertheless, to describe briefly some of the experiments which have been made.

A. *Ferro-Alloys from Ores.*—As already stated, Mr. Carcano has obtained a kind of spiegel and silico-spiegel using pyrites residues. The author of the present paper has made some experiments for the purpose of studying the economic possibility, under certain given conditions, of producing ferro-silicon by smelting, in the electric furnace, silicates of iron or iron ores too highly siliceous for treatment in the blast-furnace, also cast or pig iron that has been rejected on account of its high sulphur or phosphorus contents.

The electric furnace used for the experiments was one of those specially modified by the author, the special characteristics of which are the absence of any kind of flexible conductor, a minimum quantity of current passing through the conductors and, above all, through the electrodes, a minimum space taken up by the conductors and confinement of water-cooling to those parts of the furnace above the plane of the electric arcs.

As a result of careful attention to these details the furnaces, which have also been used, with suitable modifications, for the manufacture of carbide of calcium, have a high thermal and electric efficiency and consequently a high total of efficiency, whatever kind of current is employed.

With alternating current, which is now generally used in modern metallurgical works, the power factor of such furnaces approaches as near as practicable to unity. The current supplied to the furnace is one of 12,000 amp at 42 volts, 25 cycles. The production in round figures was about 20 tons of ferro-silicon of varying silicon percentages.

At first a silicate of iron was used of the following chemical composition:

	Per cent.		
Silica	25		
Ferrous oxide	56.22	Iron	43.85
Alumina	2.55		
CaO_3	12.50	Lime	7
$MgCO_3$	2.10	Manganous oxide	1
Moisture and impurities	1.63		

The ratio of ferrous oxide to silica, in the specimen treated, was $56.22/25 = 2.25$.

One difficulty consisted in estimating beforehand what weight of silica would pass into the slag. Supposing that the slag formed a bisilicate, $SiO_2 + MgO$, $SiO_2 + Al_2O_3$ ($3SiO_2$), the silicon of which amounted to 28 per cent, then on the basis of that composition and the percentages of CaO, MgO and Al_2O_3 contained in the ore the quantity of silica taken up by the slag would be as follows:

$$SiO_2 = 2.55(Al_2O_3)\frac{180}{102} + 7\ (CaO)\frac{60}{56} + 1\ (MgO)\frac{60}{40} = 4\ 5 + 7.53 + 1.50 = 13.53\ (SiO_2).$$

So that of the 25 units of SiO_2 contained in the ore only 11.47, corresponding to 5.39 of silicon, will have remained for the ferro-silicon alloy. On the other hand, the ferrous oxide would have yielded 43 units, making a total of 48.39 units of ferro-silicon. The percentage composition of the ferro-silicon obtained would thus consist of:

	Per cent.
Silicon	11
Iron	89

According to the known formula for the reduction, by means of carbon, of ferrous oxide and silicon, and according to the analysis of the ore, the fixed carbon necessary for 100 kg of ore would amount to

$$56.22\ (FeO)\frac{12}{72} + 11.47\ (SiO_2)\frac{24}{60} = 13.94,$$

and if the metallurgical coke at disposal contained 85 per cent of fixed carbon, the weight of coke required would be

$$\frac{13.94}{0.85} = 16.4.$$

The weight of the slag per 100 parts of the ore would be:

$$2.55\ (Al_2O_3) + 7\ (CaO) + 1\ (MgO) + 13.53\ (SiO_2) + \text{ash}\ (16.4 - 13.94) = 26.54$$

In the absence of an analysis of the ash of the coke, and in consideration of its small amount, it may be assumed that the chemical composition of the slag would be that corresponding to the formula of the supposed bisilicate. Its composition would therefore be

$$Al_2O_3\ (3\ SiO_2) = 2.55 + 4.50 = 7.05$$
$$CaO,\ SiO_2 = 7 + 7.53 = 14.53$$
$$MgO,\ SiO_2 = 1 + 1.50 = 2.50,$$

and the proportion of the several constituents would be:

	Per cent.
Al_2O_3	10.6
CaO	29.1
MgO	4.2
SiO_2	56.1

According to the triangular diagram of Boudouard of silico-aluminates of calcium, the melting temperature of such a slag is about 1350° C., which is considerably lower than that of silicon, at 1900° (according to Boudouard); of alumina, at 2200° (according to Richards); and much lower than that of calcium, 3000° (according to Moissan). The slags become in fact quite fluid, and are very easily poured off and separated from the ferro-silicon.

The melting temperature will be even lower than the theoretical, since the material also takes up a certain proportion of ferrous oxide of the furnace lining and of burnt electrodes.

The charge, consisting of 100 parts of ore and 16.4 parts of coke, should yield 48.34 kg of 11 per cent ferro-silicon, and about 27 kg of slag with a silicon content of $56.1 \times \frac{28}{60} = 26.32$ per cent; whereas the average of the analyses showed that the silicon in the slag was 23.5 per cent, and in the ferro-silicon 16.5 per cent. The weight of the slag was greater than the calculated weight.

In order to increase the silicon content quartz of the following composition is added:

	Per cent.
SiO_2	91
FeO	2.6
Al_2O_3	4.10
CaO	0.9

By this means alloys of increasing percentages of silicon can be produced, up to a maximum of 25 per cent.

Still richer alloys were obtained with the same quartz, but substituting for the silicate of iron an ore of the following composition:

	Per cent.		
Ferrous oxide	75.5	Iron	57.3
Silica	9		
Alumina	4		
Lime	0.84		
Manganous oxide	1		
Moisture and losses	9.4		

There were produced 9400 kg. of ferro-silicon. Of a 40 per cent alloy 5400 kg. were manufactured, the following materials being used per 1000 kg.

	Kilograms.
Ore	1,300
Quartz	2,330
Coke	1,000

The yield per kw-hour was 0.143 kg., and per kw-year 1.444 tons.

The electric energy was measured on the high tension connection at the switchboard, so that the pressure at the terminals

on the furnace may be taken as 5 to 8 per cent lower. The year is taken at 8000 working hours.

During these trials slags were obtained rich in silicon, even up to 38 per cent. Assuming again that such a slag was a bisilicate, the silicon by weight would have been

$$4\frac{180}{102} + 0.84\frac{60}{56} + 1\frac{60}{40} = 9.46$$

the slag by weight would have been 15.30, and its chemical composition would be Al_2O_3 = 26.1 per cent, CaO = 5.4 per cent, MgO = 5.5, SiO_2 61.8 per cent. The melting temperature of such a slag is at least 1500° according to Boudouard.

The remarkable quantity of quartz consumed accounts for the refractoriness of the slag. The furnace worked at a high temperature, and there was a noticeable volatilization of the silicon. Another series of results was similarly interpreted by the author. An ore was treated, consisting of the following composition:

	Per cent.
Ferrous oxide	33
Silica	56
Alumina	5
Lime	1

By means of calculations similar to the foregoing it was reckoned that the slag, without making additions, would have the composition, silica = 62 per cent, alumina = 32 per cent, lime = 6 per cent, the melting point of such a slag being above 1600°. The consumption of raw materials was somewhat higher than the theoretical amount. The ore actually contained 25.74 per cent of iron, 26.32 per cent of silicon, of which 4.65 passed into the slag. Hence for every 100 kg of ore there should be produced 25.74 + 21.67 = 47.41 kg of ferro-silicon. That is, for 100 kg of ferro-silicon, the ore consumption should amount to 211 kg, whereas instead it amounted to 356 kg.

From the results tabulated above, the efficiency of the electric furnace may be deduced. The following thermoelectrochemical data constitute the necessary factors:

Energy of dissociation.—(SiO_2) = 180,000 calories = 3,000 calories per kg. of SiO_2:

= 6,384 calories = 7.4 kw. hrs. per kg. of silicon.
(FeO) = 657,000 calories = 1.36 kw. hrs. per kg. of iron.

Heat Units required for Heating.—The melting temperature of anhydrous silica is about 1900°, but for the quartz used containing 91 per cent of silica the melting temperature may be assumed, on account of the impurities, to be not higher than 1800°.

Per kilogram of quartz from 0° to 1,800°, there are required 761 calories (Richards)
Per kilogram of ferrous-oxide from 0° to 1,800°, there are required 887 calories (Richards)

This latter item is based on an analogous calculation made by J. W. Richards.[1]

The mean specific heat of the carbon between 0° and 1800°

$$\frac{120}{1800} = 0.433.$$

is 0.5 —

On the basis of these figures, the energy required for the production of 1 kg of 40 per cent ferro-silicon from ores may be calculated as follows:

Energy of dissociation per 0.40 of a unit of silicon . . . = 0.40 × 7.4 = 2.96 kilowatt hours,
and per 0.60 of a unit of iron . . . = 0.60 × 1.36 = 0.816 "

Total . . . 3.78 "

Heating
Ores . . . 1.3 × 887 = 1,153 calories
Quartz . . 2.33 × 761 = 1,773 "
Coke . . . 1 × 0.433 × 1,800 = 779 "

3,705 "

Less—Coke, with 85 per cent. carbon, burnt to CO, yielding . . . 2,430 × 0.85 = 2,065

1,640 " = 1.9 kilowatt-hours

864

Total kilowatt-hours = 1.9 × 3.78 = 5.68

[1] "Metallurgical Calculations," part I.

The theoretical production per kw-hour is therefore

$$\frac{1}{5.68} = 0.176.$$

The actual production per kw-hour was 0.143.

The efficiency of the furnace, therefore, equals

$$100 \times \frac{0.143}{0.176} = 80 \text{ per cent.}$$

The efficiency of the kw-year, like that of the electric, is, of course, higher, when manufacturing ferro-silicon from quartz and scrap-iron, and the use of ore or scrap is governed by the cost of the raw material and of the electric energy. In Italy, where the cost of the kw-year is generally comparatively low, it is more economical to use ore, and in order to make the question more clear the following calculations relating to the energy necessary to produce 1 kg of 40 per cent ferro-silicon, using scrap, are given for comparison with the preceding figures:

The loss of silicon is assumed to amount to 5 per cent.

Quartz required (SiO_2 90 per cent) = $\frac{0.4 + 0.05}{0.47 \times 0.9}$ = 1 kilogram.

Scrap required (Loss = 10 per cent) = 0.66 kilogram

Energy of dissociation as above = 3.78 kw-hours. Heat units required:

Scrap. 450 calories per kg. = 450 × 0.66 = 297 calories
Quartz. 1 × 761 . . . . . = 761 "
Coke . . . 0.7 × 0.433 × 1,800 . . . . . 545 "

1,603 "

Less—Carbon burnt to CO, yielding 0.85 × 0.7 × 2430 . . . . . = 1,446 "

157 calories = 0.18 kilowatt-hours.

The energy required per kg. of 40 per cent ferro-silicon is therefore 3.78 + 0.18 = 3.96 kw-hours.

The theoretical production per kw-hour = $\frac{1}{3.96} = 0.252$,

and the efficiency of the furnace is 80 per cent.

The actual production = 0.252 × 0.8 = 0.20 kg per kw-hour = 4.8 kg per kw-day.

Hence the consumption of energy in the ore process exceeds that in the scrap process by 1000 (5.69 — 3.96) = 1730 kw-hours per ton of 40 per cent ferro-silicon. The extra cost of this is more than saved in the cheaper raw material, especially with careful attention to the slags and to the temperatures of the thermo-chemical reactions.

With the object of reducing the slag the author has tried to produce ferro-silicon by the partial substitution of pig iron containing 94 per cent of iron for the ore. The raw materials consumed per 1000 units of the product were: ores, 520; quartz, 2200; coke, 540; anthracite, 300. With this mixture some 4000 kg of 42 per cent ferro-silicon were produced. No additions were made, and the output was low on account of the considerable loss of silicon through volatilization.

B. *Ferro-Alloys from Metallurgical Products.*—The greater part of ferro-silicon manufactured in Italy is produced from quartz and scrap iron. The first works in Italy which manufactured ferro-silicon on an industrial scale were those of the Societa Elettrochimica at Bussi. The production in 1907 was 175 tons; in 1908, 808 tons; and in 1909, 606 tons.

In 1909 the Piedmontese Company for the Manufacture of Carbide of Calcium and Allied Products produced 22 tons of ferro-silicon. According to the particulars officially published, the electric furnace for smelting ferro-silicon is absolutely identical with that used for the production of carbide of calcium, and works with the same current at the same pressure.

The author has also experimented in the production of ferro-silicon, using pig iron in place of scrap, the price of which latter material is very high in Italy. The furnace was of the same type as that mentioned in the preceding paragraph. Waste pig iron was used broken to the size of nuts. The mixture con-

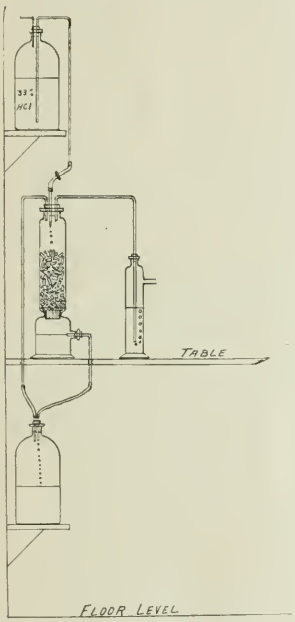
sisted of 1000 parts of pig, 3000 of quartz, and 1300 of coke. The product averaged about 65 per cent of silicon, and the total yield was somewhat under 3 tons. The raw materials used per 1000 kg of product amounted to 620 kg of pig iron, 1775 of quartz, and 810 of coke. The yield corresponded to 2.112 kg of ferro-silicon per kw-day.

Rome, Italy.

Apparatus for Making H_2S.

BY J. R. HUBER.

The accompanying sketch illustrates a convenient and satisfactory method for making H_2S in the laboratory. A Kipp generator, owing to the pressure necessary to hold up the acid when not in use, has a tendency to leak around the stoppers and consequently generates most of the time, being a continual source of annoyance besides wasting acid and iron sulphide.



APPARATUS FOR MAKING H_2S.

Hydrochloric acid, about 1 to 2, is placed in the upper bottle, the syphon filled and the flow of acid adjusted by means of a screw pinch cock. When no more gas is required the pinch cock is screwed up tight. The generation of gas ceases almost immediately.

By using an 18-in. tower the acid trickles over sufficient FeS to be entirely neutralized when it flows out at the bottom. A large cork with several small holes punched through it serves to hold up the FeS and permits the waste acid to flow into the reservoir below.

Two holes are drilled through the table top, one for the tube which carries away the waste acids and the other for a tube connecting the waste acid bottle with the top of the tower. This tube relieves the pressure in the waste acid bottle, permitting it to be tightly corked, thus preventing any odor of the gas from getting out into the room. The other bottle contains pure water for washing the gas, and after it may be placed whatever purifying or drying apparatus is considered necessary for the work in hand.

FeS in fused sticks is the most convenient form to use. The apparatus is best set up in the hood against one of the walls.

Lumen Bearing Co., Buffalo, N. Y.

Electrolytic oxygen is proposed by T. H. Aldrich, Jr., as a means of keeping cyanide solutions in an oxidizing condition. Heretofore we have depended on oxygen dissolved from the air during agitation, or on the addition of salts known to be strong oxidizing agents, for the supply of oxygen in cyanide solutions. Mr. Aldrich has found that if lead electrodes be suspended in the pulp under treatment and a current of 0.25 amp. passed through them an oxidizing condition will be maintained which effectually aids in dissolution of the gold contained in a reducing ore.

Platinum Laboratory Utensils.

At the Minneapolis convention of the American Chemical Society in a session of the Division of Industrial Chemists and Chemical Engineers, a Committee on Quality of Platinum Laboratory Utensils was appointed, which presented a preliminary report at the Indianapolis convention this summer. This committee consisted of Dr. WM. F. HILLEBRAND, of the Bureau of Standards; Dr. PERCY H. WALKER, of the Bureau of Chemistry, and Dr. EUGENE T. ALLEN, of the Geophysical Laboratory of the Carnegie Institution.

The object of the work of the committee was to study the alleged inferiority of platinum ware made in recent years as compared with that procurable many years ago.

The objections made to inferior ware are:

(1) Undue loss of weight on ignition (generally due to a content of iridium and perhaps of base metals of low boiling point).

(2) Undue loss of weight on acid treatment, especially after strong ignition (due essentially to a content of iron and calcium, which are being brought slowly to the surface and there oxidized, so as to give in extreme cases a reddish coating of ferric oxide on one hand and an alkaline reaction on the other).

(3) An unsightly appearance of the surface of the ware after strong ignition, especially in the early stages of heating (sometimes due to volatile impurities, but later caused by crystallization of the platinum, the important question arising whether this crystallization is due to the presence of an impurity).

(4) Adherence of crucibles and dishes to triangles, sometimes to such an extent as to leave an indentation of the vessel at the points of contact with the triangle, even when complete cooling has been reached before the two are separated from each other (possibly due to an inferior quality of the triangle as contrasted with the vessel).

(5) Alkalinity after ignition. (Compare above under 2.)

(6) Blistering, a defect of long standing and of somewhat infrequent occurrence.

(7) Development of cracks after continued heating, a defect "of far too common occurrence."

The committee asked for the co-operation of American manufacturers of platinum ware and received this co-operation by interviews with special representatives of these manufacturers, sent to Washington, and by the supply of platinum ware for the investigation of the committee. These samples were compared among themselves and with foreign-made ware as to behavior on heating, both as regards appearance and stiffness, loss in weight, and tests for iron and alkali. As standards for comparison were taken one crucible from Heraeus, one of his own make, of "Thermoelement" platinum, which is claimed to be free from all impurities, and three of his "Tiegel" platinum, containing, according to the latest information, not over 0.05 per cent iridium and probably less.

The committee in its preliminary reports describes the experiments made as follows:

"Surface Aspect of Ware after Heating.—Leaving out of account the discoloration due to the oxidation of iron, the surface indications shown by ware that has come under the observation of the committee at different times are not easy to describe.

"With good ware the crucible becomes uniformly affected by the blast when this envelops the whole vessel, either taking on a whitish, frosted appearance or becoming uniformly mottled inside and out with crystal plates which do not all reflect the light alike. There is great difference in the size of these plates, to almost microscopic in the case of the frost-like aspect noted from those that are easily apparent to the naked eye. Sometimes, when the plates are large, a decided blue color is imparted to the surface. The crucible of 'Thermoelement'

platinum of Heraeus which was tested showed only the frosted surface, while his 'Tiegel' platinum became in every case coarsely marked. No explanation for this difference has yet been offered, a difference that seems surprising in view of the supposed very slight differences in composition of the two.

"On poor ware the surface becomes more or less whitish, sometimes faintly brownish in places, but the discolorations are not uniformly distributed, and they often impart an unsightly appearance. As a rule, the lower part of the crucible, where it was hottest, remains bright, and from it stretch out more or less fan-shaped areas, the direction of which is affected by that of the flame, i. e., whether this was inclined or applied vertically. The areas often have the appearance of sublimates, such as are often produced from charcoal before the blowpipe. Now and then they have been seen starting sharply from a point. On direct and continued application of the flame their appearance and location may undergo change, and they may even disappear entirely.

"Surface Alkalinity."—Concerning this single phenomenon, which was first observed by Dr. H. C. P. Weber, of the Bureau of Standards, before the committee was organized, little definite information has thus far been gained. When manifest it can be detected by applying moist red litmus paper to the surface of the vessel after it has been blasted for several hours. It has been noted in varying degrees of intensity and is not always apparent. The element producing the alkalinity is calcium. Its sources might be lime from the crucibles used for melting the platinum or a lime salt used in polishing. Opinions as to this were conflicting, and the committee has not yet enough evidence at hand to warrant an expression of opinion of its own. The question is to be tested in part by comparison of ware that has been polished with Vienna lime and that which has not been so treated. What effect, if any, the presence of lime has on the quality of the ware is not known, although the observations of Mylius and Hüttner[1] and of Hillebrand,[2] have direct bearing on this point.

"Iron Content."—Rouge is sometimes used for polishing ware, and steel enters into its manufacture at more than one stage. These may account for a portion of the iron content of all ordinary ware, and for the minute traces that may occur in the specially refined ware, but the evidence all goes to show that the introduction from this source (unless possibly when hot rolling of sheet and hot shaping of utensils are practised) accounts for but a small fraction of the amount actually present.

"The committee has found that the 'Thermoelement' and 'Tiegel' platinum of Heraeus are satisfactory with respect to iron. It is either not detectable after long blasting or gives such a faint reaction as to represent a negligible quantity. One American firm has submitted what is claimed to be its own refined ware, which seems to be with respect to iron the equal of the 'Tiegel' platinum of Heraeus.

So far as the experience of the members of the committee goes, all the ware supplied by the American manufacturers as of their own make during recent years, has been strongly contaminated by iron. Sometimes its amount has been so great as to reveal itself after less than a half hour's exposure to a Bunsen burner flame, by a darkening, even reddening, of the inside of the crucible. More often the color change becomes pronounced only after long blasting. In an extreme case as much as 6 mg. of iron oxide were removed from the surface of a 22-gram dish of German make some years ago by one of the committee members. An amount less than 1 mg. in weight suffices to impart a very dark color to a crucible of 20 cu. cm. capacity. Even when the iron does not reveal itself thus, treatment with pure hydrochloric acid or with alkali pyrosulphate will often do so. Renewed heating will invariably bring more iron to the surface of the platinum if originally present. If the heating is done electrically instead of over a blast flame the color change due to iron oxide appears outside as well as inside the crucible. In one case a few years ago a dark color due to iron oxide was observed on ware from the maker.

"Loss in Weight on Ignition."—Aside from a content of volatile base metals, iridium is supposed to be the volatile constituent of platinum which contributes chiefly to the observed loss of weight on strong ignition. This belief is supported by the experience of the committee.

"The following set of tests on the ordinary commercial ware submitted by one firm is perhaps typical in general for a grade of ware that is supposed (from the statements made to the committee) to carry about 0.05 per cent iridium, but further confirmative tests must be made before final conclusions are reached.

"A crucible, weighing without cover about 16 grams, lost in four hours over a vertical blast 3.4 mg. The temperature inside the crucible and near the bottom was 1180°-1190°C. The rate of loss per hour was 0.85 mg. At 1100° the loss in less than four hours was 1.2 mg., or at the rate of somewhat over 0.3 mg. per hour.

"Another crucible of the same lot, but slightly heavier, suffered a loss over the blast in 3 hours at 1200°-1210° of 2.3 mg. (0.77 mg. per hour) and of 1.9 mg. in addition at 1100° in 4.5 hours (0.42 mg. per hour). The surface of the crucible was very unevenly and strongly frosted, in marked contrast to the crucible of refined metal.

"Other tests by one of the committee, on purchased domestic ware, showed losses of the same order, in some cases fully 1 mg. per hour at or near 1200°.

"Contrasted with these, the 'Tiegel' platinum of Heraeus above referred to shows to decided advantage. A crucible weighing about 16 grams lost in 12 hours (heatings of 4 hours each) 2.3 mg. (0.2 mg. per hour) at temperatures not exactly known but approximating 1200°. At 1100° the loss was 0.9 mg. in 8 hours (0.11 mg. per hour). Another lost at the same rate as the first in 8 hours at 1190°-1210°.

"Marked improvement was shown also by crucibles made of metal specially refined in this country, free from iron, and submitted by one firm, the rate of loss per hour being 0.3 mg. at 1200° in 12 hours, and 0.1 mg. at 1100° in 8 hours. The losses in the several periods of heating at 1200° were irregular, due perhaps to marked variations in atmospheric conditions, causing errors in weighing.

"The crucible of 'Thermoelement' platinum lost in 14 hours at 1200° 2.3 mg. (0.17 mg. per hour), and in 4 hours, at 1100°, 0.2 mg. (0.05 mg. per hour).

"The triangles furnished with the crucibles, and of the same composition, suffered corresponding losses.

"The last test shows that at the temperatures employed and over a blast-lamp even platinum that is claimed to be free from iridium has a perceptible vapor pressure or forms a volatile oxide, and it is also apparent that for chemical use the 'Tiegel' platinum seems to be quite as satisfactory in quality.

"The foregoing data may be summarized as follows:

	Rate loss per hour in mgs.	
	1200°	1100°
Ordinary ware (0.5 (?) per cent Ir.)	0.85	0.3
'Tiegel' platinum	0.2	0.11
'Thermoelement' platinum	0.17	0.05
American refined ware	0.3	0.1

"These data are not to be taken as rigidly exact, but they are probably fairly indicative. It should be remembered that the crucibles were not polished or wiped between the separate heatings.

"From the above and other tests it may be inferred also that there is a sharp rise in the volatility curve for iridium, and apparently for platinum also between the temperatures 1100° and 1200°. This point it is the intention of the committee to investigate further. It is true, it would seem that for temperatures up to 1100° the volatility loss is of little significance with refined

[1] Ztschr. Instrumentenkunde, 30, 190 (1910).
[2] Bull. 422, U. S. Geol. Survey, p. 122.

ware, but that it may be of some moment with ordinary chemical ware low in iridium and free from volatile base metals. At higher temperatures and with long-continued heating, however, the losses are appreciable even with the purest ware and are very high with commercial ware.

"In this connection it is proper to say that in all probability the maximum temperature of the blast used in the above experiments is above that commonly attained in most laboratories. The gas used was artificial. Attention may be called also to the fact that relatively few substances obtained in chemical analysis require blast ignition and that blasting of an hour's duration is seldom necessary, so that in the majority of cases the volatility loss may be ignored.

"Relative Stiffness and Hardness of Ware.—Pure platinum is soft and lacks stiffness. For these reasons it has been common practice to leave a certain amount of iridium in ware made from it or even to add to the amount already present. This addition is especially pronounced in triangles, which may contain 15 to 20 per cent of iridium. The crucibles of 'Tiegel' platinum made up in this country and also those of American-refined metal showed lack of stiffness in a very objectionable degree. In contrast with them was the crucible of 'Thermoelement' platinum which, though not so stiff as the crucibles of commercial ware, was sufficiently so for all practical purposes. The reason for this difference is not yet apparent, but it may be due to the different shape of the crucibles or to a different thickness of the metal. If so the defect mentioned in the American crucibles tested is one that can be easily remedied by the makers.

"Triangles.—In order that no error might be introduced by the use of triangles differing in composition from the crucibles, whereby iridium or other metal might pass from the former to the latter, the makers were asked to furnish with the crucibles triangles of the same composition, and this they did.

"The refined crucibles and triangles were found to stick together badly while hot, owing to the welding property of pure platinum. They could be separated when cool, but often only by the expenditure of some force and with danger of tearing out metal from the crucibles. It is probable that, in order to help maintain as far as possible the purity of a vessel of highly refined platinum, it should not be used in conjunction with a platinum triangle of different quality. Sagging, to a much greater degree than with ordinary triangles that have been some time in use, was not observed. The sticking defect is serious enough to take into consideration with refined ware. Therefore, experiments were made to test the temperatures obtainable inside a crucible with triangles of different materials and shapes. Without entering into details, it may be said that with a vertical blast flame the differences were far less than had been expected between triangles of platinum, of clay and of quartz, so that except for particular uses platinum triangles seem to have little advantage over those of clay or quartz when used under the conditions of the tests. The clay triangles with projecting points on which the crucibles rest allow as high a temperature to be reached in the upper part of the crucible as those of platinum. Quartz triangles when made in one piece are to be had, but the cheaper ones of nickel or 'nichrome' covered with quartz tubing are probably just as serviceable.

"A word of caution may be added here with regard to the use of 'nichrome' in direct contact with platinum. It has not been shown that there is no contamination from this source, and in view of the cost of platinum ware any additional source of contamination is to be guarded against.

"Blistering.—The blistering of crucibles seems to be far less frequent now than it was some years ago. It is probable that this defect resulted from past methods of manufacture rather than from other causes, and the committee has not given experimental attention to the subject.

"Cracking—One manufacturer attributed cracking not to the faulty nature of the ware but to careless handling of magnesium-ammonium phosphate, etc., and to heating with reducing flames. Another treated the ware mechanically to lessen the tendency to crack.

"Extra Cost of Refined Ware.—The refining of platinum to the extent shown by the 'Tiegel' platinum of Heraeus and by the recent product of one of the American manufacturers necessitates a higher charge than we have been accustomed to pay for commercial ware. In the experience of those who have bought the refined ware in this country the extra cost is in the neighborhood of 30 cents per gram. This is a large relative increase, but not excessive in comparison with the advantage gained from having an article that one knows to be free from all objectionable impurities in significant amounts. If platinum can be refined to the extent of removing all or nearly all iron and other base metals without removing all the iridium, the presence of the latter might be no disadvantage when extreme temperatures are not called for. The matter is important, for stiffness is essential to large vessels, and this quality iridium undoubtedly imparts to platinum ware. That iron can be so removed without removing the iridium the following experience of one of the committee at the Bureau of Standards appears to show. For the sake of completeness data not relevant to the point in question are not included.

"Two crucibles, No. 65, bought in 1905, and No. 117, bought from another firm in 1908, but never used before, were first heated electrically at 1250° for several hours. No. 65 showed then no iron, or at most a trace, and lost weight subsequently over the blast at 1100°-1200° at the rate of 0.64 mg. per hour. Crucible No. 117 yielded 1.4 mg. iron oxide after the electrical heating. After cleaning, subsequent blast ignition caused a loss at the rate of 1 mg. per hour for the first 3 hours and of 0.9 mg. for the next 8 hours; the surface became strongly alkaline and showed more iron oxide. From the stiffness of the crucibles and their high loss in weight it is practically certain that both contained iridium, yet the one was free from iron, the other heavily charged. The original source of crucible No. 65 is not known."

The conclusions and recommendations of the committee are as follows:

"Although it were premature to offer anything final in the way of recommendations, the investigation has brought out certain facts and observations which the committee has deemed it wise to lay before the chemists in the foregoing pages without waiting for completion of the inquiry. It is hoped that further tests on domestic and foreign ware and a closer study of the literature will enable it to present a more finished report later.

"It has been shown that ware of foreign make can be had against which nothing can be brought on the score of iron or other base metal content and in which the loss caused by strong blasting seems to be solely of platinum itself and to be entirely negligible, for all ordinary purposes, up to at least 1100°.

"It appears also that American-refined platinum ware can be had of nearly, if not quite, as good quality, and there does not seem to be any reason to think that its quality cannot be still further improved if there is a call for it to be done.

"In order to obtain such ware it is only necessary to *specify that it must show no marked uneven discoloration on heating and must give no test for iron after prolonged ignition (two hours may be regarded as sufficient) and that the rate of loss per hour at 1100° C, over a period of not less than four hours, and preferably longer, shall not exceed, say, 0.2 mg.* The makers are granted a fair margin of safety in this latter item. Ware which does not meet these conditions should not be accepted if the purchaser is willing to pay the enhanced cost of the better ware.

"These specifications are subject to revision."

The Institute of Industrial Research, of Washington, D. C., of which Dr. Allerton S. Cushman is the director, announces the organization and equipment of a Division of Roads and Pavements under the supervision of Mr. Prevost Hubbard, formerly chief chemist in the Office of Public Roads in the Department of Agriculture.

Flow of Heat Through Bodies.

By Carl Hering.

In electric furnaces the economy of the heat losses is of far greater importance than in fuel furnaces, on account of the greater initial cost of the energy. In fact the cost of the energy is in many cases the criterion, hence a mere reduction of the losses may at times make the difference between commercial success or failure. It, therefore, becomes important in such cases to be able to proportion and design the walls of the furnaces so as to reduce these losses to the least that the particular conditions warrant.

To do this intelligently requires a knowledge of the physical laws and constants pertaining to the flow of heat through bodies and from surfaces. The term flow of heat is here used because it is a general one, which refers to all the heat, whether it travels by conduction, convection or by radiation, or any combinations of these; hence it is not necessary to specify the particular way, thereby avoiding ambiguities and errors, such as the very common one of calling the heat conducted through the walls "radiation" losses, instead of conduction losses, even when this heat is taken up by a water jacket, in which case there is no radiation whatsoever.

Although the chief laws concerning the flow of heat are fairly well understood, at least for conduction and radiation, our knowledge of the physical constants is still deplorably meager. The heat conductivities of many materials have been determined, but generally not for the ranges of temperatures occurring in furnaces, and often not for the very materials for which they are needed most in electric furnaces. Moreover, many of them are of questionable accuracy and they are rarely if ever stated in the most convenient way for furnace calculations, nor in the most convenient units. And as for the physical constants for the emission of energy from surfaces to the air, water, etc., including therefore radiation, convection and true conduction (if any), existing data is still more meager, vague and unsatisfactory.

The purpose of the present article is to give a compilation of such physical constants as the writer has been able to gather from various sources, and to present them in the form in which they are most convenient to use by the electric-furnace engineer. The writer does not vouch for their correctness and in fact doubts the accuracy of many of them; he assumes responsibility only for the reduction to the units here used. They are given here merely as a collection of existing data for the benefit of those who wish to use them as they are, and with the hope that such a table will be of sufficient interest to induce others to contribute other data, corrections or more reliable figures.

From the nature of furnace walls it is far simpler to use resistances and resistivities (specific resistances) in the calculations, instead of the older and more common conductances and conductivities (specific conductances). Hence the physical constants have here all been reduced to resistivities, in which form they are not generally given in books.

The units here used are those proposed and recommended by the writer some time ago, as by far the simplest to use in calculations, namely the watt, the centigrade degree and the thermal ohm. The watt (energy per second) is used for the flow of heat in preference to the calorie per second, because all power and energy calculations of the electrical engineer are preferably made in watts and watt-hours (or kilowats and kw-hours), hence this avoids troublesome reductions; moreover, it is an international unit. The centigrade degree is used because it is the more rational scale, is international, and because most thermal data in tables is given in that scale; moreover, it is, in the opinion of the writer, the moral duty of every progressive engineer to do what he can to discourage the use of that senseless, awkward, irrational and unfortunate inheritance, the Fahrenheit scale, which the more progressive nations have abandoned many years ago, if indeed they ever used it.

Lastly, the thermal ohm is used here (for the unit of thermal resistance), because it greatly reduces the calculations, in fact makes them the simplest possible. This new unit was proposed and recommended by the writer about a year ago.* Briefly described, the thermal ohm is that thermal resistance through which 1 watt of heat flow will pass when the temperature drop is one centigrade degree. Hence, if R is the thermal resistance in thermal ohms, H the flow of heat in watts and T the temperature in centigrade degrees,

$$H = T/R$$

which might be termed the thermal Ohm's law.

Moreover, if r is the specific thermal resistance, (resistivity) in thermal ohms per centimeter cube or inch cube, then

$$R = r\,L/S$$

in which L is the length and S the cross section of a thermal conductor, either centimeters or inches being used consistently throughout. This again is like the similar formula for electric resistance. Thermal resistivities, however, seem to have high temperature coefficients and the temperature ranges are great; the resistivities of refractory materials are said to decrease at the higher temperatures; this should be taken into consideration by using the modified value of r for that range of temperature when that data exists, or to allow for it when it is not known.

It will be seen, therefore, that by using these units no conversion factors are necessary, as all coefficients become unity. For the calculations of the electrical engineer the tabulated values of physical constants are therefore best reduced once for all to thermal ohms.

To reduce a thermal conductivity in gram calories per second to the resistivity in thermal ohms, when both are for 1 cm. cube, or for 1 in. cube, multiply the reciprocal of the conductivity by 0.238882. To reduce units in terms of 1 cm. cube to those for 1 in. cube, multiply by 0.393700 when they are resistivities, and by 2.54001 when they are conductivities. To reduce thermal conductivities given in terms of British thermal units (b.t.u. or pound Fahrenheit units) per hour, per square foot surface, per inch thickness, per degree Fahrenheit, to thermal ohm, inch cube units, multiply the reciprocal of that number by 273.013. To reduce gram calories per second to watts, multiply by 4.18617. To reduce a drop of temperature in Fahrenheit degrees to the drop in centigrade degrees, multiply by 5/9. For further reduction factors see the writer's paper above cited, and his Conversion Tables.

The original data for some of the figures in the accompanying table gave the conductivities relatively to silver, water or air, hence required the absolute values of these three materials to reduce them to thermal ohms. For this purpose the conductivity of silver in gram calories per second, centimeter cube units, at ordinary temperatures was taken at 1.0; that for water at room temperatures as 0.0013; and that for air at 0°C. as 0.0000553 (Winkelmann); those for silver and air being the reduction figures recommended and used in the Landolt & Boernstein tables, and that for water being approximately the one there recommended.

Figures concerning such mobile bodies as water and air are reduced here from those given but without any endorsement of the rationality of such data, because with such mobile fluids it would seem that some transfer of heat by convection (bodily movement of the heated material) must have taken place when the figures were determined; and in gases there must have been some radiation also. Even if these other ways of the transmission of heat had been eliminated during the test, they would be present in the practical apparatus to which these figures are applied. Moreover to the writer's knowledge there may be the equivalent of a relatively very high resistance at the surface of contact of a solid and a gas or liquid, hence unless this was eliminated when the determinations were made the results obtained may be far wrong. Hence, in the writer's opinion, fig-

*"Thermal Resistance and Conductance; the Thermal Ohm and Thermal Mho," this journal, Vol. 9, January, 1911, p. 13.

ures for such mobile materials are at best only very crude, if indeed they have much value at all.

A number of values have been included here merely for the sake of comparisons, and because published values of thermal conductivities are sometimes stated relatively to some other material, and if the absolute value of any one of them is known they can all readily be reduced to their absolute values. The writer has constructed furnaces which are virtually cold on the outside, hence low-temperature insulating materials then become available; but in such cases the designer should be cautioned against the dangers of over-insulation, when the material cannot stand a possible increase of temperature.

Another table giving figures for the flow of heat from surfaces will be published in a subsequent article.

The sources of the original data from which these values were deduced are indicated by the letters in the last column, which refer to the notes following the table.

Although the data in this table are arranged to be in the best from for calculations of electric heat, they can also readily be used for steam heat, refrigeration, etc., by first completing the single calculation in terms of the watts flowing through the insulation and then converting those watts into horse-powers, b. t. u. per hour, calories per hour or second, etc., as desired, by means of a single multiplication. Watts × 0.00134111 = horse-powers. Watts × 0.0568776 = b. t. u. per minute. Watts × 0.0143329 = kg-calories per minute.

TABLE OF THERMAL RESISTIVITIES.

APPROXIMATELY IN ORDER OF RESISTIVITY.

Temperatures in Centigrade Degrees.

	Thermal Ohms. Inch Cube.	Centimeter Cube.	Reference.
Silver, 0°-100°	0.094	0.24	LB
Copper, (electrode mean), 100°-197°	0.090	0.23	H
Copper, (electrode mean), 100°-837°	0.11	0.27	H
Copper, 0°-100°, about	0.11	0.27	LB
Copper	0.13	0.32	LB
Copper, cast	0.12	0.29	CJ
Copper, rolled	0.11	0.28	CJ
Copper, rolled	0.13	0.32	WF
Aluminum, 0°-100	0.27	0.69	LB
Graphite, Acheson, (electrode mean), 100°-390°	0.28	0.71	H
Graphite, Acheson, (electrode mean), 100°-914°	0.32	0.82	H
Brass, 0°-100°	0.36	0.92	LB
Iron, (electrode mean), 100°-398°	0.43	1.1	H
Iron, (electrode mean), 100°-1245°	0.49	1.3	H
Iron, wrought	0.22	0.55	CJ
Iron, wrought, 0°	0.46	1.2	LB
Iron, wrought, 275°	0.76	1.9	LB
Iron, wrought	0.79	2.0	WF
Iron, cast	0.26	0.66	CJ
Iron, cast, 30°	0.63	1.6	LB
Steel	0.24	0.60	CJ
Steel	0.81	2.1	WF
Steel, various	0.81	2.1	LB
Steel, 10% manganese	3.0	7.7	LB
Platinum	0.25	0.63	CJ
Platinum, 18°-100°	0.55	1.4	LB
Platinum	1.1	2.9	WF
Carbon, (electrode mean), 100°-942°	0.72	1.9	H
Carbon, (electrode mean), 100°-360°	1.05	2.7	H
Lead	0.33	0.83	CJ
Lead	1.10	2.8	WP
Lead, 0°-100°	1.2	3.0	LB
Plumbago brick, about 1000°	3.8	9.6	WQ
Carborundum brick, about 1000°	4.1	10.3	WQ
Mercury, 0°-50°	5.5	14.1	LB
Quartz, 0°	5.9	15.	LB
Graphite, (probably plumbago), 7°	8.0	21.	LB
Retort carbon, 0°	9.1	23.	LB
Magnesia brick, about 1000°	13.	34.	WQ
Stone, calcareous, fine	16.	42.	P
Chromite brick, about 1000°	16.	42.	WQ
Ice	16.	42.	LB
Boiler scale, 60°	17.	44.	LB
Marble, fine grained, gray	9.8	25.	P
Marble, coarse grained, white	12.	31.	P
Marble, 30°	19.	48.	LB
Stone, calcareous, ordinary	20.	51.	P
Fire brick (probably room temp.)	21.	53.	D
Fire brick, about 1000°	22.	57.	WQ
Fire brick, mean for 500°-1300°	23.	57.	Z
Fire brick, mean for 0°-1300°	30.	77.	Z
Fire brick, about 400°-800°	44.	112.	CE
Fire brick, mean for 0°-500°	67.	171.	Z
Checker brick, about 1000°	24.	61.	WQ
Gas retort brick, about 1000°	25.	63.	WQ
Slate, 94°	26.	67.	LB
Building brick, about 1000°	29.	72.	WQ
Glass pot, about 1000°	35.	89.	WQ
Porcelain, 95°	38.	96.	LB
Feuerstein	39.	99.	LB
Terra cotta, about 1000°	41.	104.	WQ
Chalk (solid?)	43.	109.	LB
Cement, Portland, neat, 35°	44.	110.	N
Cement, Portland, 90°	132.	336.	LB
Lava	47.	120.	LB
Silica brick, about 1000°	47.	120.	WQ
Kieselguhr brick, about 1000°	52.	133.	WQ
Baked clay, brickwork	57.	140.	P
Building brick wall, average, 8" to 40" walls	62.	160.	W
Water, room temperature	72.	180.	LB
Glass, 28°	87.	220.	LB
Plumbago, 20°-155°, 26.1% solid matter	96.	240.	O
Fine sand, 20°-155°, 51.4 solid matter	109.	276.	O
Coarse sand, 20°-155°, 52.9% solid matter	110.	280.	O
Cork (solid?)	131.	333.	LB
Plaster of Paris, 0°	105.	266.	LB
Plaster of Paris, 20°-155°, 36.8% solid matter	221.	562.	O
Slag concrete, 34.3 lbs. per cu. ft., 50°, 1 pt. by weight slag and 0.61 cement	178.	453.	N
Pumice stone, 18.2 lbs. per cu. ft., 50°	169.	430.	N
Pumice stone	187.	477.	LB
Pumice stone, 20°-155°,34.2% solid matter	219.	558.	O
Brick dust, sifted	204.	518.	P
Asbestos, 20°-155°, 8.1 solid matter	139.	353.	O
Asbestos, 36 lbs. per cu. ft., 600°	166.	422.	N
Asbestos, 36 lbs. per cu. ft., 50°	221.	562.	N
Asbestos, with air cells	416.	1060.	S
Card board, below 0°	239.	606.	LB
Ebonite, 48°	251.	637.	LB
Petroleum, 13°	265.	672.	LB
Wood pine, parallel to fiber	313.	796.	LB
Many liquid compounds (hydrocarbons, oils, etc.), about	313.	796.	LB
Anthracite coal	317.	803.	LB
Chalk, 20°-155°, 25.3% solid matter	332.	844.	O
Very porous slag, 22.5 lbs. per cu. ft., 50°	356.	905.	N
Zinc, white, 20°-155°, 8.8% solid matter	398.	1010.	O
Fire felt, 21°-175°	418.	1060.	B
Gas works breeze	440.	1120.	C
Infusorial earth (fossil meal), 21°-175°	415.	1050.	B
Infusorial earth 20°-155°, 11.2% solid matter	435.	1110	O
Infusorial earth, 20°-155°, 6% solid matter	472.	1200.	O
Infusorial earth, burnt, 12.5 lbs. per cu. ft., 450°	263.	675.	N
Infusorial earth, burnt, 12.5 lbs. per cu. ft., 50°	476.	1220.	N
Infusorial earth, loose, 21.8 lbs. per cu. ft., 350°	427.	1090.	N
Infusorial earth, loose, 21.8 lbs. per cu. ft., 50°	562.	1430.	N
Infusorial earth	745.	1890.	C
Magnesia, carbonate, 85%, 20°-188°, average	537.	1370.	S
Magnesia, calcined, 20°-155°, 28.5% solid matter	160.	407.	O
Magnesia, calcined, 20°-155°, 4.9% solid matter	544.	1380.	O
Magnesia, calcined, 20°-155°, 2.3% solid matter	584.	1410.	O
Magnesia, calcined, 21°-175°	572.	1450.	B
Charcoal, pine, 20°-155°, 11.9% solid matter	494.	1260.	O
Charcoal, from leaves, 11.9 lbs. per cu. ft., 100°	537.	1370.	N
Charcoal, from leaves, 11.9 lbs. per cu. ft., 50°	603.	1530.	N
Charcoal	723.	1840.	C
Feathers, 20°-155°, 2% solid matter	577.	1470.	O
Sawdust, 13.4 lbs. per cu. ft., 50°	614.	1560.	N
Sawdust	620.	1570.	C
Sawdust, compressed	765.	1950.	LB
Cork, granulated and compressed, 20°-188°	467.	1190.	S
Cork, ground, 10 lbs. per cu. ft., 200°	614.	1560.	N
Cork, ground, 10 lbs. per cu. ft., 50°	797.	2030.	N
Air, 20°-155°	143.	364.	O
Air, 0°	1701.	4320.	LB
Cotton wool, 20°-155°, 1% solid matter	596.	1520.	O
Cotton wool, 20°-155°, 2% solid matter	659.	1670.	O
Cotton wool, 5.05 lbs. per cu. ft., 100°	572.	1460.	N
Cotton wool, 5.05 lbs. per cu. ft, 50°	627.	1600.	N
Cotton wool	830.	2110.	C
Cotton wool, loose	2170.	5500.	LB
Cotton wool, compressed	2810.	7129.	LB
Hair felt, 20°-155°, 9.2% solid matter	633.	1610.	O
Hair felt, 21°-175°	790.	2010.	B
Hair felt	865.	2200.	C
Hair felt, below 0	1080.	2740.	LB
Lampblack, 20°-155°, 5.6% solid matter	697.	1770.	O
Fine quartz sand	718.	1820.	LB
Silk, 6.3 lbs. per cu. ft., 100°	662.	1690.	N
Silk, 6.3 lbs. per cu. ft., 50°	760.	1920.	N
Wool, sheep's, 20°-155°, 4.1% solid matter	616.	1570.	O
Wool, sheep's, 8.5 lbs. per cu. ft., 50°	676.	1720.	N
Wool, sheep's, 8.5 lbs. per cu. ft., 100°	745.	1890.	C
Wool, sheep's	803.	2050.	N
Mineral wool, 21°-175°	737.	1870.	B
Mineral wool, 0°-18°	1010.	2570.	C
Hard rubber	1060.	2680.	LB
Wood, pine, radially	1070.	2720.	LB
Loose fibrous materials, 9°	1540.	3920.	LB
Flannel	2650.	6720.	LB

B—George M. Brill. *Trans.* Am. Soc. Mech. Eng., XVI, p. 827. Coverings on 8-in. steam pipes.

C—J. J. Coleman. *Engineering*, Sept. 5, 1884, p. 237. Ice melted in cube surrounded with the materials. Temperatures 0-18° and 0-38° C. The values were given relatively to each other; to reduce them to absolute measure it is here assumed that the value for sawdust is 620, thermal ohm, inch cube units.

CE—Cement & Ely.

CJ—Calvert & Johnson. Relative values based on silver. Reduced here or the basis that the conductivity of silver is 1.0 in gram calories per second, centigrade, centimeter cube, units.

D—Depretz. Hood "Warming and Ventilating Buildings," p. 249. Given relatively to marble, here assumed to be 10 thermal ohms, inch cube, units.

H—Carl Hering. "The Proportions of Electrodes for Furnaces." (Table.) Paper read before the Am. Inst. Elec. Eng., March 31, 1910. Mean values when materials are used as furnace electrodes.

LB—Landolt & Boernstein tables. The values here chosen are mostly approximate means of the generally numerous and sometimes greatly differing values given by different observers. For the individual values and for the authorities, see those tables. They also include values for very many other materials.

N—Wilhelm Nusselt. *Zeit. Ver. Deut. Eng.*, June, 1908, p. 906, table p. 1006. Materials were placed between two concentric metallic spheres or cubes. Heat generated electrically in interior. Temperature measured with thermo-couples at numerous depths in the material after several days' heating. As here given they represent the resistivities at the temperatures stated, not the means over a range. Probably the best and most reliable determinations published. His conductivities are here assumed to be in terms of kilogram-calories per hour, centigrade, meter cube, units; although not so stated directly in the original, it is undoubtedly

what is meant. An abstract appeared in the *Eng. Digest*, August, 1908, p. 168, in which the units are reduced to thermal units, feet, inches and Fahrenheit degrees; the formula there given omits to say that it is necessary to multiply by the temperature also.

O—Prof. Ordway. *Trans.* Am. Soc. Mech. Eng. Vol. VI, 1884-5, p. 168. Tested in plates 1 in. thick between two flat iron surfaces, one of them heated by steam, the heat emitted by the other being measured calorimetrically. Extended, carefully made researches; presumably very good values. There is an error in the heading in table VII, square inch should read square meter, as in the others.

P—Peclet. Box "Practical Treatise on Heat." Presumably ordinary weather temperatures.

S—H. G. Stott. Power 1902. Pipe coverings. 200 ft. of 2-in. pipe heated electrically to constant temperature. Coverings were somewhat over 1 in. thick; they are here reduced to 1 in. Heat transmitted to air, hence these resistances include that at the surface.

W—Wolff. *Jour. Frank. Inst.*, 1893. The transmission of heat from the interior to the exterior of buildings through the walls; hence ordinary weather temperatures. Prescribed by law by German Government for heating plants. Said to agree well with good American practice. The value here given is an average of all the individual ones, omitting the first one, which differed greatly from all the others.

WF—Wiedemann & Franz; relative values based on silver. Reduced here on the basis that the conductivity of silver is 1.0 in gram calories per second, centigrade, centimeter cube, units.

WQ—Wologdine, Queneau. This journal, September, 1909, p. 383. The temperatures were about 1000° C.; the materials were those of commerce and do not refer to extra pure nor to inferior grades. The present writer is of the opinion, based on the method used in the tests, that these values are probably too low.

Z—Source lost, but probably fairly good values. See this journal, February, 1909, p. 72.

Temperature Coefficient.—Very little data seems to exist on the changes of the thermal resistivity with the temperature. It is generally recognized, however, that thermal insulators decrease in resistivity quite appreciably with increase of temperature. Nusselt, in the researches above cited, paid especial attention to this and finds from his measurements that as a general rule the thermal conductivity of such thermal insulating materials increases at the rate of 1/273 part for every degree centigrade above 0°C. This is of interest also because if this rate continues below 0°C. it means that the curve goes to the absolute zero, which would mean that such materials become absolute thermal insulators at absolute zero (when heat no longer exists.) On the other hand electrical conductors are believed to conduct without resistance at absolute zero.

Iron and Steel Institute.

In our last issue an account was given of the October meeting of the Iron and Steel Institute in London with abstracts of part of the papers. We herewith give abstracts of those papers which had to be omitted in our last issue on account of limitation of space.

Oxy-acetylene Welding.

A paper by Mr. F. Carnevali is entitled "Autogenous Cutting and Welding of Metals." He deals solely with the oxy-acetylene process. The advantages of this process are pointed out and the most important conditions for securing good results are summed up as follows:

1. The use of the purest acetylene obtainable.

2. The use of a blow-pipe so designed as to insure careful adjustment in the proportions of the mixed gases, and to secure their exit at a velocity capable of maintaining the metal sufficiently fluid without the melting flame being too rigid.

3. The provision of an absolutely neutral zone in the melting flame.

4. The use of a pure conducting material of well-defined composition, similar to that of the metal to be melted, in the form of small rods, of such a size as to be rapidly melted without hindering the simultaneous heating of the adjacent parts.

5. Care should be taken that the edges should be free from impurities, and that, whenever possible, they should be arranged at an angle to each other, so as to allow the neutral zone of the flame to penetrate well into the inside, and so completely melt the two surfaces before the extraneous metal is introduced.

6. The use of deoxidizing and fluxing powders for the purpose of eliminating the oxides present, together with the incidental impurities. In view of the unavoidable oxidation of the metal subjected to the melting process, it is necessary to bear in mind the relation between the temperature of the melting points of the oxides and that of the metal itself.

7. Rapidity in melting, in order to avoid excessive heating, which will not only alter and deteriorate the original structure of the metal, but would even favor the occlusion of gas—particularly hydrogen—and so occasion the formation of blow-holes in the melted zone.

In addition to these conditions, care should be taken to see that no sudden cooling should take place owing to any sudden projection of the flame, while, on the other hand, the conditions must, of course, be modified with regard to the conductivity and specific expansion of material as well as with regard to the thickness, size and shape of the pieces operated upon.

In the author's investigations the precautions sketched above were observed and his investigation extended chiefly to the changes in chemical composition and mechanical properties caused by the welding process and to methods of avoiding such changes.

The first two parts of the paper deal with his experiments on the oxy-acetylene method of welding of mild steel and of steels containing medium and high percentages of carbon. While the mild steels and the higher carbon steels are discussed under separate chapters, the results for both classes of steels are summed up together as follows:

1. The violent heating and the sudden melting undergone by the metal alter profoundly its quality. This alteration becomes specially manifest by an internal strain of the metal, meaning by this word an injurious modification of the metal's mechanical properties, caused by the thermal and mechanical phenomena, owing to the high temperature of the process, as the stress and hardness tests show, although uninfluenced by modifications in the chemical composition of the steel. The absence of deformation in the single structural constituents in stress tests is a further confirmation.

2. The alteration in the mechanical qualities (tenacity and elasticity) cannot be remedied by simple hammering on the molten zone—which often proves to be most injurious to high-carbon steels—but must be coupled by proper thermal treatment, such as annealing, which is so much more efficient when prolonged over a sufficient time (30 or 40 minutes), so that the interior stress can be nearly wholly eliminated. Further, the other thermal processes (quenching, etc.) have the greatest influence on high-carbon steels.

3. A remarkable influence on the mechanical properties (more especially of the later steels) is to be attributed to cooling processes, the slowest cooling being the best and most effective.

4. Apart from the alteration in mechanical properties (the principal of which is the change in the elongation), very important modifications take place in the chemical constitution, especially in high-carbon steels. These changes consist chiefly in the diminution in the percentage of carbon, silicon, and manganese. Such variations, which are proportional to the percentages originally present in the steel, are, it is very easy to understand, caused by the strong oxidizing influence to which the metal is subject during the melting, and become modified by the different methods employed in this process. It is easy to remedy this defect by using as a cementing material a steel of analogous composition to the pieces intended for welding, but containing higher percentages of the more easily oxidized elements.

5. The mechanical properties are, owing to the short duration of the process, only slightly altered when, instead of the fusion process, a simple heating with the flame is substituted, as is easy to foresee. As the metal during the melting is subjected to a strong internal stress, with every increase in the depth of the fused portion, the efficiency of the remedial heat treatment, undergone after the melting, rapidly decreases.

The author then investigated the oxy-acetylene welding method of **pig iron**, the experiments being limited to grey varieties.

The results for pig iron are summed up as follows:

1. The rapid melting and cooling of the pig iron subjected to the oxy-acetylene process cause, as might have been foreseen, a fine compact structure in the zone of fusion.

2. Owing to the high temperature and to the energetic conditions of heating which take place during the process of welding, a composition of the pig iron (and therefore, of its internal structure as well) undergoes considerable modification, chiefly owing to the loss in carbon and silicon (both of which strongly influence the structure of the metal). This loss, of course, varies with the different melting processes employed. An easy method for partly remedying this loss is to employ an appropriate cementing material which will supply the elements eliminated by oxidation.

3. The mechanical properties of the metal are not greatly influenced by oxy-acetylene welding, when properly carried out. The internal stress suffered by the metal confers great hardness on the zone of fusion, as shown by the Brinell tests. The brittleness tests, although they reveal the good quality and homogeneity of the metal, do not appear to be affected by the slight variations due to the melting of the metal.

4. The heat treatment and the cooling conditions, though they exert considerable influence on pig iron subjected to the process, do not appreciably modify the mechanical properties, and are of interest only when it becomes necessary to take into consideration the dilatation of the metal in relation to its low thermo-conductivity in the welding of special engine parts in which the physical properties are of great importance.

Phosphates in Basic Slag.

A paper by Mr. VICTOR ADOLPH KROLL, of Luxemburg, gives a detailed account of the author's extended "Researches on the nature of the phosphates contained in basic slag derived from the Thomas-Gilchrist (basic Bessemer) dephosphorization process." It is a very long paper accompanied by analytical tables and some fine microphotographs.

The annual production of basic slag is about 2,600,000 tons. Th slag contains nearly one-fifth of its weight of phosphoric acid, combined in a peculiar form the exact nature of which has hitherto defied the closest investigations.

After a very complete and careful history of former investigations on this subject, the author gives an account of his own researches. He distinguishes the following phosphates in basic slag.

First, "carnotite," discovered by Carnot, the typical silico-phosphate, in form of flat prisms, probably belonging to the monoclinic system, or in form of prismatic crystals as slender needles; the formula of carnotite is

$$(3\ CaO,\ P_2O_5;\ 2\ CaO,\ SiO_2)_3$$

Second, "hilgenstockite," the tetraphosphate of Hilgenstock, in the form of yellowish flattened prisms, with the formula

$$(3\ CaO,\ P_2O_5;\ CaO)_3$$

Third, "steadite," the silico-phosphate of Stead and Ridsdale with the formula

$$3\ [(CaO)_3,\ P_2O_5]:\ 2\ (CaO),\ 1\ (CaO)_2,\ SiO_2$$

This salt is therefore neither tribasic nor tetrabasic, but intermediate between the basicity of the two, and shows by its constitution that in all these slag phosphates it is not the basicity which counts, but the sum total of the foreign salts and bases linked up with the orthophosphate molecule.

"The silico-phosphate, which only contains tribasic acid, is perfectly equivalent to the tetraphosphate, because it is not the addition of a base, and a base alone, which confers a special constitution on a phosphate, but it is to a much greater extent the addition of a neutral silicate which produces the new and sufficiently stable molecule while preserving absolutely the same nature as the basic salt in which the silicate is found badly enough replaced by lime."

The theoretical part played by the tetraphosphate vanishes before that played by the neutral silico-phosphate, the apparent part played by the higher basicity being nothing more than a mere error of observation.

This composition of steadite assures to it considerably more influence in the liquid slag than its predecessors possessed. As a matter of fact, the slag in practice seldom reaches the percentage of silica and of lime corresponding to the ratio $5CaO, P_2O_5; SiO_2$, which is peculiar to carnotite, nor yet to so small a percentage of silica as to yield simultaneously hilgenstockite in the solution. Artificial enrichment accompanied by synthetical chemical reactions is therefore necessary in order to form these two constituents, and it is the selective crystallographic action occasioned by slow cooling which causes this phenomenon.

On the other hand, the composition of steadite more closely approaches that of the ratios found in the slag, and it is formed preferentially when the percentage of the latter in silica is not sufficient to give rise preferentially to more acid compounds, nor the cooling sufficiently slow to afford the crystallization a choice of selection as to the form it will take on assuming the solid state.

The main object of the author's researches, however, was to find a fourth phosphate in basic slag, intermediate between steadite and carnotite, the presence of which was sought for on the grounds of theoretical deductions. It is called "thomasite" and was found in all the slags collected from different districts.

"Thomasite" is nearly always found in the form of homogeneous cryptocrystalline masses, which may occasion difficulties in its identification. The reason is that it forms the mother-mass of the slag itself, and all the substances previously described are nearly always the products of selective crystalline decomposition, which occurs during slow cooling in the heart of an originally homogeneous mass of thomasite.

As slow cooling is just the condition necessary for obtaining fine crystals, a well-crystallized mother-mass is less frequently encountered than the products of its decomposition, which are always sought toward the center of the blocks of slag, whereas the main phosphoric mass is altogether cryptocrystalline, and is passed over unnoticed by those who are seeking to find crystals.

It is indeed its character of a mother-mass which has led to its escaping the notice of those who are in search of crystals. On a closer search, however, it can be found in all basic slags no matter what their source, and it will then be seen that it forms a very considerable cluster, out of which appear, in favorable localities, little crystals of greenish-blue color.

Thomasite is a silico-phosphate and a double salt of lime and of ferrous oxide. The formula given by the author for thomasite is

$$6\ CaO,\ P_2O_5;\ 2\ FeO,\ SiO_2$$

The phosphate in this is hexabasic, and therefore intermediate between the tetraphosphate and the octobasic salt. The reasons for speaking of hexaphosphates are discussed at considerable length.

Italy.

In view of the original intention of the Institute to hold the meeting at Turin, Italy, and to combine with it visits to the principal centers of the iron and steel industry of Italy, a large number of the papers prepared for the meeting deal with the present status of the Italian iron and steel industry.

A paper by Mr. CATANI, giving an interesting sketch of the present developments in the electrometallurgy of iron and steel in Italy, is published in full elsewhere in this issue.

A paper by D. DOMPE, of Milan, and Mr. F. S. PUCCI, of Milan, deals with "the present state of the metallurgical industry of Italy." The first part, by Mr. Dompe, gives statistical data on the production of pig iron, wrought iron, and steel, and brief descriptions of the principal works and their equipment. The second part, by Mr. Pucci, discusses the further treatment of the iron and steel and its manufacture into various products, a list of the different plants being given in a geographical arrangement according to provinces of Italy.

A paper by Mr. A. Ciampi, of Florence, deals with the iron ore deposits of Central Italy; a page by Prof. G. La Valle, of Rome, with the iron ore deposits of southern Italy and Sicily; a paper by Mr. L. Testa, of Igkesias, with the iron ore deposits of Sardinia, and a paper by Cavalier G. Calvi with the production of iron ores in the Brembana Valley.

Finally a paper by Dr. L. Leigh Fermor, of Calcutta, is to be mentioned which discusses the origin of the iron ores of Swedish Lapland.

A Continuous Carbon Dioxide and Temperature Recorder and Its Application to Combustion Efficiency *

By Ed. A. Uehling.

After the possibility to utilize the pressure of steam as a motive force had become fully recognized, the first problem was to construct a practical machine through which the heat energy stored in steam under pressure could be converted into mechanical motion. In the solution of this problem the plain slide-valve engine was evolved. With the advance of the mechanic arts this type of engine was improved in power and effectiveness.

Meanwhile improvements in the construction of the steam boiler had made higher steam pressures possible. To utilize the greater expansive force thus made available the automatic cut-off was invented, compounding was resorted to, and by adding the condenser the expansive force of steam was utilized to the practical limit. By virtue of these strides in the progress of steam engineering the consumption of steam per horse-power hour was reduced from an average of 35-40 lb. down to 12-15 lb. For a while it seemed much more could not be expected.

Then came the steam turbine. Its rapid development in the direction of high-power units caused a revolution in the application of steam as the motive force in generating electricity, and is making heavy inroads on the domain of the reciprocating steam engine in the propulsion of ships.

Compactness of construction, directness of action, avoidance of all reciprocating strains, absence of large rubbing surfaces, elimination of lubricating oil in contact with the steam, absence of cylinder condensation, are all points of superiority of the steam turbine as a prime mover.

In the direction of economy the turbine is not superior to the highest type of reciprocating engine; it excels, however, in utilizing the lower ranges of the expansive force in steam, and by combining the high-pressure reciprocating engine with the low-pressure turbine a steam consumption of 10 to 12 lb. per horse-power hour is being practically achieved.

Hand in hand with the progress in the engine room went of necessity improvements in the boiler room. In the direction of fuel economy we early find the feed-water heater and later the economizer. Every effort was directed toward increased capacity and higher steam pressure; this necessitated increased strength and resulted in the development of the water-tube boiler.

In the direction of mechanical equipment wonderful strides have been made in the past two decades. Mechanical stokers, forced-draft automatic coal and ash-handling machinery, etc. are now found in the boiler houses of all the larger up-to-date power plants. All these vast improvements in the development of the application of steam power have been accomplished by the mechanical engineer with the aid of the physicist and metallurgist, but without the direct assistance of the chemist.

This is deplorable and accounts for the fact that in utilizing the steam, which is a physical and mechanical problem, an economy of 60 to 70 per cent has been achieved, whereas the generation of the steam, which is a fundamental chemical problem, nothing worth mentioning has been gained in the past generation, 30 to 40 per cent of the heat energy stored in the coal still passing up the chimney, due mostly to lack of control of the process of combustion.

The possibilities in this direction have been shown by the brilliant investigations of Professor Bone; but while we may readily concede that surface combustion may ultimately prevail, we must not for that reason relax in our efforts to improve existing methods, and this can be accomplished only by the aid of the chemist.

I venture to say that in this country to-day, not one in a hundred of the managers, engineers or superintendents of steam-power plants knows the composition of the coal he is burning, not one in ten knows its contents of fixed carbon, volatile matter, ash, and moisture. Not in one plant out of a hundred are the products of combustion systematically sampled and analyzed. Yet in the composition of the flue gas lies the key to fuel economy.

Of the heat generated from the coal burned in the average steam plant less than 70 per cent, and in many cases not even 60 per cent is transmitted to the water in the boiler; the bulk of the balance is carried off by the product's combustion. To reduce this loss to a minimum we must know its cause, and this can be determined only from having a complete knowledge of the chemical composition as well as the physical character of the fuel used, and a continuous control over the process of combustion.

I lay special stress on the continuity of control over the process of combustion, because the conditions are continully changing, not only from day to day, and from hour to hour, but from minute to minute. Changes in atmospheric condition as to pressure, temperature, and humidity; changes in the physical structure of the coal as to preponderance of lumps or fines; changes in the condition of the fire, due to accumulation of ashes, formation of clinker, etc., and, last but not least, changes in the rate of combustion necessary because of variable steam consumption.

The best results are obtained by securing complete combustion with a minimum excess of air, and since there is true relation between the per cent of CO_2 in the flue gas to the excess of air supplied we have in this constituent an index to efficient firing. If, therefore, the per cent of CO_2 be continuously revealed to the fireman it will serve as a guide and enable him to control his fire so as to obtain the most economical results.

If we know the composition of the fuel the loss of sensible heat up the chimney can be readily calculated from the stack temperature and the per cent of CO_2 contained in the flue gas. By basing the calculation on a unit weight of the total carbon contained in the fuel the formula for computing the waste heat from combustion become very simple.

A number of instruments have been devised which automatically sample and analyze the flue gas and record the per cent of CO_2 it contains at short intervals. There is also one which does this continuously and at the same time continuously indicates the per cent of CO_2 at or near the boiler front where the fireman can see it without leaving his post, and I have been honored with the privilege to bring this instrument to your attention to-night. (See the article on p. 681 in this issue.)

Formulæ for Determining the Heat Carried Off by the Flue Gases.

NOTATION.

A = weight of air supplied in burning 1 lb. of carbon.
Ac = " " " required to burn 1 lb. of carbon.
Ah = " " " " " the available hydrogen per pound of carbon.
Ae = weight of air supplied in excess of that required to burn a weight of fuel containing 1 lb. of carbon.
An = weight of dry gas contained in the flue gas per lb. of carbon burned.
Av = weight of water vapor in atmosphere per lb. of air supplied.

* A paper presented at the joint meeting of the New York Sections of the American Chemical Society, the American Electrochemical Society, and the Society of Chemical Industry, on November 10.

Ov = weight of volatile oxygen contained in fuel per lb. of carbon.

Ht = weight of total hydrogen contained in dry fuel per lb. of carbon.

$Ha = Ht - \frac{Ov}{8}$ = weight of available hydrogen in fuel per lb. of carbon.

$Wh = 9/8\ Ov$ = water of hydration in fuel per lb. of carbon.

Wm = weight of water (moisture) in fuel per lb. of carbon.

P = per cent of CO_2 contained in flue gas.

Pc = " " CO " " " "

L = B.t.u. loss due to sensible heat in flue gas per lb. of carbon when burned with dry air.

Lc = B.t.u. loss due to fuel value of CO contained in flue gas per lb. of carbon burned.

Lv = B.t.u. loss due to moisture in air supplied per lb. of carbon burned.

Lh = B.t.u. loss due to water of hydration in fuel per lb. of carbon burned.

Lm = B.t.u. loss due to moisture in fuel per lb. of carbon burned.

T = temperature of flue gas on leaving the boiler.

t = temperature of air in boiler house.

$S = 0.23$ = specific heat of flue gas.

$Sw = 0.48$ = specific heat of water vapor.

Pure air contains	by weight	by volume
Oxygen	23	21
Nitrogen	77	79

One cu. ft. of air at 62 deg. F. weighs .0761 lb.

" " " " nitrogen at 62 deg. F. weighs .0738 lb.

Complete Combustion of Carbon with Theoretical Amount of Air.

One lb. of carbon requires for complete combustion $Ac = 11.6$ lb. of dry air, which measures 152 cu. ft. at 62 deg. F. The products of complete combustion of 1 lb. of carbon, with the theoretically required amount of air, will weigh $Ac + 1 = 12.6$ lb. Hence the loss of heat up the chimney will be:

$$L = 12.6\,(T - t)\,S \qquad (1a)$$

Substituting the value of S

$$L = 2.9\,(T - t) \qquad (1)$$

Complete Combustion of Carbon with an Excess of Air.

Air is composed of 21 per cent of O_2 and 79 per cent of N; hence, according to Avogadro's law, 21 per cent of CO_2 is the maximum obtainable.

When carbon is burned with an excess of air (Ae), which by volume equals $\frac{Ae}{0.0761}$ cu. ft., then

$$P = \frac{152 \times 21}{152 + \frac{Ae}{0.0761}}$$

from which we get $Ae = \frac{21 \times 11.6}{P} - 11.6$

The total weight of air $A = Ac + Ae$. Substituting the values of Ac and Ae we have

$$A = \frac{21 \times 11.6}{P}$$

$$L = (1 + A) \times (T - t)\,S \qquad (2a)$$

Substituting the values for A and S and reducing to the lowest terms we have

$$L = \left(0.23 + \frac{56}{P}\right) \times (T - t)$$

Calculation of Heat Lost in the Flue Gas When Hydrogen Forms an Appreciable Part of the Fuel.

In the chemical reaction which constitutes the combustion of hydrogen, 8 lb. of oxygen and 1 lb. of hydrogen unite to form 9 lb. of water vapor. This requires 34.8 lb. of air and results in 35.8 lb. of combustion product. In commercial flue-gas analyses no account is taken of the moisture, hence only the nitrogen derived from the combustion of hydrogen effects the analytical result. The burning of 1 lb. of hydrogen with the theoretically required volume of air therefore results in 26.8 lb. of N_2 by analyses.

Hence if fuel consisting of carbon and hydrogen is burned with the theoretically required air,

we have $P = \frac{11.6 \times 21}{11.6 + 26.8\,Ha}$

if excess air is used

we have $P = \frac{11.6 \times 21}{11.6 + 26.8\,Ha + Ac}$

from which we get

$$Ac = \frac{11.6 \times 21}{P} - 11.6 - 26.8\,Ha$$

$$An = Ac + 0.77\,Ah + Ac$$

Substituting and reducing

we have $An = \frac{11.6 \times 21}{P}$

$$L = (An + 1) \times (T - t)\,S + 9\,Ha\,(T - t)\,Sw \qquad (3a)$$

Substituting the values of An, S and Sw and reducing

we have $L = \left(\frac{56}{P} + 0.23 + 4.3\,Ha\right) \times T - t \qquad (3)$

Loss Due CO in Products of Combustion.

When the per cent of CO (Pc) is known, the heat loss due to its presence (Lc) is easily determined. The weight of carbon in a molecule of CO_2 is the same as in a molecule of CO_2 and it follows from Avogadro's law that the weight of carbon burned to CO is to that burned to CO_2 as the per cent of CO is to the per cent of CO_2 contained in the combustion products.

Thus, letter X = the weight of carbon burned to CO,

we have $\frac{X}{1 - X} = \frac{Pc}{P}$ from which we get $X = \frac{Pc}{P + Pc}$ = carbon burned to CO per unit of carbon contained in the fuel. Since CO burned to CO_2 per unit of carbon develops 10,150 B. t. u.

we have $Lc = 10{,}150 \times \frac{Pc}{P + Pc} \qquad (4)$

Loss Due to Water Vapor in the Air.

Atmospheric moisture exerts a chilling effect on the fire because of its high specific heat and because at least part of it will be decomposed. But since under normal conditions the hydrogen which was set free by decomposition will reunite with the oxygen from the air in the combustion chamber and will evolve as much heat there as was absorbed in the fuel bed in setting it free, there results no heat loss due to this double reaction. It will be necessary, therefore, to consider only the sensible heat carried off by moisture contained in the air, hence

$$Lv = Av \times (An + 0.23\,Ah)\,Sw \times (T - t) \qquad (5a)$$

Substituting and reducing we have

$$Lv = Av\ \left(\frac{117}{P} + 0.88\,Ha\right) \times (T - t) \qquad (5)$$

All natural solid fuels contain water in two forms; moisture (Wm), i. e. water which can be driven off by drying at about 212 deg.; and water of hydration (Wh), which passes off at much higher temperature with the volatile combustible matter, the estimation of which is ordinarily but wrongly included in the volatile combustable matter.

Loss Due to Water Hydration.

If we know the weight of oxygen contained in a given fuel per lb. of carbon get the water of hydration.

$$W'h = 9/8\ Ov$$

Hence $Lh = 1.125\ Ov \times (T - t)\ Sw$ (6a)

Substituting the value of Sw

we have $Lh = 0.54\ Ov \times (T - t)$ (6)

Loss Due to Moisture.

Moisture in fuel ($W'm$) exists as water, which must first be raised to 212 deg., then evaporated, the vapor passing off at the temperature of the flue gas.

Hence $Lm = W'm\ (212 - t) + 966 + (T - 212)\ Sw$ (7a)

Substituting the value of Sw and reducing

we get $Lm = W'm\ [(0.48\ T) + (1076 - t)]$ (7)

Recapitulation.

(1) Heat in B.t.u. carried off by the products of combustion per lb. of carbon burned when there is no excess of air.

$$L = 2.9\ (T - t) \quad (1)$$

(2) Heat in B.t.u. carried off by the products of combustion per lb. of carbon burned with an excess of dry air.

$$L = \left(\frac{56}{P} + 0.23\right) \times (T - t) \quad (2)$$

(3) Heat in B.t.u. carried off by the products of combustion per lb. of carbon when fuel containing hydrogen is burned with an excess of dry air.

$$L'' = \left(\frac{56}{P} + 0.23 + 4.3\ Ha\right) \times (T - t) \quad (3)$$

(4) Heat value in B.t.u. carried off by the products of combustion containing CO per lb. of carbon burned.

$$Lc = 10150 \frac{Pc}{P + Pc} \quad (4)$$

(5) Heat in B.t.u. carried off by the products of combustion due to water vapor in the air supplied in burning a weight of fuel containing 1 lb. of carbon.

$$Lv = Av' \left(\frac{117}{P} + 0.88\ Ha\right) \times (T - t) \quad (5)$$

(6) Heat in B.t.u. carried off by the products of combustion per lb. of carbon due to water of hydration in the fuel.

$$Lh = 0.54\ Ov \times (T - t) \quad (6)$$

(7) Heat in B.t.u. carried off by the products of combustion due to moisture contained in the fuel, per lb. of carbon.

$$Lm = W'm\ (0.48\ T + 1076 - t) \quad (7)$$

These formulæ show clearly that the heat wasted up the chimney depends on two factors, the per cent of CO_2 and the temperature of the flue gas. The per cent of CO_2 in the flue gas is an index to the efficiency of the boiler as a heat generator and the temperature is an index to its efficiency as a heat absorber.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Turin Exhibition.

Your correspondent spent a week in Turin early in September, naturally giving much of the time to the International Exhibition there. There was very much to be seen that was praiseworthy. For one thing the popular sideshows which exasperate the serious visitor were at a minimum.

The general engineering exhibits were decidedly interesting. Of outstanding interest in the frontier between the spheres of engineering and chemistry was the excellent French exhibit of electric furnaces. The American exhibits failed to do justice to American potentialities, and no single section in the United States pavilion was really representative. Metallurgically the British display was worse, and the German display was quite indifferent.

In regard to pure chemistry a special feature of the whole exhibition was the British display, organized by the Board of Trade, of a completely equipped laboratory with apparatus and reagents for both commercial routine testing and for special research. Next to this was a first-class electro-physical laboratory organized by the National Physical Laboratory.

This is not the place to write of the display in such industrial arts as weaving, spinning ceramics or automobiles, where the dispaly was decidedly good. On the electrical side there was a very fine display of heavy generating plant, single-phase railway equipment and extra high-tension overhead material.

The British Industrial Outlook.

Since I wrote my letter at the end of August there has been a decided improvement in the outlook. The Royal Commission on the Railway Strike has reported fully and impartially; the extreme trades union element is, of course, dissatisfied, but the more responsible among their leaders are trying to smooth down the trouble. The South Wales coal area is unsettled, and the neighboring copper works have had trouble. There are, too, still threats of a big coal strike in the north.

Irresponsible socialist extremists are preaching a combined railway and coal strike, which the great mass of the general public, artizan as well as middle-class, will not stand.

The Government efforts to rush through their industrial insurance bill making provision for sickness, and also in certain trades for unemployment, are disturbing to some of their usual supporters. The reason at the root of their action is probably not so much to clear the way for other legislation next year as to discount the force of the Socialist agitation. Disturbed supporters are, however, fearful of the fruits of hurried legislation.

Taking things all round the outlook is better than a month ago, trade conditions are good, and engineering works are now being hampered by that inevitable concomitant of a trade boom, the difficulty of getting the raw materials.

The general confidence is strong, although the European situation still has potentialities of international trouble. Morocco is almost settled, but the Turco-Italian trouble may spread to the Ægean Sea or the Balkans. Assuming the best rather than the worst, economists can then give their attention, and educate an intelligent public attention, to the inter-action of scarcity in food stuffs, heavy gold production and the decreased purchasing power of gold.

Engineering Imports and Exports.

The Board of Trade returns for the nine months ended Sept. 30 give the value of imports of iron and steel including manufactures, as £8,201,274, with exports £31,851,601; showing an increase in imports of £1,692,389, and a decrease in exports of £83,303 as compared with the corresponding period of last year. For other metals, including manufactures, the imports were £20,730,879 and exports £8,024,961, the respective increases being £3,136,716 and £398,697.

Imports of electrical goods reached £1,012,643, and exports £1,990,852—a decrease in imports of £142,936 and a decrease in exports of £1,043,873. Imports of machinery were £4,411,648 and exports £22,535,942; showing increases of £1,068,346 and £1,001,425 respectively. Imports of new ships amounted to £60,916, with exports £4,039,482—an increase in imports of £38,581, and a decline in exports of £3,535,579.

The effect of the recent labor troubles is shown by the fact that the returns for the month of September show considerable decreases in the exports of all these classes of goods except non-ferrous metals which increased by £10,810. The decreases are: Iron and steel, £301,337; electrical goods, £222,357; machinery, £350,799; and ships £186,080. Imports of iron and steel increased by £77,025; non-ferrous metals by £435,176, and machinery by £82,331; but imports of electrical goods were lower by £53,532, and the value of imported ships declined £482.

Market Prices in October, 1911.

Copper has been rather dull. Commencing at £55, it had dropped by the 10th to £54-5-0, and, after a short rise, to £56, again dropped and finished £55-5-0.

Tin has been, on the whole, going up. Commencing at £182.0-0 it rose in two jerks to £188-10-0 by the 10th, dropping again to £183-5-0 by the 12th. It then rose to £192 (20th) and closes at £186-15-0.

Lead has been steadily appreciating, with very slight dip on the 10th. Opened at £15-5-0 and closes at £15-15-0.

Hæmatite opened at 61/6, dropped sharply on the 11th, reaching 60/3 on the 12th, then rose to 61/-, and closes at 60/9.

Scotch Pig has been fairly steady, fluctuating only from 52/- to 52/4½. Closes at 52/3.

Cleveland has been moving slightly in sympathy with Scotch Pig. Opened at 46/-, closes at 46/2.

India-rubber has been rather on the decline, opening at 4/7, holding at 4/5½ in mid-month, dropping to 4/3 by the 25th, and closing at 4/3.

	£	s.	d.
Alum, lump, loose, per ton	5	12	6
Antimony, black sulphide powder, per ton	22	0	0
Borax, British Refined Crystal, per ton	16	0	0
Copper Ore, 10 to 25%, per unt.....9/4½ to		9	10½
Copper Sulphate, per ton	20	0	0
Carbolic Acid, liquid, 97/99%, per gal		1	7
Camphor, 1-oz. tablets		1	9½
Caustic Soda, Ash 48%, ordinary, per ton	5	10	0
Hydrochloric Acid, per cwt		5	0
Mica, small slabs, original cases, per lb.....6d. to		2	0
Naphtha solvent, 90%, 160° C., per gal			8
Platinum, per oz., nominal	9	5	0
Petroleum, Russian spot			6
Quicksilver, per flask, Spanish	8	10	0
Sal Ammoniac, lump, first delivery U. K., per ton	42	0	0
Sulphate of Ammonia, f.o.b., Liverpool, per ton	14	5	0
Sulphur, recovered, per ton	5	0	0
Shellac, Standard T. N. Orange Spots, per cwt	4	4	0
Tin Ore, 70% per ton.....£117 to	119	0	0
Zinc, Vielle Montagne	31	2	6

The following differences will be noted:

Rise.	*Amount.* £	s.	d.
Lead	5	5	0
Copper Ore, per unit			3
Quicksilver, per flask		2	6
Sulphate of Ammonia, per ton		7	6
Tin Ore, per ton	5	0	0
Tin, per ton	7	5	0
Copper, Standard, per ton		5	0

Fall.	*Amount.* £	s.	d.
Zinc V. M., per ton	1	7	6
Hæmatite		1	0
Scotch Pig			3
Cleveland			4
Para Rubber			5

Recent Chemical and Metallurgical Patents

Iron and Steel.

Combined Open-Hearth and Electric Furnace for Steel Refining.—The interest taken by the United States Steel Corporation in electric steel refining is further evidenced by a patent of Mr. WM. R. WALKER and Mr. JAMES H. GRAY. Instead of refining steel first in an open-hearth and then transferring it in molten condition to an electric furnace for further treatment, they carry out the whole operation in one and the same furnace, which is first used as open-hearth and then as electric furnace, thus eliminating any transfer of molten metal. (A furnace of this type was in use a year or so ago in Homestead, but none of the results obtained have so far become available for publication.) The furnace has the form of an ordinary open-hearth furnace and is first used as such, silicon, carbon and phosphorus being eliminated by means of an oxidizing slag. This slag is then removed and a fresh basic slag is supplied, while simultaneously electrodes are introduced into the furnace from the top. The electric-furnace treatment now begins and in its non-oxidizing atmosphere the steel is desulphurized and deoxidized. During this second electrical stage of the process, the regenerative chambers are cut off from the main heating chamber. (1,000,217, Aug. 8, 1911.)

Titanium Treatment of Steel.—A patent of Mr. AUGUSTE J. ROSSI, of the Titanium Alloy Manufacturing Company, refers to the treatment of steel with ferro-titanium, only such a quantity of titanium being used as is required to combine with the undesired compounds and elements, and to secure a final product containing substantially no titanium. The object is to procure a close grained, dense structure, without blowholes or cavities. The method is applicable in the crucible, converter and open-hearth steel processes for castings, and in the production of ferro-alloys. (1,003,805, Sept. 19, 1911.)

Electrolytic Refining of Iron.—For the electrolytic production of pure iron, free from carbon, from impure iron, like cast or wrought iron scrap, cast iron turnings or borings, etc., Mr. ALEXANDER S. RAMAGE, of Buffalo, N. Y., uses the process sketched in Fig. 1. The electrolytic cell 1 is divided by a diaphragm 2 of asbestos cloth into anode and cathode compartments 3 and 4. The anodes 5 are of sheet lead, the cathodes 6 of iron. When a solution of ferrous sulphate is electrolyzed in both compartments, iron is deposited on the cathodes, while ferric sulphate and sulphuric acid are formed at the anode. From the anode compartment the acid liquor is passed into the scrubber 7, where it is subjected to the action of a small proportion of sulphur dioxide. (Only enough sulphur dioxide

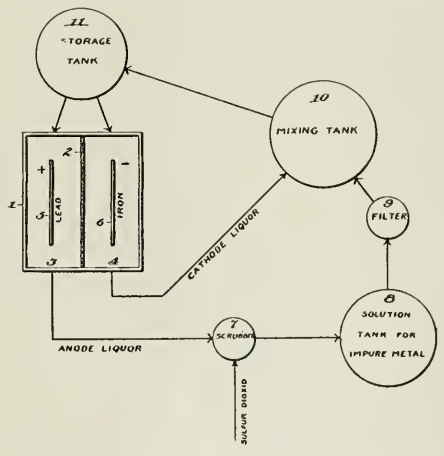


FIG. 1.—ELECTROLYTIC REFINING OF IRON

is used to form sufficient sulphuric acid to replace that lost by leakage, but not so much as to effect the reduction of the ferric sulphate to ferrous sulphate.) The acid liquor then passes to solution tank 8, where it is brought in contact with the impure iron to be refined (scrap, etc.), together with ferrous sulphide, which under the action of the acid liquor generates hydrogen sulphide. Under these conditions there occurs a complete pre-

ciptation of those metals whose sulphides are insoluble in dilute acids. To aid the solution of the iron, heat is employed, and the solution tank is also fitted with an insoluble anode buried in the scrap and with an iron cathode in the liquor above the scrap, so as to aid in the solution of the scrap iron and diffuse the hydrogen throughout the solution. The hydrogen and the sulphuretted hydrogen resulting from the solution of the ferrous sulphide effect the reduction of the ferric sulphate to ferrous sulphate. When the liquor has become practically neutral, it is forced through a filter press 9, flowing thence to a mixing tank 10, where it is mixed with the effluent liquor from the cathode compartments. The mixed liquor flows to a storage tank 11, whence it is distributed to the several anode and cathode compartments of the electrolytic cells, thus completing the cycle of operations. The preferred temperature during electrolysis is 145 deg. Fahr., a suitable current density is 10 amp. per square foot, the voltage approximating 2.5 volts per cell. With revolving cathodes current densities approximating 100 amp. per square foot may be used, with a correspondingly increased voltage. The chief feature of the process is that it avoids the solution of such metals as are liable to contaminate the electro-deposited iron, notably carbon and certain metals, such as arsenic and antimony, whose sulphides are insoluble in dilute acid solutions. (1,007,388, Oct. 31, 1911.)

Steel for Armorplates.—A low-carbon steel for armor plates and gun tubes is patented by Mr. Wm. Beardmore, of Glasgow, Scotland. In addition to the usual ferrite, it contains a percentage of carbon, not exceeding 0.25, and molybdenum and nickel in proportions sufficient to compensate for the deficiency in the strength and toughness of the steel due to the low percentage of carbon. The composition is 0.15 to 0.25 per cent C, 1 to 2 per cent Mo, 5 to 7 per cent Ni, further not over 3 per cent W, and not over 0.7 per cent Mn. This steel does not sufficiently harden by mere quenching when hot. It must be subjected to a process of cementation. By suitably regulating the degree and extent of differential hardening treatment to which the steel plates are subjected, it is possible to secure great hardness in some parts while retaining at other parts a ductile and tough condition. "In the case of armor plates the qualities possessed by the steel enable them after cementation to resist the penetration of modern capped projectiles and at the same time to resist the racking action of the projectiles, particularly in the case of plates of less thickness than the caliber of the guns used in attacking them. In the case of steel for gun tubes, an increased resistance to bursting or cracking of the tube by the shock of discharge is obtained. (1,007,055, Oct. 31, 1911.)

Treatment of Iron and Steel to Prevent Rusting.—Mr. Thomas Watts Coslett, of Birmingham, England, proposes to prevent the rusting of iron or steel by subjecting it to an acid solution of zinc phosphate. A concentrated solution of 6 oz. zinc, 1 pint phosphoric acid and 1 pint water is made up and applied directly to the iron or steel articles for a short time or is diluted with water before the iron or steel articles are immersed in it. For dilution 1 oz. of the concentrated solution is added to each gallon of water. For the purpose of renewing the dilute solution there is added to each gallon thereof ¼ oz. or less of the concentrated solution as and when required. Under certain circumstances it is of advantage to treat the iron with a solution containing both zinc phosphate and iron phosphate; in this case iron filings and granulated zinc are dissolved in phosphoric acid. "The formation of the deposit on the iron or steel surface under treatment may be accelerated by passing an electric current through the dilute solution, which may be either in a hot or a cold condition," with iron anodes. (1,007,160, Oct. 31, 1911.)

Gold and Silver.

Continuous Vacuum Filter.—A novel form of continuous filter for separating cyanide solution from ore pulp has been patented by Mr. Bruce W. Taylor, of New York. As shown in Fig. 2, which is a sectional elevation, the machine consists of a tank in which a chain of filtering pans is mounted, adapted to travel around sprocket wheels, one of which is mounted outside and slightly above the top of the tank, while the other is located in the lower part. The vacuum pans have the usual construction of wire cloth, cocoa matting and filter cloth, and each pan has suitable connections with vacuum and

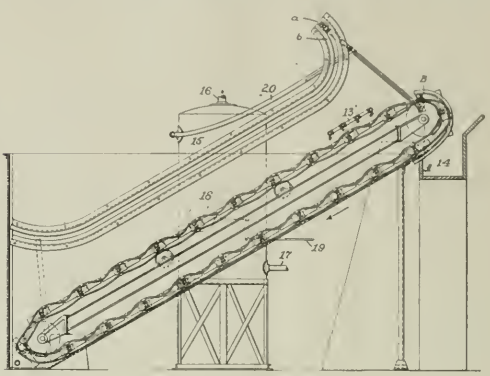


FIG. 2.—CONTINUOUS FILTER

air pressure. The point at which the pans are connected with the vacuum and where the air pressure closes is indicated at *A*, while the point at which vacuum is cut off and air pressure applied is shown at *B*. As the pans descend into the tank which is filled with the pulp to be filtered, they are connected with vacuum and consequently accumulate a cake of slime. As the pans rise out of the tank they first pass under a water spray shown at 13, where they receive a wash. Later the vacuum is broken at *B* and air pressure applied which forces the cake from the pan into a discharge launder 14.

The vacuum tank is indicated at 15, and has dry vacuum connection at 16 and wet vacuum connection at 17. The pump operating the latter is controlled by a float 18, 19, so that it will work fast or slow according to the amount of solution in the tank. There is a continuous vacuum on the hose 20,

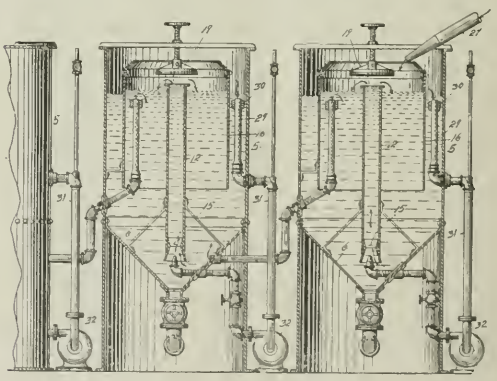


FIG. 3.—CONTINUOUS AGITATOR

and the vacuum on the several pans is controlled by suitable valves. The pipe connection at the end of the hose is provided with a trolley *a* and track *b*, and the pipe travels back and forth as the chain of filter pans revolves, the hose also following the line of travel. (1,000,856, Aug. 15, 1911.)

Pulp Agitator.—The improvement made by continuous agitation of ore pulp in cyanide solution has led to a number

of designs of agitators being patented. One of the latest is that of Mr. WILLIAM C. PATERSON, of Denver, Col. The agitator may be used either singly or in plurality. In the accompanying drawing, Fig. 3, the agitator is shown in plurality for continuous work.

The machine belongs to that class of agitators in which a centrifugal pump is the means of agitating the pulp, and in which means are provided in the tank for permitting only clear solution to go to the pump, thereby avoiding excessive wear due to the abrasive action of the mineral particles. In the drawing the number 5 designates a cylindrical tank having a cone bottom 6, with a central pipe 12, which is slightly reduced in diameter at 15. Surrounding the upper part of the central pipe is a circular partition 16, while immediately above the central pipe is a baffle plate 19, which is movable vertically. Pulp is fed to the system at 27, and the agitation and circulation are accomplished by centrifugal pumps 32. The pump draws practically clear solution from outside the circular partition, through the pipe 31, which terminates in a flexible portion 20, with a float 30.

Similar arrangement is provided on the opposite side of the tank, but within the circular partition, for the purpose of carrying a portion of the pulp from one tank to another. By this arrangement, agitation is accomplished by the circulation of clear solution through the central pipe, and the transfer of pulp through the system is made by continuously drawing off a uniform sample of the pulp as it falls from the top of the central tube. (1,000,689, Aug. 15, 1911.)

Improvement in Zinc Dust Precipitation.—Mr. CHARLES W. MERRILL, of Berkeley, Cal., who had used zinc dust precipitation of cyanide solutions for some years, has recently been granted letters patent for an improvement in the system. He claims that the ordinary zinc fume or dust is rendered less efficient as a precipitant on account of the slight coating caused by oxidation of the zinc. It is further claimed that this inefficiency is more noticeable in treating low grade or weak solutions. Mr. Merrill proposes to add to the usual precipitating system a grinder, such as a small tube or barrel mill, through which the entire mixture of gold-cyanide solution and zinc dust is to be passed. This apparatus will be placed somewhere between the tank containing unprecipitated solution and the filter press which is to separate the solid from the precipitated solution. The grinding operation will remove the deterrent coating from the zinc fume and increase its efficiency as a precipitant. (1,006,865-6, Oct. 24, 1911.)

Copper.

Copper Converter Tuyere.—Another patent has been added to the list of those issued to Mr. ELIAS A. C. SMITH, of Baltimore, Md., in connection with the development by him and Mr. Peirce of the basic copper converter. This latest patent refers to the construction of the converter tuyere, and the only claim reads as follows: "A blast tuyere, provided at its outer end with a Dyblie valve and its casing having a removable cap, the main bore of the casing being in alignment with and of the same diameter as the bore of the tuyere, and said cap having a perforation of but slightly less diameter than that of the tuyere; whereby, during the continuance of the blast, a punch bar may be employed, making a machine fit with said perforation and approximately but loosely fitting the bore of the tuyere, and whereby, on removing the cap during interruption of the blow, a second punch bar may be employed, still more neatly fitting the bore of the tuyeer." (1,007,149, Oct. 31, 1911.)

Treatment of Copper-Zinc Sulphides in a Converter.—With a view to improving the methods used for the treatment of copper and zinc sulphide ores, M. DÉSIRÉ KORDA, of Paris, has invented a procedure which he claims to be more economical than any yet devised. He provides an ordinary converter of the kind used for converting copper and uses a high proportion of coke in its lining. This lining may be formed of an intimate mixture of 60 parts quartz, 20 parts coke and 20 parts refractory clay. The lining around the tuyeres has no coke. After the converter is dried and heated in the usual manner and lined with incandescent coke, covering the tuyeres with combustible material. A few minutes' blowing brings the contents to a white heat, when the ore is then charged until the converter is full. It is preferable to charge the ore in the form of rock. In order to start the operation quickly, the cleanest sulphides are charged first, the less fusible ores following. Finally a charge of coke of nut size is added.

The blast is continued at a moderate pressure until a liquid bath forms in front of the tuyeres, after which the pressure may be increased. At the end of an hour the charge is melted to a homogeneous mass and the maximum blast is obtained. Zinc is reduced and discharged at the mouth of the converter, where it burns to oxide and is conducted to recovery chambers. Copper is converted to matte, and the operation is stopped when the latter approximates 60 per cent copper. Five per cent of pyrite is now added to the converter, having the effect of cleaning the slag of copper. The converter is then skimmed and the matte blown to metal.

The following figures are given by the inventor as showing an actual test of the process: Ore: Cu, 6.77 per cent; Zn, 16.31 per cent; Fe, 24.00 per cent; S, 23.00 per cent; SiO_2, 15.00 per cent. Matte: Cu, 67.22 per cent; Zn, 2.36 per cent. Slag: Cu, 0.54 per cent; Zn, 4.64 per cent. Oxide: Cu, 0.10 per cent; Zn 55.00 per cent. The net recovery is thus 93 per cent of the copper and 78 per cent of the zinc. Owing to slight waste in the recovery of the latter, however, this figure is reduced to 60-70 per cent of the zinc really recovered in the form of oxide. (1,003,051, Sept. 12, 1911.)

Lead and Zinc.

Refining Lead and Lead Alloys.—The removal of impurities from lead and the alloys of lead with antimony and tin is highly desirable from a commercial point of view. Chief among these impurities is arsenic, the presence of which renders lead unfit for certain commercial uses. Mr. LELAND E. WEMPLE, of St. Louis, Mo., proposes to accomplish the removal of arsenic and other impurities by treating the molten lead or lead alloy with an alkaline hydroxide, such as sodium hydroxide, bringing the metal in intimate contact with the solvent for the impurities.

The apparatus used is represented diagrammatically in Fig. 4. It consists of a series of pots adapted to be heated from fire boxes, and having spouts for conducting molten metal from

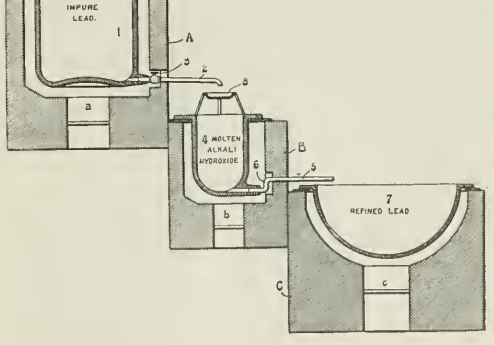


FIG. 4.—LEAD REFINING

one to another. The impure metal is contained in pot 1, and is conducted to the alkali pot by pipe 2, falling first into a perforated pan 8 placed above the alkali pot. This has the effect of disintegrating the molten stream into innumerable globules, exposing a large surface of metal to the action of the alkali. The refined metal falls to the bottom of the alkali pot and passes through the pipe and trap 5, 6, into the pot 7. It will be

inevitable that some of the alkali will also pass into pot 7, and this is removed by treating the molten mass with dry nitre cake, which is forced to the bottom of the pot 7. This quickly unites with any alkali in the mass and rises to the surface, forming a slag which is skimmed off.

If it is desired to refine lead alloys, such as those of tin and antimony with lead, it is necessary to retard the action of the alkali, so that tin and antimony will not be dissolved. This is done by adding a reducing agent such as zinc dust, iron filings or aluminium powder. In case iron scale is used, the inventor prefers to mix 20 parts of iron scale with 100 parts of alkali hydroxide in the reaction pot.

For the purpose of accelerating the removal of some impurities which are not readily acted upon, such as bismuth, tin and antimony, it is well to mix with the alkali hydroxide an oxidizing substance such as the nitrate, chlorate or peroxide of sodium or potassium. (1,006,323, Oct. 17, 1911.)

Extraction and Electrolytic Precipitation of Zinc.—The treatment of low grade zinc ores, containing from 5 to 15 per cent of zinc, with or without other metals, is a problem which attracts a great deal of attention from metallurgists. In a patent recently granted to Messrs. John R. Williams, Henry W. Bradley and Benjamin Bradley, of Sheffield, England, a process is outlined which contemplates the removal of the zinc by dissolution in a weak acid solution of zinc sulphate, and the electrolytic reduction of zinc from this solution.

Sulphide ores must be first roasted, after which they are leached in a suitable tank with a solution containing 6 to 8 per cent neutral zinc sulphate and 0.3 per cent sulphuric acid. These strengths were arrived at by experimentation. It was found that with quantities of zinc sulphate from 5 to 16 per cent the rate of deposition was no greater than at 5 per cent, while below 5 per cent the rate rapidly decreased. Also, with the higher percentages of zinc sulphate in the extracting solution, the loss of solution in the ore was too great. The acid strength is determined by the tendency of the ore to form a gelatinous mass with the acid, and by the effect of the acid on the cathode. With regard to gelatinization, the limit may be 2 per cent, but with regard to the action on the cathode the limit must be as low as 0.5 per cent. If the current density does not exceed 11 amp per square foot of lead anode surface, the anodes may be run for 300 days continuously without renewal. It is preferable to use a current density of about 10 amp per square foot of surface, and to start electrolysis with a neutral solution.

The inventors have a preferred form of cell which they use for electrolysis. It consists really of a number of practically separate cells parallel to one another. The cathodes are made of iron, preferably tempered and galvanized, and the anodes of lead plate. The anodes are placed one on either side of the cathode so that both surfaces of the latter are operative.

Copper ores may be similarly treated with a slightly acid solution of copper sulphate, or mixed copper and zinc ores may be leached out together and recovered separately by electrodeposition under proper electrical conditions. (1,006,330, Oct. 17, 1911.)

Electric Zinc Furnace.—Two patents of Mr. A. L. J. Queneau, of Philadelphia, Pa., refer to an electric zinc furnace which, in its latest construction, is shown in Fig. 5. It is a rotary furnace provided with a molten resistor D of molten iron containing 12 per cent phosphorus (to increase the electric resistivity) or preferably of molten slag. To get high fluidity of the slag, florspar or phosphoretic slag from an open-hearth steel furnace may be added. The furnace has an outer metallic shell *k* of low-carbon steel. At the feed end it is closed by a metallic plate *h*, insulated from the metallic shell *k* and provided with a flange *v* for making contact with one of the brushes *y* of the external electric circuit. The opposite end of the furnace is closed by a plate h^1 which is electrically connected with the metal shell *k*. The other brush *Z* of the external elective circuit makes contact with the shell *k* through the intermediary of the tire ring *d*. The shell *k* is provided with an outer lining of fire brick and an inner lining of magnesite brick or chrome brick. The end courses *C* are made of a conducting mixture of graphite, dead-burned magnesia, and tar. Exterior to these courses *C* are the raphite end linings *n*, adjacent to the end plates *h* and h^1. The electric circuit is, therefore, completed from brush *y* to plate *h*, to grpahite end lining *n* and conducting course *C* to the fluid resistor *D*, thence to the conducting course *C* at the other end of the chamber, the graphite lining *n*, plate h^1, shell *k* and ring *d* to the other brush *Z*.

At the left hand of the diagram the condenser is shown, with the shell k^1 lined with graphite compounded with clay Within the condenser are arranged the baffle dip partitions *L*, provided with apertures *o*. The construction may best be seen from the separate diagram *A* shown at the top left corner of

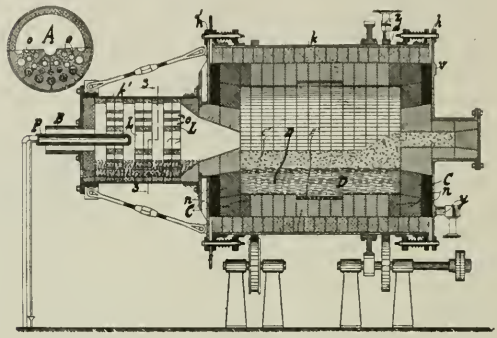


FIG. 5.—ELECTRIC ZINC FURNACE

the illustration, which gives a cross section of the condenser along the lines 33 in the main diagram. The combustible gases, such as carbon monoxide, escape from the condenser through the prolong *B*. To regulate the temperature in the condenser, the pipe *P* is provided through which cooling water is passed.

The charge consists of a mixture of zinc ore and coke, wherein the coke is but slightly in excess of that theoretically required for reduction of the ore. The charge is preheated to as high a temperature as the coke will stand without ignition, say 500° C, within a rotary furnace of the muffled type, heated from the outside and containing a number of steel balls so as to serve as a ball mill. The preheated charge is then supplied to the electric furnace. (1,006,876 and 1,006,877, Oct. 24, 1911.)

Nickel.

Treatment of Nickel-Copper Ores.—An improvement in the treatment of the nickel-copper ores of Sudbury, Ontario, Canada, is made the basis of a patent granted to Franklin R. Carpenter, deceased, and Arthur H. Carpenter, his son, of Denver, Col. The process has to do with the removal of the copper and sulphur from the ore, leaving the iron and nickel as a roasted product suitable for reduction to ferro-nickel alloy. The inventors first give the ore a dead roast, leaving the iron, copper and nickel practically as oxides. They then add a suitable amount of raw pyrite or pyrrhotite, together with the necessary salt, and give a chloridizing roast. Owing to the greater ease with which copper is chloridized, the process can be controlled in this respect, so that when the resulting mass is lixiviated only copper will be leached and nickel will remain with the iron. The nickel iron residue can then be smelted directly or with other iron ores for the production of ferro-nickel alloys, thereby saving the cost of separating the nickel and giving complete utilization of the iron. (1,006,355, Oct. 17, 1911.)

Alloys.

Ferrochromium Made in Continuous Resistance Furnace.—Instead of a discontinuously operating arc furnace, Mr.

EDGAR F. PRICE, of the Electro Metallurgical Company, proposes to use a continuous resistance furnace for the production of ferrochromium. A body of molten metal occupies the lower portion of the furnace. Above this rests a deep body of molten slag, and on the top of it is placed the charge mixture of chromite, coke, lime, etc. The body of slag is heated by a current passed through it between electrodes suspended from the top into the charge mixture and into the layer of molten slag. As reduction proceeds, the iron and chromium alloy and the alloy settle through the slag into the layer of metal beneath. The slag produced combines with the slag resistor and the excess slag and metal are removed from time to time through tap-holes, additions being made to the charge as required. The reduced metals thus quickly separate from and are maintained out of contact with the free carbon in the charge. The resulting alloy is, therefore, low in carbon. (995.-481, June 20, 1911.)

Nickel Titanium.—To produce a binary alloy of nickel and titanium, Mr. AUGUSTE J. ROSSI, of the Titanium Alloy Manufacturing Company, mixes nickel, nickel oxide and titanic acid with sufficient aluminium for the reduction of the oxides. The mixture is then heated in an electric furnace and the reduced titanium *in statu nascendi* alloys with the nickel. The reactions during the reductions are

$$3\ TiO_2 + 4\ Al + Ni = 2\ Al_2O_3 + Ni + 3\ Ti$$

and

$$3\ NiO + 2\ Al = Al_2O_3 + 3\ Ni$$

Instead of aluminium, carbon may be used as a reducing agent, but a higher temperature is then essential. (1,003,806, Sept. 19, 1911.)

Aluminium Bronze.—Mr. P. H. G. DURVILLE, of Paris, France, patents the use of manganese as a purifier for copper-aluminium alloys. He uses the following composition: 86 to 95 per cent copper, 0.05 to 5 per cent manganese and 6 to 11 per cent aluminium. The metals can be melted in the order indicated, but a portion at least of the manganese must be added in the form of an alloy with copper or with aluminium at the time of casting. Manganese in small proportions plays the part of a purifier as regards impurities other than oxides, which are not removed from the alloy by aluminium. The greatest proportion of manganese, up to 5 per cent, increases the hardness and the malleability of the alloy. These alloys have all the qualities of binary aluminium bronzes (consisting of copper and aluminium only), that is, great strength, good color and resistance to chemical agents, while being very easy to work. They are characterized moreover by the advantage that very sound ingots are obtained which can be easily forged, stamped, rolled or drawn. (1,007,548, Oct. 31, 1911.)

Pyrophoric Metal Alloy.

An improvement in pyrophoric bodies is claimed in a patent granted to GEORG F. HOFMANN, of Munich, Germany. It consists in adding metallic cerium to an alloy of manganese and antimony. Pyrophoric bodies of antimony and manganese are known, but their properties are not such as adapt them for use in ignition appliances, for the reason that they are not easily abraded by files or other frictional means. The inventor claims that the addition of about 5 per cent metallic cerium renders the antimony manganese alloy soft enough to be easily filed or abraded by friction so that particles fly from the body in a stat of incandescence. This alloy is different from that in which cerium is the main source of the pyrophoric property, where about 70 per cent cerium and 30 per cent iron are combined. (1,006,840, Oct. 24, 1911.)

Chemical Engineering.

Generating Steam from Molten Slag.—In Figs. 6 and 7 are shown two recent methods of generating steam from molten slag. Fig. 6 shows the arrangement designed by WALTER L. JOHNSON, of Middlesbrough, England. The main points in his invention comprise means for eliminating objectionable gases from the steam before use. Referring to the figure, *a* represents a slag spout which delivers hot slag from a furnace to the boiler *c*. A stirrer *d* may be used if desired. The boiler contains water and milk of lime for the absorption of sulphurous gases. The granulated slag is removed by elevators as shown, while the steam passes through the tower *h* to the tower *i* where it encounters a falling spray of hot caustic soda contained in the tank *k*. This is designed to neutralize any acid

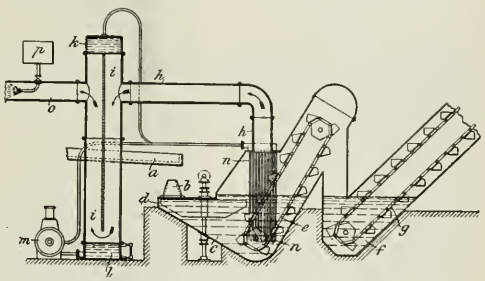


FIG. 6.—GENERATING STEAM FROM SLAG

gases, but a further neutralization is provided for by an ammonia spray *p* in the pipe *o*. The caustic solution is circulated by the pump *m* through coils *n* in the boiler, so that it is kept at a temperature of about 212 deg. Fahr. The ammonia may be recovered by condensing the engine exhaust. (1,005,711, Oct. 10, 1911.)

Fig. 7 represents the design of WILLIAM E. MCKEE, of Bisbee, Ariz. It compries a boiler and elongated leg in which is a bucket elevator for the removal of the granulated slag. The molten slag is first received in a hopper placed above the boiler. Between the hopper and boiler is a specially designed valve which can be closed to shut off the flow of slag and prevent the escape of steam. An opening for the escape of steam to the

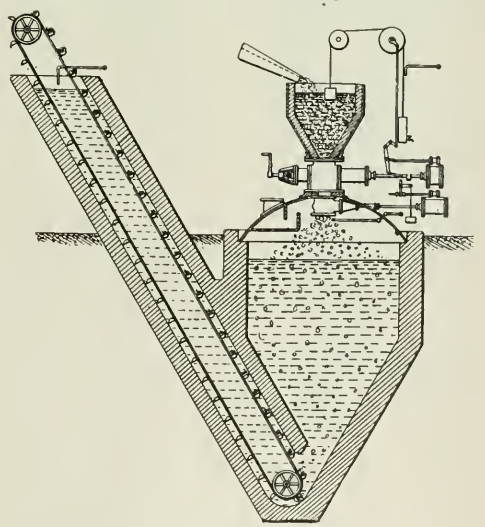


FIG. 7.—GENERATING STEAM FROM SLAG

point of utilization is shown in the boiler head. It will be noticed that provision is made for a difference in head of the water in the boiler proper and in the leg. This difference amounts to 2 ft. head for every pound of steam pressure in the generator. Feed water is introduced through the leg so that practically all the heat of the slag is utilized before its discharge. (1,001,290, Aug. 22, 1911.)

Electric Furnace Designs.

Electric Resistance Furnace.—In electric resistance furnaces with an electrode at the top and an electrode in the bottom, the upper electrode is kept in good contact with the upper part of the charge. When the charge sinks down in the furnace, this contact is liable to be broken and an arc is formed. This increases the resistance and reduces the power supply to the furnace and causes other disadvantages, like greater electrode consumption, concentration of the heat on the top, etc. To prevent this, Mr. Albert Petersson, of Odda, Norway, arranges the feeding device in such a way that the fresh charge is fed in a horizontal direction toward the upper electrode pressing against it and preventing the formation of arcs. Quite a number of such arrangements are described, of which one is shown in Fig. 8 (1,002,444, Sept. 5, 1911).

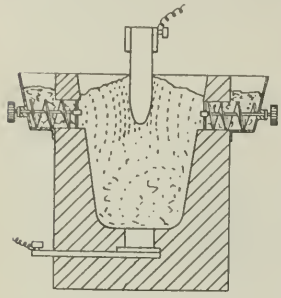


FIG. 8.—ELECTRIC RESISTANCE FURNACE

Electric Furnace with Conducting Hearth.—Mr. Georges Massip, of Paris, France, patents an electric furnace with an upper movable electrode, while the hearth itself acts as the lower electrode. The hearth is made of a material, like magnesia, which is a non-conductor of electricity when cold, but becomes a conductor when heated. To heat it, there are embedded within the hearth (but removed from the inner surface of the hearth) metallic heating resistors, suitably connected, like the upper electrode, to an electric circuit. (1,005,725, Oct. 10, 1911.)

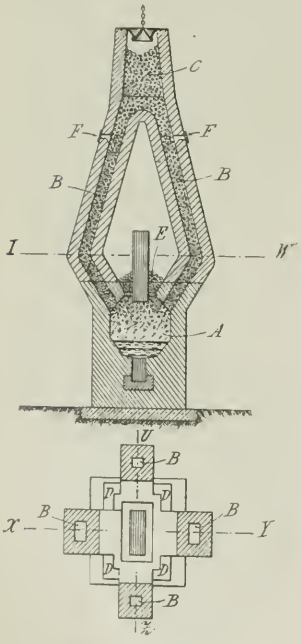


FIG. 9.—ELECTRIC REDUCTION FURNACE

Electric Reduction Furnace.—Mr. Hans Bie Loventzen, of Notodden, Norway, has patented the furnace shown in vertical and horizontal cross-section in Fig. 9 for operation with single-phase current by means of an upper and a lower electrode. Surrounding the upper electrode are a series of vertical channels *B* which terminate in a common charging shaft *C*. By means of a fan, the gaseous products of reaction are withdrawn through the channels. Through the valves *F* air may be blown into the channels, if desired. The furnace construction is such as to permit separate feeding of ore and carbon, namely, the ore *C* through the channels *B* and the coke *E* around the upper electrode. (996,492, June 27, 1911.)

Electric Furnace.—An electric-furnace construction of Mr. Alvois Helfenstein, of Vienna, Austria, is shown in Fig. 10, in vertical and horizontal cross-section, the two diagrams not being drawn to the same scale. The two side walls *A* and *B* are connected by means of the end walls *g* and *h*. On the top of the furnace two parallel supports d_1 and d_2 are arranged in the form of water-cooled girders, resting on the end walls of *g* and *h*. The two girders are spaced apart so as to form a charge opening, which extends the whole length of the furnace chamber, and a superposed receptacle *b* is provided to confine the charge. The lateral arched coverings of the furnace chamber a_1 and a_2 are likewise supported by those girders. An upper vertical electrode *i*, longitudinally arranged, is disposed in the charge opening and extends nearly the whole length of the furnace chamber. Horizontally arranged in the bottom *C* of the furnace is the bottom electrode *k*, also extending nearly the whole length of the furnace chamber. The openings *o* are provided for inspection and manipulation of the charge. As will be seen, on both sides of the furnace, above the charge, separate chambers e_1 and e_2 are formed into which the reduction gases can escape, being subsequently withdrawn through the outlets f_1 and f_2. (1,000,805, August 15, 1911.)

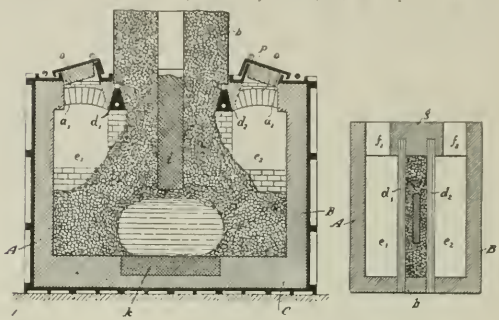


FIG. 10.—ELECTRIC REDUCTION FURNACE

In another patent of Mr. Helfenstein a slightly modified furnace construction is shown. The top of the charging shaft is closed and the whole arrangement is again such that free spaces are formed on the sides above the charge, into which the gases evolved during the reaction can escape. If they are combustible it is proposed to utilize their calorific effect within the furnace. For this purpose air is blown into these spaces and the gases are burnt, thus preheating the ore before it enters the melting zone. (1,002,988, Sept. 12, 1911.)

A third patent of Mr. Helfenstein refers to the construction of a battery of electric furnaces, all with one single common bottom electrode. That is, a single massive body serves as a common bottom to all of the furnaces and serves as bottom electrode for all of them. Upon this bottom various furnace shafts are constructed, whereby each single shaft may be used as an independent working hearth so that it is possible to conduct melting operations of different character in the different furnaces. (1,002,843, Sept. 12, 1911.)

Two-Phase Electric Furnace.—A patent of Mr. Axel R. Lindblad, of Ludvika, Sweden, refers to a two-phase furnace, which has already been the subject of an article by Mr. T. D. Robertson in our November issue (p. 573). The essential electrical connections are shown in Fig. 11. 1 and 2 are the

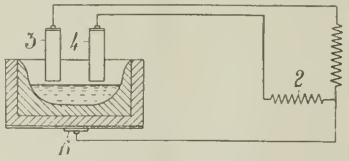


FIG. 11.—TWO-PHASE FURNACE

two phases of a two-phase system, each being connected with one of the upper electrodes 3 and 4, while the neutral point is connected with the contact 6 in the bottom of the furnace. As the two circuits have a phase difference of 90 deg. relatively to each other, the current which passes through the smelting bath and contact 6 is V_2 times the current passing through each of the electrodes 3 and 4. The tension between both electrodes is also V_2 times the tension consumed in each arc. If the current supply is three-phase, it is changed to two-phase by the well-known Scott transformation. (1,002,286, Sept. 5, 1911.)

Electric Furnace.—Mr. LARS YNGSTRÖM, of the Stova Kopparbergs Bergslage Aktiebolaz, of Falun, Sweden, patents the furnace shown in Fig. 12. A crucible 1 forms the working or melting chamber, and above it is provided a stack 2 for feeding the charge downward. But between the walls of the stack and the crucible there is a ring-shaped space 4 through which more charge is introduced into the furnace, while it also serves for the introduction of the electrodes 3, which are thus surrounded by charge material. (Bottom electrodes may also be provided.) A vault or roof for the crucible which would be liable to destruction is thereby avoided. 6 is the tapping hole. Two modifications of this furnace design are also described. (1,001,915, Aug. 29, 1911.)

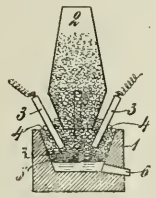


FIG. 12—ELECTRIC REDUCTION FURNACE.

Furnace Electrodes.

Dimensioning of Furnace Electrodes.—Mr. CARL HERING'S well-known laws of electrode losses are the basis of a recent patent granted to him for dimensioning electrodes. He proposes to dimension them in such a way that no heat passes through the hot electrode end (within the furnace) in either direction. No heat flows, therefore, from the furnace interior through the electrode to the outside. Only the heat generated within the electrode from electrical energy (the C^2R heat) passes out of the cold electrode and to the outside. The first claim reads as follows: "In an electric furnace, an electrode having a resistance such that the C^2R heat developed therein by current transmitted therethrough to said furnace shall be substantially equal to twice the heat conduction loss when no current flows." (990,719, Aug. 1, 1911.)

A second patent of Mr. Hering relates to this dimensioning of electrodes in combination with other features. It will suffice to give here the first claim: "In an electric furnace, molten conducting material serving as a furnace resistor, and an electrode in electrical communication with said resistor for communicating heating current to said resistor, the length and cross-section of said electrode being so related that the resistor heating current passed through said electrode raises said electrode to a temperature preventing heat conduction through said electrode from said molten material." (990,720, Aug. 1, 1911.)

Protective Coating for Carbon Electrodes.—To protect carbon electrodes against oxidizing influences, Mr. JOS. L. K. SNYDER, of the National Carbon Company, of Cleveland, Ohio, provides a protective coating to the electrode surface, while in a green (unbaked) and therefore plastic condition. During baking or later in the electric furnace this coating is automatically changed by the heat into a protective and very firmly adhesive slag. If the carbon mix, of which the green electrode is formed, contains acid impurities, like silicious material, sodium carbonate with a binder may be used for the coating. Under the influence of heat this will combine with the silicious material and form a slag-producing glazer. Or, if the carbon mix contains basic impurities, like ferric oxide, potassium bisulphate or borax with a binder may be used for the protective coating. Or calcium chloride may be used for the coating, as it will form with the carbon calcium carbide. Pitch and tar are suitable binders. (1,000,761, Aug. 15, 1911.)

Large Electrode.—The construction of an electrode for very large currents, like 25,000 amp., for calcium carbide furnaces is the subject of a patent of Messrs. WM. T. GIBBS and R. A. WITHERSPOON, of the Shawinigan Carbide Company. The head of the electrode (Fig. 13) consists of a casting of circular shallow horizontal box form *b* with a diametrical hollow flange *c* upon its top. The electrode head is water-cooled, *i* and *j* being the feed and discharge pipes for the box and *k* and *l* for the flange. The flange *c* acts as a terminal and also as a means for connecting the electrode with a crane. The horizontal box position *b* has a central opening encircled by a series of similar openings, all provided with screw-threaded sockets. A series of short graphite plugs *m* are screwed into the sockets in the head, and into their lower ends (as at *o*) are threaded amorphous carbon rods *p*. Thus these electrode pencils consist in their upper parts of graphite and in their lower parts of amorphous carbon. A sheet-iron shell encloses the pencils and a filler *s*, consisting of 100 parts by weight of ground coke, 27½ parts redistilled tar and 27½ parts pitch, is packed around and between the pencils, and is preferably molded and baked before the shell is set in place. The reason for using a graphite electrode in the water-cooled head is that amorphous carbon, when used in such water-cooled heads, is liable to break, while graphite does not break. For the lower parts of the pencils amorphous carbon is used on account of its cheapness. The electrode is shown in the illustration applied to a calcium carbide furnace, supported on a wheel truck 4 and having a carbon base 2 and limestone walls 3. (1,003,354, Sept. 12, 1911.)

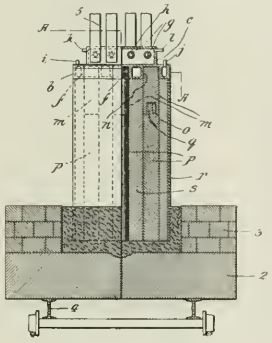


FIG. 13—LARGE ELECTRODE OF GRAPHITE AND CARBON.

Various Electric Furnace Processes.

(See also above under the headings for the different metals.)

Electrically Heating Carbon.—In electrically heating carbon, for the production of electrode compositions, battery fillers, etc., a close adjustment of time and temperature is required to convert the given raw material into a product of the desired character. While in practice the intermittently operated furnace is always employed, Mr. WILLIAM ACHESON SMITH, of the International Acheson Graphite Company, has devised a continuously operating furnace in which the charge is progressively advanced in a direction parallel to the lines of flow of the electric current. In this way the charge may be subjected for any desired period (depending upon the rate of advance of the charge) to uniform temperature conditions. Two slightly different designs of the furnace are shown in Fig. 14. In the left-hand drawing the furnace charge 8 serves as the resistor interposed between the upper and lower electrodes 6 and 7. The lower electrode is supported by a hollow water-cooled metal bar or casing 9 carried by the furnace walls. In the furnace walls adjacent the heating zone a hollow space 4 is provided which communicates by inclined apertures 4*a* both with the heating chamber and the external air so that the furnace gases may readily escape from the heating zone and a mantle of such gases may be burned within the furnace wall to economize heat. Water pipes 5 are provided between the heating zone and the discharge hopper. The latter is provided with a slightly coned disk, rotated by a worm gear, so that the material is discharged over the periphery of this disk only while it is rotating. In the right-hand diagram the upper and lower terminals 6 and 7 are connected together by a carbon or graphite rod 14, which is particularly useful when the charge,

in the cold state, is a poor conductor of electricity. There are also some other small differences in design in the two furnaces, which can be seen in the illustration. (Process patent 1,004,923; furnace patent 1,004,924, both of Oct. 3, 1911.)

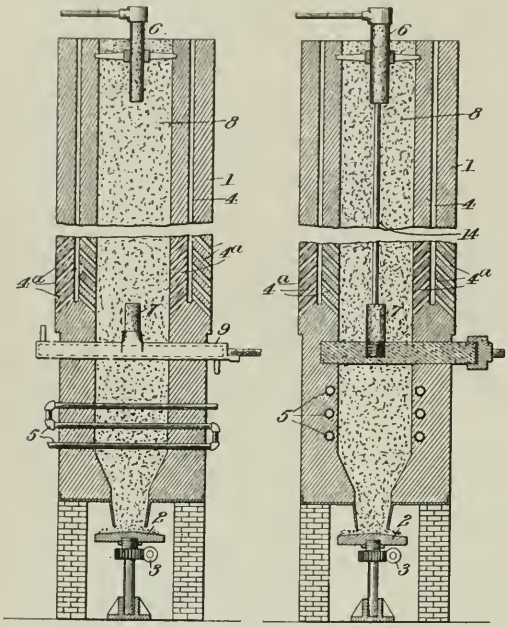


FIG. 14.—CONTINUOUS FURNACE FOR HEAT TREATMENT OF CARBON.

Bulk Graphite.—In the conversion of carbon into graphite in electric furnaces it is necessary to shield the carbon from the direct action of the air, but to permit the free escape of the distillation gases. For this purpose the carbon is surrounded by a granular mass of carborundum, white crust, etc. Under usual operating conditions it is impossible to maintain a sharply defined demarcation plane between the carbon and the surrounding material, especially near the upper and lower surfaces of the charge. It is, therefore, usually necessary, in unloading each furnace charge, to discard a considerable mass of graphite, by reason of the admixed carborundum, etc. Mr. Alfred L. Robinson, of the International Acheson Graphite Company, overcomes this difficulty by the furnace construction shown in Fig. 15. The crushed petroleum coke or gas coke 2,

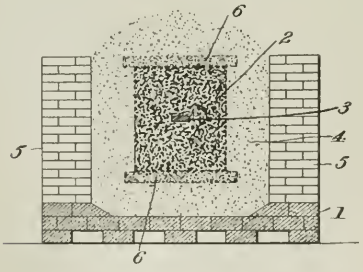


FIG. 15.—GRAPHITE FURNACE

which is to be changed into graphite, is placed between two lower and upper layers 6 of a mixture which during the furnace treatment cakes into a strong solid mass (for instance, a mixture of 25 parts of bituminous coal, 50 ground coke and 25 ground furnace crust, consisting of carborundum, siloxicon, etc.). 3 is a starting core of carbon, used when the initial conductivity of the charge is low; 4 is the embedding material supported by the walls 5 of loosely assembled firebrick. In charging the furnace admixture of the coke with the embedding material is avoided by the use of vertical separating walls of paper. At the close of the operation it is easy to take the furnace to pieces. (1,004,469, Sept. 26, 1911.)

Abrasive and Refractory Material of Alumina and Magnesia.—Research work carried out by the Carborundum Company, of Niagara Falls, has resulted in the production of several interesting new products, for abrasive and refractory purposes, which are disclosed in some patents of Mr. Thomas B. Allen. All of them have alumina as chief constituent. The first product consists essentially of alumina and magnesia, with not over 25 per cent magnesia. Bauxite is used as source of alumina and magnesite as source of magnesia. They are calcined to remove the water from the bauxite and the carbon dioxide and water from the magnesite and the calcined materials are then mixed together with a sufficient amount of carbon to reduce the major portion of the oxide of iron and oxide of silicon. For example, a mixture of 100 parts of calcined bauxite, 15 calcined magnesite, and 6 carbon, treated in an electric furnace, yields a product consisting of 14 parts MgO and 84 $Al_2 O_3$, the remainder consisting of oxide of titanium, iron and silicon. By keeping the temperature of the electric furnace relatively low, the oxide of iron and silicon can be reduced without affecting the oxides of aluminium and magnesium. The resulting ferrosilicon can be separated. The content of magnesia in the finished product affects its toughness. It should not be more than 25 per cent, and by varying it a series of products of varying hardness and toughness can be prepared. (1,001,497 Aug. 22, 1911.)

Abrasive and Refractory Material of Alumina, Magnesia and Boric Oxide.—In a second patent of Mr. Allen boric oxide is added to the alumina-magnesia product. Boric acid is mixed with calcined magnesia, for instance, in the proportion of 35 to 65, and heated in a fuel-fired furnace to about 1300 deg. The product is crushed and 25 parts of it are mixed with 100 parts of calcined bauxite and 8 parts coke and this mixture treated in an electric arc furnace. The product contains 12 per cent magnesia, 6 per cent boric oxide, and 79 per cent alumina, and is a dark colored crystalline material of great hardness and very brittle. Addition of boric oxide increases the brittleness and hardness. (1,001,570, Aug. 22, 1911.)

Abrasive and Refractory Material of Alumina and Beryllia.—A material consisting of alumina and less than 18 per cent of beryllia (that is, less than represented by the formula Be O $Al_2 O_3$) is the subject of a third patent of Mr. Allen. He treats in an electric furnace a mixture of 100 parts by weight of calcined bauxite, 75 parts of beryl, 12 parts of coke (to reduce the oxides of silicon and iron) and 50 parts of oxide of iron (to remove more readily the reduced silicon and titanium). The product contains 7½ per cent of beryllia and 89 of alumina. The content of beryllia gives to the product an improved crystallization, while the toughness decreases with increase of beryllia and the hardness increases with increase of beryllia. (1,001,571, Aug. 22, 1911.)

Purification of Abrasive and Refractory Materials.—In a fourth patent of Mr. Allen a method is described for removing aluminium carbide and ferrosilicon, which are the chief impurities in abrasives produced in an electric furnace. Sodium chloride under the action of heat is used as the purifying agent. The crushed abrasive material is mixed with sodium chloride and heated to a temperature of from 800 to 1200 deg. C. This is followed by washing with water or a dilute solution of a mineral acid, in order to remove any excess of sodium chloride or products of the reaction. Two to five per cent by weight of sodium chloride added to the fused alumina are sufficient for purification. The reactions which eliminate the ferrosilicon and aluminium carbide are given as follows:

$$Fe_4\ Si + 16\ Na\ Cl = 4\ Fe\ Cl_3 + Si\ Cl_4 + 16\ Na$$

and

$$Al_4\ C_3 + 12\ Na\ Cl = 4\ Al\ Cl_3 + 3\ C + 12\ Na$$

These reactions are reversible, going from left to right at high temperatures. If they are carried out at a sufficiently high temperature in an oxidizing atmosphere, then, in spite of their reversibility, these reactions can be carried through to a finish, because the Fe Cl_3, the Si Cl_4, etc., are volatilized as soon as formed and the Na and C are oxidized, thus removing all newly formed compounds (those given on the right hand of the equation) from the sphere of action, and permitting the above reactions to go on continuously without stopping at some intermediate equilibrium. (1,001,572, Aug. 22, 1911.)

Purification of Emery Ore.—Emery ore consists principally of alumina, but is always associated with impurities such as iron oxide and usually silica and often small quantities of titanium oxide, etc. To remove these impurities, Mr. FRANK J. TONE, of the Carborundum Company, of Niagara Falls, employs an electric-furnace process, the impurities being reduced by means of carbon. It is important not to reduce any alumina, since in this case an entirely different, easily friable product would be obtained. For this reason only sufficient carbon is used to reduce the impurities. Further, the temperature is maintained above the reduction temperature of silica, but below that of alumina. As this would be impossible with an arc furnace, a resistance furnace is used. The mass of emery ore and carbon is placed between the electrode terminals of a furnace and current is passed through it. As the heating of the mass progresses and it becomes partially fused, its conductivity increases, and when it reaches a well-fused condition the charge becomes a good conductor of electricity. The furnace now acts as a simple resistance furnace. The purified fused alumina collects at the bottom with the impure alumina resting on it on top. The lower layer of pure fused alumina is a poor conductor of electricity, so that the distribution of the current is self-adjusting, the current confining itself practically to the layer of impure alumina on the top. The electrodes are accordingly raised from time to time. (1,002,608, Sept. 5, 1911.)

Boron.—A number of patents of Dr. E. WEINTRAUB, of the General Electric Company, deal with an elaborate method in producing pure boron. The usual method for producing boron includes the reduction of boric anhydride with magnesium, the general process being to heat a mixture of magnesium with an excess of boric anhydride. This reaction is accompanied by the production of magnesium borate and by the formation of other compounds. The resultant product, therefore, contains a relatively high percentage of material other than boron. The usual washing with water and hydrochloric acid decreases the percentage of impurity, but does not yield a product which could be termed, in the chemical sense, "pure boron." This impure product can be greatly improved, however, by "fractional decantation," consisting in grinding the product with water or hydrochloric acid and then decanting off the top of the liquid before complete precipitation has occurred. Boron has a greater specific gravity than the impurities, such as magnesium boride. By repeating this treatment, the proportion of impurities can be reduced several per cent.

The resulting product, in form of a brown powder, is compressed into the form of a stick or rod, say 6 cm long and 5 mm in diameter. The stick is then highly heated in an electric vacuum furnace to dissociate and drive off various of the impurities, like magnesium boride and boron hydride. This treatment may be continued for several hours at a temperature of say 1200 deg. C. without any danger of contaminating the boron by the carbon of the resistance heater. As a result of this treatment, an important physical change takes place, since the boron stick which before the treatment was a non-conductor becomes a relatively good conductor of electricity.

The next step in this process consists in melting down this conductive boron by heating it as anode in a vacuum arc. For this purpose an electric vacuum furnace is used in form of a hermetically closed glass vessel, with some mercury as cathode and the boron stick as anode. Hydrogen gas is introduced into the vessel and a pressure of the hydrogen of from 2½ to 15 cm of mercury is maintained. This hydrogen gas is chemically inert, but serves to concentrate the arc on the lower end of the boron stick.

While chemical text-books describe boron as being a non-conductive material which vaporizes at high temperatures without melting, the boron produced by the above process is a relatively good conductor of electricity, is a dense and solid body and undergoes complete fusion without apparent vaporization. Such pure conductive boron may be used for a variety of purposes and may be shaped into rods or glow bodies for use in incandescent lamps and furnaces. (997,879; 997,880; 997,881; all of July 11, 1911.)

Vacuum Furnace for Treatment of Refractory Materials.—The same principle of melting a refractory material as anode in a vacuum mercury arc may be applied to the treatment of refractory materials other than boron. For instance, for the fusion and purification of tungsten, thorium, osmium, etc., in form of rods pressed from the powdered metal. Several vacuum furnaces for this purpose are described by Dr. Weintraub. The principle is the same in all. (997,882 and 997,883, July 11, 1911.)

Distillation Process for Antimony and Arsenic.—Distillation at reduced pressure is one of the features of a process for treating ores containing antimony and arsenic, devised by Dr. COLIN G. FINK, of the General Electric Company. It may be applied, for instance, to electrolytic slimes containing antimony, and a very pure antimony can be obtained from a slime containing 35 per cent of antimony, the yield being almost 95 per cent. At atmospheric pressure the vaporization point of antimony is between 1500 and 1700 deg. C. At a pressure of 5 to 6 mm mercury it is only 735 deg. C. At a pressure of 40 mm mercury, as used in the author's process, it is between 1200 and 1300 deg. C. For the treatment of stibnite, $Sb_2\ S_3$, using an ore containing 62 per cent of antimony, 11.3 parts of stibnite, 12 parts of limestone, 2½ parts of coke and 6 parts of fluorspar are ground to 20 mesh and charged into an air-tight electric resistance furnace, the charge serving as resistor. The temperature is raised to 600 or 700 deg. C. and maintained there for one and a half hours to let the following reaction proceed:

$$Sb_2\ S_3 + 3\ Ca\ CO_3 + 6\ C = 2\ Sb + 3\ CaS + 9\ CO$$

A vacuum of 40 mm mercury is maintained by constant operation of the air pump. Then the temperature is raised to 1300 deg. C. and maintained there for about one hour, while the antimony is distilled off and condensed in a water-cooled condenser which is in communication with the furnace chamber.

The reduction of the stibnite may also be accomplished by heating the powdered ore with finely divided iron. Two parts by weight of stibnite are mixed with one of iron. The temperature of the charge is raised to 700 deg. C. for 20 minutes, to reduce the stibnite:

$$Sb_2\ S_3 + 3\ Fe = 3\ Fe\ S + 2\ Sb.$$

The temperature is then raised to 1200 or 1300 deg. C. for an hour and the antimony distilled and condensed. Instead of metallic iron, an oxide ore, as hematite or magnetite, may be used, the oxygen of the hematite serving partly to unite with the sulphur of the stibnite to form sulphur dioxide, SO_2, and the remaining sulphur combining with the reduced iron to form sulphide of iron. From two to three parts by weight of powdered carbon to one part of ore are added to the charge, to increase its conductivity and to permit the carbon monoxide formed to keep the charge porous.

Instead of an iron ore, a chrome iron ore, like chromite, may be used, $Fe\ O\ Cr_2\ O_3$. For example, about four parts of finely ground stibnite are mixed with two parts of finely ground chrome iron ore. One part of powdered coke or charcoal is added for the purposes mentioned. The finely ground mixture is heated to 900 deg. for one or two hours, to reduce the antimony and chromium:

$$3\ Fe\ Cr_2\ O_4 + Sb_2\ S_3 + 12\ C = 3\ Fe\ S + 12\ CO + 2\ Sb + 6\ Cr$$
$$2\ Fe\ Cr_2\ O_4 + Sb_2\ S_3 + 6\ C = 2\ Fe\ S + SO_2 + 6\ CO + 2\ Sb + 4\ Cr.$$

The temperature is then raised to 1500 deg. C. for one or two hours to distil and condense the antimony. (996,474, June 27, 1911.)

Silica Glass.—Mr. H. A. Kent, of the Silica Syndicate, of London, England, patents a process of making silica glass (fused silica), the first claim reading as follows: "The manufacture of silica glass by causing grains or particles of silica to fall or be projected separately through a heated zone and onto a movable heated body in a chamber, said particles being subjected to a temperature sufficient to sinter them by this single operation into a homogeneous mass of silica glass." One method of making a silica rod or tube is shown in Fig. 16.

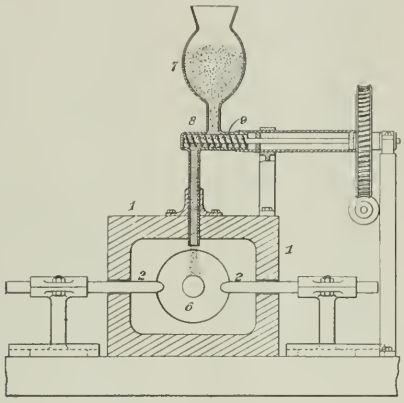


FIG. 16.—MANUFACTURE OF SILICA GLASS.

The electric furnace 1 is provided with adjustable electrodes 2. The ground silica grains are carried from the container 7 through the tube 8 with the revolving worm 9 and drop down onto the heated cylinder 6 of silica-glass upon which they form a homogeneous mass of further silica glass. (1,003,271, Sept. 12, 1911.)

Burning Cement with Electric Heat.—Methods have been proposed for burning cement by the direct heat of the ordinary short arc, but for practical reasons the scheme has not been adopted. Mr. Carleton Ellis, of Montclair, N. J., has recently patented a process for the utilization of a long arc in cement burning. An alternating current of from 4000 to 6000 volts is used, and this for commercial reasons may be a three-phase system. Three or more arcs may be arranged to work on the three phases of such a system, with the kiln connecting with the neutral junction. With a kiln about 20 ft. long and about 20 in. in diameter, a current of from 200 to 300 amp and at 4000 to 6000 volts may be used. The length of the kiln determines the voltage necessary to strike the arc, while the amperage is adjusted in accordance with the heating effect required.

In order to prevent the arc "blowing out," the inventor injects a spiral current of air moving tangentially around the arc, as by this means he is enabled to project arcs of great length without loss of stability. In order to protect the lining of the kiln as much as possible, the kiln may be rotated at high speed so as to carry the load well up on the side of the kiln, and by the application of the electrode magnet the arc may be deflected at one or more points into close contact with the material to be clinkered, and away from the opposite and unprotected side of the kiln. (999,494, Aug. 1, 1911.)

Gas Reactions.

Nitric Acid from Air.—The Pauling process for the fixation of atmospheric nitrogen by the electric arc has come into considerable prominence by the two commercial plants in Austria and France employing it on a large scale (this journal, Vol VII, p. 430; Vol. VIII, p. 102, 196) and also by the decision to use it in a plant in course of erection in the South. In the existing Pauling plants the arrangement is very similar to that of the horn lightning arrester, as described in the articles referred to. It appears, however, from two recent patents of Mr. Harry Pauling that he is also paying attention to quite different designs, using a long vertical arc. But the arrangement is very different, for instance, from the long arc of the Badische Company's furnace. While in the latter the air passes in spiral form along the whole vertical arc from the bottom to the top, Pauling provides gas inlets and outlets alternately all along the arc. The furnace has a central vertical open shaft within which the arc plays. The walls contain gas outlets alternately from the bottom to the top. When the air enters through one of the inlets it travels only a comparatively short distance along the arc, during which time the nitrogen combines with the oxygen, and the gas mixture formed is then withdrawn. The principal advantage claimed for the arrangement is that it is possible to reduce the velocity of the air current to a value which obviates all risk of blowing out the arc. (999,586, Aug. 1, 1911.)

A second patent of Mr. Pauling refers to the difficulty experienced with a long arc in a comparatively narrow shaft on account of the fact that the walls of the shaft being heated to a high temperature may become conductors and carry some of the current. This is prevented by subdividing the surface of the furnace walls facing the flame into a number of areas or elements, each of which is separated from the surrounding elements by gas passages which penetrate into the wall to such a depth as to obviate any risk of the wall acting as a conductor when it becomes hot. (999,587, Aug. 1, 1911.)

The long vertical arc of the Schoenherr furnace of the Badische Company was originally caused to play between a solid iron core as lower electrodes (where the air enters) and a hollow cooled upper electrode (where the gas mixture leaves). In their latest construction Dr. Otto Schoenherr and Mr. J. Hessberger employ a hollow cooled electrode also at the lower entrance end of the furnace, arranged in such a way that the starting point of the arc at this lower electrode moves about on its surface, while the arc as whole remains stable and stationary. (1,003,299, Sept. 12, 1911.)

In a patent of Mr. Johan Jetmund Thoresen, of Kristiania, and Mr. Filip Tharaldsen, of Meraker, Norway, the method is patented for treating air by means of an arc made to rotate by a rotary magnetic field. The process is described by reference to the induction motor in which the primary pole pieces produce a rotary magnetic field and this rotating field causes the secondary (the rotor or the armature) to rotate. "The electro-dynamic action between the electrically rotatory primary field and the current induced in the induction coils produces a torque, which causes the secondary core to rotate along with the said field. By preventing the secondary core as well as the coils from rotating and by making a rupture in the said coils and placing the ends of the rupture (electrodes) in the field perpendicular to the direction of the latter, an electric arc or series of electric discharges will be produced in the space between the cores, and the current will pass through said arcs perpendicularly to the field. The electro-dynamic force produced by the combined action of the current in the arcs and rotary field will cause the arcs to be deviated to one side and in the direction in which the field rotates. As the direction of the lines of force of the rotary field is determined by the shape of the field cores and the path of the arc being always perpendicular to the field, it is evident that by giving the field cores a cylindrical or other shape and moving the field accordingly the arcs will assume a cylindrical or other shape. By making the cores cylindrical and causing the field to rotate, the elec-

trodes being placed at suitable intervals, we obtain a continuous row of arcs, each forming a part of a cylindrical surface, and all the said arcs together constitute a complete cylindrical surface. The arc may be supplied with current from a separate single, two or three-phase transformer or from a source of direct current instead of supplying the same with current from the secondary coils. It may be necessary to have in series with the arcs a certain amount of self-induction in order to impede the flow of current. We can also connect the electrodes in series with the coils that produce the magnetic field." (996,951, July 4, 1911.)

Mr. CARLETON ELLIS, of Montclair, N. J., uses a long arc and causes it to revolve about its normal axis in the path of the current of air and at a slightly lower speed than the current of air. The extinguishing effect of the air blast is minimized by the almost equal speeds of arc and air, while "the aureola contact is greatly reduced over methods which involve the travel of the air longitudinally with the arc. The process therefore yields a large output with simplicity of furnace construction, small number of units and therefore reduced labor cost and maximum electrical efficiency." (1,002,249, Sept. 5, 1911.)

Ozone Generators.—The details of construction of an ozonizer, invented by Mr. LOUIS GOLDBURG, of Indianapolis, Ind., are well described in his third claim: "An ozonator, comprising a casing having a contracted inlet opening at one end, a block fitting within said casing adjacent to said inlet and having therethrough a plurality of distributed air passages communicating with said inlet, a cylindrical shoulder formed on said block, a tubular dielectric fitting said shoulder and adapted to be supported thereby in said casing, an outer electrode disposed around said dielectric tube, an inner electrode having screw-threaded connection with said block and located concentrically within said tube, said electrodes being formed to provide opposed projections distributed throughout the contiguous areas of said electrodes, the projections on one of said electrodes being uniformly spaced away from those on the other, the end of said casing which is opposite said contracted air inlet being arranged to permit said block, dielectric tube and electrodes to be readily withdrawn therethrough, and means for connecting said electrodes with a suitable source of electromotive force." (995,958, June 20, 1911.)

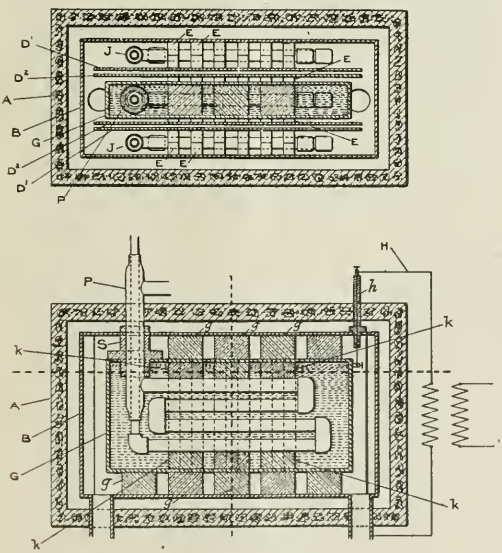


FIG. 17.—OZONIZER.

In an ozonizer of Mr. JAN STEYNIS, the chief feature is the elimination of some of the heavy insulation usually required. This object is obtained by grounding one side of the high-tension circuit and also grounding the cooling apparatus. Fig. 17 shows a plan and a central longitudinal vertical section of the ozonizer. *A* is the external casing, *B* is a box of thin metal connected to the grounded side of the circuit. On each of the longer sides of box *B* and on the inside thereof are a series of vertical metallic bars *E*, *E*, etc., which constitute the discharge points of the electrode. The bars *E* are perforated at intervals in a direction parallel to the longer side of the box, and through these perforations extend a series of metallic pipes forming a cooling coil *J* for receiving a gasified liquid, or in some cases cold water or brine. Since the said box *B* is grounded, the pipe coil need not be insulated as heretofore. Against the faces of the discharge bars *E* is placed a thin sheet of glass or micanite D^1 which is separated a slight distance from a parallel sheet of like material resting against the faces of similar discharge bars on the outside of the central electrode. Between these sheets passes the air to be ozonized.

The central electrode consists of an internal metallic box *G* with the vertical discharge bars *E*, *E*, etc., on its outer sides. The box is insulated from the surrounding box by blocks, *g*, *g*, etc., and is connected to one side of the circuit by a wire *H*, passing through and insulating tube *h* in the top side of the box *B*. Within the central metallic box *G* is a cooling coil *P*, like the coils *J*. It is, however, insulated by oil or other liquid insulator from the box *G* in which it is contained, being supported in the box by insulating blocks *k*, *k*, etc., and where it passes through the top side of the box the outgoing end is enlarged to receive within it the incoming end, and thereby call for only one insulating bushing *S*. The coil *P* will be connected to the coils *J* and to ground, so that the only insulation required in the apparatus is that for the central electrode. (997,339, July 11, 1911.)

Electrolytic Furnaces.

Partition for Electrolytic Furnaces.—In electrolytic furnaces partitions are sometimes required, for instance, for separating the anode gases from the metal deposited on the cathode. It has been proposed to form such non-conductive partitions by chilling a layer of the molten electrolyte itself against a water-cooled metallic curtain. However, it is not possible to maintain such layer of chilled electrolyte of sufficient thickness and at a sufficiently low temperature to avoid its conducting a portion of the current from the molten electrolyte to the curtain. Messrs. GEORGE O. SEWARD, of East Orange, N. J., and FRANZ VON KÜGELGEN, of Holcombs Rock, Va., in a patent assigned to the Virginia Laboratory Company, overcome this difficulty by using two curtains, side by side but insulated from each other,

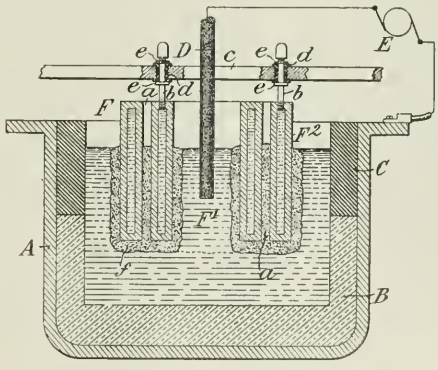


FIG. 18.—PARTITIONS FOR ELECTROLYTIC FURNACES.

instead of a single one. In Fig. 18 *A* is the circular electrolytic vessel or furnace with an insulating lining *B* in its lower part and a conductive carbon lining *C* at its upper part, which serves as anode. In the center of the furnace is the cathode *D*. Between anode and cathode is provided the non-conductive partition *F* which consists of two tubular concentrical cylindrical water-cooled curtains F^1 and F^2, between which is an annular

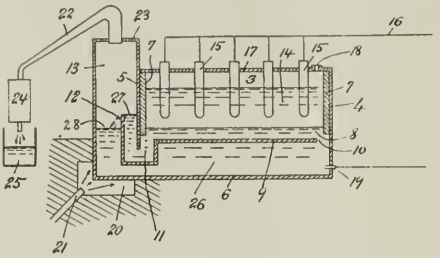


FIG. 19.—PRODUCTION OF METALLIC SODIUM FROM FUSED SODIUM CHLORIDE.

space *a*, which is kept filled with congealed salt or packed with insulating material, such as fire-clay or asbestos. (13,277, July 25, 1911.)

Metallic Sodium by Electrolysis of Fused Sodium Chloride.—In the electrolysis of fused sodium chloride with a cathode of molten lead the sodium set free at the cathode alloys with the lead and it is necessary to remove the sodium-lead alloy from contact with the molten electrolyte for two purposes: First to provide a fresh molten lead cathode surface ready to alloy with sodium, and second to recover the sodium from the sodium-lead alloy in the form in which it may be desired, for instance, as metallic sodium. For this removal of the lead-sodium alloy from the electrolytic cell, Mr. Robert J. McNitt, of Niagara Falls, N. Y., in a patent assigned to the Roessler & Hasslacher Chemical Company, makes use of differences of specific gravity to provide the circulation of the molten electrode material. Pure molten lead has a higher specific gravity than a molten alloy of lead and sodium. If, therefore, in Fig. 19 the channel 26 is first filled with molten pure lead and the compartment 8 (just above the partition 9) and the seal 11 with a molten alloy of lead and sodium on which rests the molten electrolyte 14 of sodium chloride, there will be a difference between the hydrostatic levels at the points 27 and 28. When a direct current is then passed from the anodes 15 through the sodium chloride 14 to the lead cathode 8 (electrical connection being made at 19 to the external circuit), the sodium chloride is decomposed, the sodium alloying with the molten lead 8. More sodium chloride is now fed into the cell, etc. Meanwhile, of course, a portion of the cathode alloy is forced down through seal 11 under wall 5, over wall 12 into the outside furnace chamber 13. In this chamber 13 the temperature is maintained high enough so that some or all of the sodium in the lead-sodium alloy is driven off by distillation and is condensed in 24. There remains back in 26 a weak alloy which returns through opening 10 into the electrolytic chamber 3. The difference of specific gravity is, therefore, utilized to maintain a difference of levels at 27 and 28 and therefore provide an overflow from 27 to 28. (995,476, June 20, 1911.)

Electrolytic Processes.

Mercury Cathode Cell.—A patent of Mr. Meyer Wildermann, of Ealings, London, England, refers to improvements in his mercury cathode cell, in which the mercury is contained in superimposed troughs (see our vol. I, page 589). The details of improvement are indicated in the first claims: "In electrolytic cells for the electrolysis of alkaline salts provided with a mercury cathode contained in superimposed troughs, with division ribs dipping into the mercury, with stirrers for mixing the mercury and amalgam on the anode side, with carbons floating on the mercury on the cathode side, the provisions of means for preventing the carbons from rising over a certain height in the troughs. (998,063, July 18, 1911.)

White Lead.—A process differing in various respects from his former white-lead process is the subject of a patent of Dr. Carl Luckow, of Cologne, Germany. While the specification

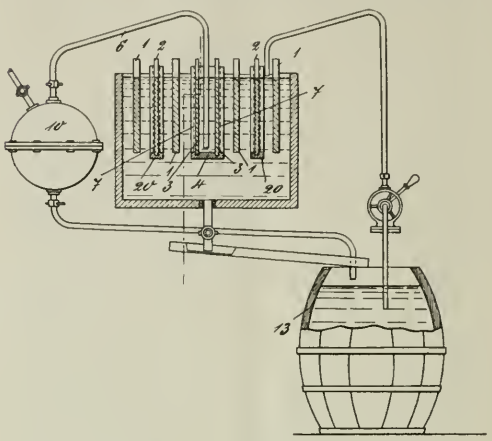


FIG. 20.—ELECTROLYTIC PRODUCTION OF WHITE LEAD.

relates in general to the electrolytic manufacture of difficultly soluble salts of heavy metals and gives data on the production of chrome yellow, Prussian blue and Paris blue and white lead, only the white-lead process will here be described. The anodes 1 of Fig. 20 consist of soft lead. The cathodes 2 are skeletons of soft lead or hard lead with active fillings of spongy lead oxide. To prevent enrichment of the electrolyte with caustic alkalies, the dilute solution caustic, as it is being formed during electrolysis, is constantly withdrawn by means of what is called "filtering cathodes" 3, 4. All the cathodes 2, including the filtering cathodes 3, 4, are inclosed with wooden frames 20 and filtering cloths 7. The electrolyte is a diluted solution containing from 1 to 2 per cent of a mixture of from 90 to 95 parts by weight of sodium chlorate and 10 to 5 parts of sodium carbonate calculated free from water. During electrolysis the carbonate and water are always added as they are consumed, while the soda-lye is always extracted, as it is

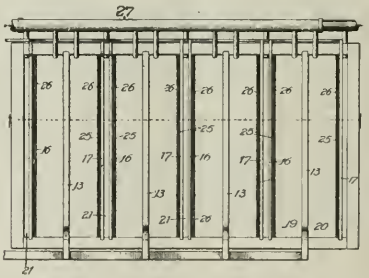


FIG. 21.—ELECTROLYTIC TREATMENT OF WASTE LIQUORS.

produced, from the filtering cathodes 3, 4, by means of a syphon 6 and passed through a receptacle 10 for carbonic acid into the tank 13, from whence it is fed back into the cell. (1,006,389, Oct. 17.)

Electrolytic Recovery of Waste Liquor.—In a patent assigned to the National Tube Company, Mr. Fred. F. Farnham, of McKeesport, Pa., recovers iron and sulphuric acid

from solutions of ferrous sulphate resulting from cleaning steel by means of sulphuric acid. He uses a diaphragm cell, in which diaphragms of asbestos 25 with an outer layer of burlap 26 (see Fig. 21) are fastened to the perforated lead anodes 16 and 17. The iron sheets 13 are the cathodes. Between each cathode 13 and the adjacent anodes 16 and 17 are formed the cathode chambers 19 and 20, while between adjacent anodes 16 and 17 the anode chambers 21 are formed. During electrolysis sulphuric acid accumulates at the anodes and iron is deposited on the cathode. The ferrous sulphate solution is fed from the inlet pipe 27 into the cathode compartments. It gradually seeps or filters through the diaphragms and the associated anodes into the anode compartments. While the solution is in contact with the anodes, sulphur dioxide is formed with the consequent formation of sulphurous acid, which is immediately oxidized to sulphuric acid, the reactions being $SO_2 + H_2O = H_2SO_3$ and $2\,H_2SO_3 + O_2 = 2\,H_2SO_4$. This sulphuric acid is withdrawn from the anode chamber through the outlet pipe. (1,006,836, Oct. 24, 1911.)

Synopsis of Metallurgical and Chemical Literature

Gold and Silver.

Melting Zinc-Box Precipitate.—Some interesting points bearing on the production of high-grade bullion from zinc-box precipitate is given in the August, 1911, issue of the *Monthly Journal* of the Chamber of Mines of Western Australia, by Messrs. J. B. AARONS and HERBERT BLACK. The authors were confronted with conditions which gave them low-grade bullion, and experimented with a view to raising the grade and thereby reducing refining and banking charges.

Their present procedure is as follows: All the precipitate and short zinc are sent to the acid vat, and only the well-washed long zinc is returned to the boxes. No scrubbing or screening of short zinc is attempted. After acid treatment the precipitate is filter-pressed, washed three or four times with hot water, dried with air and dropped into cast-iron trays, weighed and placed in a cast-iron muffle, where it is subjected to a roast at dull red heat, or merely dried. The dried or roasted precipitate is not screened, but is directly fluxed with borax and sand, and in some cases with nitre. A typical charge is 100 parts precipitate, 50 parts borax, 20 parts sand, and when nitre is used, 7 to 10 parts nitre. The charge is smelted in clay liners in plumbago pots and poured into conical molds. The bullion thus obtained is subsequently melted in plumbago pots and cast in bars of 500 oz.

The authors believe that it is easier to produce high-grade bullion direct from the precipitate than to produce low-grade bullion direct from the precipitate and afterward refine it; moreover, there is less chance of loss when the refining is carried out through the medium of the flux than in the refining of the metal. They further believe that if the acid treatment and subsequent washing are carried out thoroughly, roasting to dull red heat is unnecessary, and that drying at 100 degrees Centigrade is sufficient. The production of high-grade bullion is mainly a question of thorough acid treatment, careful washing, and the selection of the proper fluxes in the fusion. The use of fire-clay liners in the plumbago pots is a strong factor in the production of high-grade bullion, whether nitre is used or not, as it guards against the reduction of zinc from the zinc sulphate or silicate present.

Tables of figures are given to show that the use of nitre is not attended with the baneful effects usually attributed to that flux. Equal quantities of precipitate, one fluxed with nitre and the other without, gave results favorable to the use of nitre. Further evidence is offered in tables showing a comparison between the gold called for by assays of the precipitate and the gold recovered in bullion, showing practically no loss of gold.

A comparison between smelting in plumbago pots and clay liners in plumbago pots gives results very favorable to the latter, showing a higher grade and fineness of bullion. The use of nitre, however, is found to be unnecessary when clay liners are used.

As regards roasting to a red heat versus drying of the precipitate, the authors believe that the latter is all that is necessary unless the acid treatment is poorly done, or in the case of coatings of base salts on the zinc. In the latter cases roasting is necessary or else treatment with nitre in clay pots. With regard to the elimination of the base metals, zinc, copper and lead, thorough acid treatment and washing, with reduction in clay pots, should eliminate the zinc; nitre flux and the use of clay liners should remove the copper, and similar treatment should take care of the lead, although manganese dioxide may be added to facilitate operations.

The following deductions are made as a result of the authors' work: No screening of short zinc is necessary, the cost of extra acid being more than compensated for by the reduction of refining and banking charges. Acid treatment should be as complete as possible. Washing in the press is by no means sufficient and leaves much to be desired. Roasting at red heat is unnecessary except as noted heretofore; drying is all that is necessary. Clay liners to plumbago pots are essential to the production of high-grade bullion unless all the zinc salts can be washed out of the precipitate. The use of thees liners gives slags of lower value; more quiet fusion; the absence of obnoxious fumes and lower bullion realization costs.

Losses in Cyanidation.—Under this caption Mr. H. FORBES JULIAN discusses the relative merits of filtering devices, and some errors in their operation, in the October, 1911, *Mining Magazine*. Leaving out of consideration the losses which occur before filtering commences, the author assumes the preparation of a slime pulp of a consistency ranging from 1½ tons to 3 tons of liquid for one ton of solid, although this may be thickened to a consistency of 1 : 1.

Cake Formation and Thickness.—When filtering slime containing much amorphous matter it is well to apply a low suction, about one-quarter of the maximum, until the cake is about ⅛ in. thick, and then apply the maximum. Operators as a rule pay too little attention to this variation in vacuum, but keep it constant throughout cake formation. The thickness to which a cake is allowed to build up is a matter that must be determined by experiment, for it may be only ½ in. in the case of a gelatinous slime, or 2 in. when crystalline material is filtered. The rate at which the filtrate is drawn may be utilized to determine the best thickness.

The porosity of a cake is affected by the pressure of formation. Thus a cake deposited by vacuum contains 40 to 65 lb. dry ore per cu. ft., according to the specific gravity and fineness of the material, while the same material filter-pressed in a cake contains 70 to 110 lb. dry ore per cu. ft. This explains why high-pressure filters give thin, sloppy cakes from which the valuable solution has not been fully pressed, when filtering amorphous material, whereas the vacuum filter working on the same material forms thick, porous dry cakes.

Characteristics of Filter Cakes.—Vacuum filters will produce a thicker cake from so-called natural slime, and one that is more uniform in its porosity; but on crystalline material a pressure filter will produce a heavier cake in given time, but not heavier in proportion to the power expended, and not uniform. The cake formed on a vacuum filter is more uniform, being made up of a number of layers, whereas in the pressure filter, segregation is unavoidable. This makes washing more uniform in the vacuum filter. Moreover, in washing vacuum filter cake a turbid wash solution can be used which will soon adjust the porosity of the cake if there is any tendency to uneven flow.

Care of the Cake.—During the time the cake is exposed to air it must be handled with extreme care. If too high a vacuum is held it has a tendency to disturb the grain of the cake and disturb its even porosity. If suction is cut off entirely there is danger of portions of the cake falling off the filter. Vibrations also must be reduced to a minimum, as these disturb the integrity of the cake.

Washing the Cake.—When the cake is ready for washing

it contains 30 per cent to 35 per cent of its weight of valuable liquor, together with undissolved metallic gold and silver particles and compounds. The liquor may be washed out by water, but if the latter values are to be recovered they have to be removed by a solution wash. The solid gold and silver remaining in the pulp may be still in its natural state, or it may have been reprecipitated during the agitation process. In any event these solid particles dissolve slowly, the latter more so than the former, and therefore the rate of solution washing must be slow enough to give the necessary time for contact between the solution and the particles.

It is well known that in the dissolving and agitating process, a point is reached beyond which the cyanide solution will not dissolve any more gold. This is due to reaching a point of equilibrium due to the presence of various salts in solution. If it were possible to bring fresh, potent solution in contact with the pulp at that time, further solution could take place. This is exactly what is possible in washing a filter cake with barren solution, but sufficient attention is not paid to it.

Reprecipitation and Resolution.—Reprecipitated metal does not dissolve with the same readiness that is noted in the natural state. Reprecipitation is due to local action of physical and chemical forces. Graphite, tellurides, sulphides, arsenides, charcoal, grease, oils, and even silica or alumina in a finely divided state, precipitate gold and silver from working cyanide solutions, and this precipitate shows little tendency to again dissolve in the same solution. In fact, it is difficultly soluble in fresh solutions. Hence in washing a cake with solution for the purpose of removing as much as possible of the undissolved or reprecipitated gold time should be allowed for contact. A cake ¼ in. thick requires about as much time to be solution-washed as does one 2 in. thick.

Preparation and Application of the Wash Solution.—If it is not already aerated it should be treated to give it maximum oxidizing power, and if not already turbid it should have some slime added. It should be made stronger in cyanide than that originally used in the treatment tanks, and some free alkali should be added. In applying this solution it is necessary only to give it such a rate of flow as would prevent backward diffusion, but in practice this time cannot be allowed. Nevertheless, the time allowed should be as great as possible, and not as absurdly low as in some instances.

Water Wash.—The object of the water wash is to remove cyanide solution remaining in the pulp from the solution wash. This would appear to be a simple matter, but in practice it is found that a great deal more water must be drawn through the cake than that indicated by a simple calculation. A low vacuum throughout the whole wash is desirable for a maximum strength of filtrate, but a high vacuum at the start would save time if maintained no longer than is necessary to just withdraw the stronger portion of the filtrate. But even then some cyanide will be held by absorption, requiring time of contact to diffuse into the water wash. The more porous the cake the lower should be the vacuum, and the finer and more amorphous the material the longer should be the time of contact.

Washing by Filtration and Decantation.—The simplest of all methods of washing is by decantation, but in practice there are serious objections to it. The unavoidable loss of dissolved metal remaining in the pulp is the greatest. In filtering, the amount of dissolved metal left in the cake has no relation to the original richness of the solution. In decanting, the metal contents are removed by two to four washes or dilutions, while in filtering this is done by an infinite number of washes. If the total quantity of liquor used in the decantation process be made the same as in the filter process, say 3 : 1, the theoretical separation of dissolved metal by three decantations would be only 78.4 per cent. In practice 75 per cent would be considered fair work. In the filtration process with this same proportion of liquor there is no difficulty in removing 99 per cent of dissolved metal. But in order to get 99 per cent of dissolved metal by the decantation process it would require a total of 12 tons of wash liquor to 1 ton of solid. Wash waters generally contain small quantities of dissolved metal, and the solution in the residue pulp generally contains this metal in proportion to the amount of moisture retained, which for filtration is about 1 of solution to 2 of solid, and for decantation 1 : 1. Hence this loss in decantation is twice what it is in filtration. Again the wide difference in the quantity of wash water used in the two processes affects the cost per ton of ore. In filtration the amount of wash liquor used seldom exceeds twice the weight of the solid, while in decantation it is common to use five of wash liquor to one of solid. Finally reprecipitation and absorption are greater in decantation than in filtration, as the very dilution with water in the decantation process is sufficient to cause silica and alumina to precipitate and carry down gold and silver and cyanide.

The Brakpan Mill, Transvaal.—One of the newer cyanide mills on the Rand is that known as the Brakpan, in which are installed such modern features as zinc dust precipitation, agitation of slime in Pachuca tanks, and filtration in Butters vacuum filters. A complete description is given in the *South African Mining Journal* for Aug. 12, 1911.

The ore is first separated into coarse and fine portions, sorted, and the coarse portion crushed. The battery bins have a capacity of 6,000 tons of fine ore. Stamps of 2,004 lb. weight are used; at present there are 160 installed and provision is made for 40 more. The stamp heads are 42 in. long, 9½ in. in diameter and weigh 788 lb. Stems weigh 696 lb., and are 14 ft. 6 in. long and 4½ in. diameter. Tappets weigh 232 lb., and a shoe weighs 288 lb., and is 14 in. long and 9¼ in. diameter. Ten stamps are driven by one motor.

The stamped ore is distributed to the tube mill cones, which are of the Caldecott diaphragm type. There are six tubes, 5 ft. 6 in. by 22 ft., and provision is made for the installation of four more. The feed contains 35 per cent moisture. Following each tube is a set of five stationary amalgamation plates, 12 by 5 ft., with a grade of 18 per cent. The amalgamated pulp is classified in cones, the overflow going to the cyanide plant, while the underflow is returned to two of the tubes for further grinding.

The cyanide work is divided into separate treatment for sand by percolation, and slime by agitation. The pulp delivered to the cyanide plant is, therefore, classified into sand and slime by means of cones. The sand is delivered to the collection tanks with 30 per cent moisture, but additional water is added, so that clean sand is delivered to the 60-ft. collectors. The overflow of the collectors is again cone-classified to remove the last traces of sand from the slime. From the collectors the sand is conveyed by belts to a Blaisdell distributor which serves the treatment tanks. After treatment the sand is removed by a Bleichert system of aerial haulage running from beneath the tanks to a cantilever tower where it is dumped. This system was installed with a guarantee to dump tailings at a cost not to exceed three cents per ton.

The slime overflow of the classifying cones passes to slime collectors where it is allowed to settle. It is treated in six Pachuca tanks, 15 ft. by 45 ft., where four hours agitation is given. Stock tanks are placed between the Pachucas and the Butters Vacuum filter, the latter consisting of two boxes and a total of 330 leaves. The filtered solution is clarified in two sand filters, used alternatively, and passes to gold solution storage tanks. Precipitation is made by zinc dust in Merrill presses. The zinc emulsion is made in emulsifying cones in which the zinc dust is agitated with weak solution before being added to the gold solution. The solution from sand and slime treatment, together with the zinc emulsion, is pumped to the Merrill presses by three plunger pumps driven by 10-hp motors.

Electric power is used throughout the plant, the aggregate power of motors installed being 10,000 hp for mine and mill. In the mill the distribution of motors is as follows: Stamp battery and tube mills, 1950 hp; reductions works pumping plant, 1000 hp. The power supply is delivered through underground cables, paper insulated and lead covered. These cables are laid

on a ring system, so that supply can be maintained to any point through an alternative part of the system in the event of a break in any particular cable.

Zinc.

Electric Zinc Smelting in Sweden and Norway.—For several years extensive experiments have been carried out on the use of the electric furnace in zinc metallurgy in Sweden and Norway, as has been described repeatedly in this journal, but details have been maintained secret with great care. However, now information regarding the process has become available for the first time, owing to the controllers of the works seeking English and German capital for the purpose of continuing the trials and enlarging the plant. According to the October issue of the *Mining Magazine*, of London, an English company has been formed for the purpose of acquiring two electric smelting works, one at Sarpsborg, Norway, and one at Trollhättan, Sweden, and also the rights in connection with the construction of another at Tysse, near Odde, in Norway. The process and furnace used at these works are intended for the treatment of complex sulphide ores and other material, such as Broken Hill slime and the intimately mixed sulphides of lead, zinc, copper and iron that occur in large quantities in various parts of the world, at present of little or no commercial value. Reports have been made by J. C. Moulden for the Sulphide Corporation and by F. W. Harbord, the latter of whom made thorough tests of the plants at Sarpsborg and Trollhättan. The information here given is called from the reports of these experts:

"The electric furnace employed has carbon electrodes and the heating and melting of the charge is effected by resistance with a current of low voltage. One electrode is in the bed and the upper one is removable through the roof. The cross-section of the upper electrode measures approximately 2¼ sq. ft. In the first furnace the charge was introduced through the roof, but more recently the door is placed in the side and the operation is thus made more continuous. The capacity is 3 tons, and 2.8 tons of roasted ore can be treated per 24 hours. The horsepower per furnace is 350, and at Trollhättan the cost of power supplied by the hydroelectric company is 30s. 3d. ($7.56) per horse-power year. At Mr. Harbord's tests on roasted Broken Hill slime the energy used per ton of ore was 2078 kw-hours. There are 11 furnaces in Trollhättan. The roasted ore, coke or anthracite dust and flux are charged into the furnace, and most of the zinc and some of the lead is volatilized. The greater proportion of this is condensed as a crude spelter with high lead content, but much of it is precipitated as metallic fume and oxide, and has to be returned to the furnace in subsequent charges. The lead that is not volatilized is reduced as liquid metal, and it contains a considerable proportion of the silver. Matte is formed of the iron, copper and sulphur, if present in the ore, and some of the lead, zinc and silver passes into it. Some lead also escapes with the slag. The crude zinc is refined in other electric furnaces and the final zinc product is exceptionally pure, averaging 99.9 per cent and commanding a premium price in the market."

"At the time Mr. Harbord made his examination the furnaces were built of porous material, apparently similar to that used in old-fashioned zinc distilling retorts, and some of the lead and silver percolated through and had to be caught underneath; this he calls 'leakage' lead. He found also that the lead that exuded in this way during his experiments did not necessarily come from the charges under his observation and was partly produced during the treatment of other ore at an earlier period. Thus the silver contents of the lead did not at all coincide with that of the ore treated by him. The general results of his trials were by no means ideal, and he makes many suggestions for improvements; but his conclusions are that the process may easily be developed into one of prime importance, especially at the new plant proposed at Tysse, where the estimated cost of a horse-power-year is only 14s. ($3.50).

"During his experiments at Trollhätten seven furnaces treated during 27½ days, 518 tons of roasted Broken Hill slime, 19 tons of calamine, 22½ tons of zinc-lead powder produced previously, and produced 160.8 tons of crude zinc which by refining yielded 112.4 tons of pure zinc and 24.7 tons of lead, together with 36 tons of powder, in addition to which 41 tons of lead were tapped containing 141 oz. silver per ton and 'leak' lead amounting to 17 tons was recovered assaying 27 oz. silver per ton; also 9 tons of skimmings containing zinc, lead and silver. The recovery from these various sources was 64 per cent of the zinc, 74 per cent of the lead and 40 per cent of the silver, not reckoning the metals recovered in the powder form. Mr. Harbord goes into details as to many points where increased extraction and lower costs may be obtained."

"Mr. Moulden is of opinion that, by means of improvements already indicated, the extraction should eventually be 75 per cent of the zinc, 80 per cent of the lead and 80 per cent of the silver, and on this basis he figures the profit obtainable on slime bought from the Sulphide Corporation and shipped to Sweden. The slime is taken to average 32 per cent zinc, 23 per cent lead and 25 oz. silver, and on the current basis of payment would cost £2 16s. 8d. per ton at Broken Hill. The freight to Trollhättan would be £2 5s. 3d., making the total cost per ton at the smelter £5 8s. 5d. The value of the recovered metals on the percentage figured by Mr. Moulden would be £9 9s. 11d. The cost of treatment he estimates at £3 os. 7d., so that the margin of profit would be £1 9s. ($7.25) per ton of slime. It is unnecessary to say that the process is in its infancy and that already many improvements are obvious; also that everything depends on the cost of electric heat. Enough is shown, however, in these reports to make it clear that the process and method of metallurgy deserve close attention."

The subject is discussed by the *Mining Magazine* in an editorial which gives figures on the capitalization of the companies and contains the remark that "the present cost of electrodes is staggering, amounting to £204 ($1020) during the treatment of the 550 tons of material" in Mr. Harbord's tests. Attention is drawn to the fact that a great many other electrometallurgists are experimenting on the same lines and that "theoretically their furnaces both in principle and in method of construction are practically identical with those at Trollhättan, so the relative skill of manipulation, attention to detail, and cost of power will decide the actual success of any of these rival processes." It is further pointed out that "the practice in zinc metallurgy in retorts had been so much improved during the last year or two that the lead content is no longer a drawback; in fact, the buyers desire to have at least 5 per cent lead, for this is found useful in collecting the silver, and the leady residue left in the retorts is sent to the lead furnace." It had recently been said that the slime from Broken Hill, if concentrated by flotation and judiciously mixed with the zinc product, is profitably marketable. It appears, however, that, in negotiations with German and Belgian firms, all sorts of difficulties are raised, and when the sale of concentrated slime without admixture with zinc material is proposed the terms and conditions become impossible. Thus the Broken Hill producers are desirous of trying some alternative process for the purpose of realizing the great accumulations of slime. The Sulphide Corporation is the only Australian producer that is selling slime concentrate to the newly formed firm, operating in Sweden and Norway, the others apparently waiting for the Imbert process (see, for instance, our May issue, p. 259) to become established. This is under option in London to a Broken Hill group that had previously backed the Swedish-Norwegian venture.

Chemical Engineers.

Firing Boilers with Producer-Gas.—*Stahl und Eisen* of Oct. 26, 1911, p. 1771, contains some interesting tests on firing boilers with producer gas, made by the Compagnie des Mines d' Anthrazite de la Mure (Isère) a la Motte d'Aveillans (France). The boiler house contains three boilers with a total heating surface of 4521 sq. ft. Two of these boilers were

double cylindrical with fire tubes of 1615 sq. ft. each and the other was a double cylindrical boiler with water tubes above and of 1202 sq. ft. One fire-tube and one water-tube boiler were used for the test with a total heating surface of 2907 sq. ft. and heated by producer-gas which was generated in a Kerpely producer of 6.9 ft. inside diameter. Fuel was a slaty anthracite, obtained as a waste product from the mines, the size of the coal ranging from 4/5 in. to 2 2 5 in. Two coal analyses may be given:

	I.	II.
H_2O	3.31	2.9
Ash	23.35	22.7
Carbon	71.76	69.4
Vol. Matter	1.58	5.0

Ash analysis:

SiO_2	76.54 per cent
Fe_2O_3	8.84 per cent
Al_2O_3	4.30 per cent
CaO	4.42 per cent
Alcalis	5.90 per cent

Both boilers were filled up to a mark on the gauge glass when started, the test lasted 10 hours and the boilers were then filled up again to the same level. The feed-water temperature was 151 deg. F., and had been used before as cooling water in the water-jacket of the producer. The water was measured in a tank, the steam pressure was kept constant at 80.9 lb. per square inch. The arrangement had already worked for 10 days before the test without any trouble. The total evaporation during the ten hours of the test was 73,193 lb. of water, the coal consumption 11,850 lb., therefore evaporation 6.18 lb. H_2O per pound coal for feed water temperature of 151 deg. F. or 5.5 for water of 32 deg. F. This was very satisfactory since by direct firing this coal evaporated only 2.5 lb. of water per pound of coal. This was due to incomplete combustion on grates, due to its high content of slate. To gasify this coal is advantageous because it permits making use of this coal, of which only the better grades could be sold at all; moreover, the expenses for boiler firing are thereby considerably reduced. One man for a battery of boilers is enough to regulate the gas and air inlet at the burners. Having automatic coal feeders and ash removers, one man can handle two producers with a capacity of 88,184 lb. coal per day. It may be remarked that during the first four hours of the test the producer ran on cool gas with 4 5 per cent CO_2 (Period I). During the following six hours the producer was run somewhat hotter (Period II), causing a poorer gas with 7-8.5 per cent CO_2, but despite this fact the evaporation was higher than before. In the latter case the gas contained more sensible heat. From previous tests it is estimated that the temperature varied between 302 and 482 deg. F. (200 and 250 deg. C.).

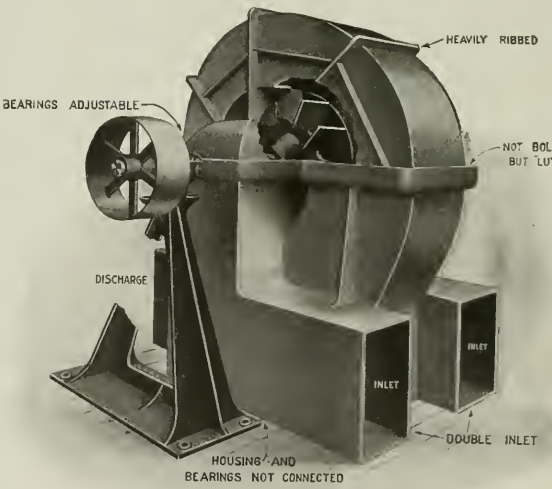
During the first period of 4 hours 26,455 lb. water were evaporated, by 4060 lb. coal, which equals 5.3 lb. H_2O per pound coal; during the second period of 6 hours 46,738 lb. water by 6834 lb. coal, or 6.8 lb. water per pound coal. In other tests similar results were obtained on a slaty hard coal with 35-40 per cent ash, giving an evaporation of 6.5, the producer being run like Period II. In this case the gas main should be as short as possible to reduce radiation losses. For illustration, another example may be cited where a Kerpely producer was directly connected with a heating furnace, the coal was from the Saar district (Germany). In the beginning the producer was operated in the normal way, but since a few years the producer has been run on hotter gas, (but with more CO_2) and it was found that the furnace works better and with a lower coal consumption per ton of production.

Hours.	Boiler pressure in pounds per sq. in.	No. of bags containing 110.2 lb. coal each.	Pounds coal consumption.	No. of tank fillings 881.8 lb. each.	Water evaporated in pounds per hour.	Feed water temperature, degrees Fahr.	Remarks.
7-8	80.9	12	1322.8	7	6173.	149	Normal gas temp.
8-9	"	12	1322.8	7	6173.	149	"
9-10	"	12	1322.8	8	7054.7	151	"
10-11	"	9	992.1	8	7054.7	156	"
11-12	"	11	1212.5	9	7936.6	154	Running hotter.
12-1	"	11	1212.5	9	7936.6	147	"
1-2	"	9	992.1	9	7936.6	150	"
2-3	"	10	1102.3	8	7054.7	149	"
3-4	"	11	1212.5	9	7936.6	153	"
4-5	"	10	1102.3	9	7936.6	153	"
	80.9	107	11,794.7	83	73,193	151	

Induced Draft for Sulphuric Acid Chambers.

The advantages of fans for producing induced draft in the manufacture of sulphuric acid by the chamber process have long been known to acid manufacturers, but more or less difficulty has been experienced in designing and manufacturing a fan that would meet the requirements.

The principal advantages to be derived from induced draft are:



ACID FAN.

(1) Greater production per cubic foot of chamber volume

(2) Reduced cost of nitre per pound of acid so produced.

(3) Concentration of a greater amount of chamber stock to acid of 60 deg. Baumé in the Glover tower.

(4) Better absorption of nitre in the Gay Lussac towers on account of the proper excess of oxygen in the chamber process.

(5) Even draft on the sulphur burners.

(6) Successful use of intermediate towers, which are not successfully used where natural draft is used, except in rare instances.

Poor design and construction of fans have been one of the drawbacks to their successful use in induced draft. It would seem to be a simple matter to use a fan for this purpose, but on account of the limiting conditions there have been many

failures. The only material of which a fan for this purpose can be constructed is lead, for the gases are hot and corrosive; but, on the other hand, the great weight of lead and its tendency to distortion have been the cause of failure.

Various designs have been suggested. Some make use of a rigid metal such as iron, covered with lead, but in these the lead has a tendency to become separated from the iron and render the fan inoperative.

More designs have used overhung blast wheels where, on account of the necessary thickness and weight of the lead, the wheel has to be balanced perfectly and the journal boxes adjusted to a fine point. If these conditions are not observed, the blast wheel, projecting beyond the bearings into the fan housing, would revolve eccentrically; the great weight would bend the shaft, and the blades striking the housing would be broken; or the metal in the blades would become crystallized from vibrations and break. Considering the speed at which these fans must be operated, the possibility of breaking the blades becomes a source of danger as well as trouble and expense. Fans of this type also become unbalanced due to corrosion.

In order to overcome these difficulties and objections in acid fans, a design shown in the accompanying illustration has been made by Mr. N. L. Heinz and Mr. L. B. Skinner, who are practically engaged in acid manufacture. The machine is made and marketed by the Stearns-Roger Manufacturing Company, Denver, Col. It probably is more expensive to manufacture than other designs, but, when its service is considered, the increased cost is more than counterbalanced by continuous operation without constant attention and frequent repairs.

The lead alloy of which the fan and housing are made is of the proper mixture to resist acid corrosion, and the tensile strength of the metal is not impaired by improper handling. The blades are of the right number and shape to give the highest efficiency, fitting closely to the housing. The latter is heavily ribbed to give stability, and avoid the necessity of constant adjustment due to warping. The bearings of the fan are separate from the lead housing, thus reducing the possibility of tearing the latter by vibration. Accessibility is provided by luting the cover of the housing to the base, instead of bolting the two parts together.

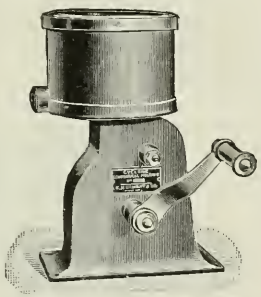


FIG. 1.—HAND-DRIVEN 5-IN. CENTRIFUGAL.

The general construction of this machine is such as to make the high rate of speed of the fan absolutely safe as far as possibility of accident is concerned. It operates at 475 r.p.m., and will furnish the gas necessary for the operation of a chamber set of 300,000 cu. ft. with intermediate towers. The approximate shipping weight of the machine is 7500 lb.

Centrifugals for Laboratory Work.

One of the quickest and most efficient methods to remove crystals and precipitates from mother-liquors and solutions is undoubtedly to use centrifugal force.

This fact is amply demonstrated by the enormous amount of large centrifugals in use in many industries. In the laboratories of chemical and pharmaceutical factories, sugar works, dye-houses, technical schools and metallurgical industries, the only difficulty which has heretofore prevented the use of smaller centrifugals has been their relatively high price and bulkiness as they were generally the reduced models of the big machines.

In the accompanying illustrations, however, is shown a centrifugal which, on account of its compactness and extreme simplicity, can be produced at a much lower cost than has heretofore been possible.

The Chavant "cyclone" laboratory centrifugal has been on the market for many years. It was formerly made in Jersey

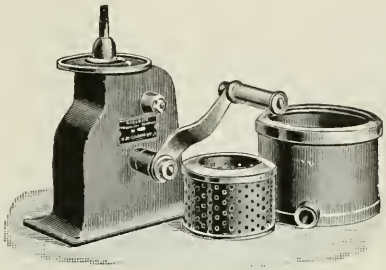


FIG. 2.—BASKET AND CASING REMOVED.

City, N. J., but the works were recently moved to Fontenay-sous-Bois, France, where a larger plant and better manufacturing facilities were to be had with the enormous European field near at hand.

The design and construction of the "cyclone" centrifugal have been improved in many respects. The new model, known as

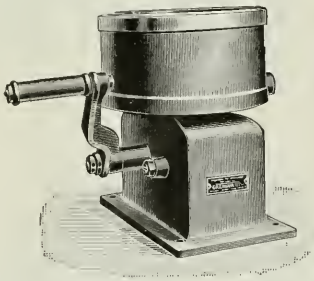


FIG. 3.—HAND-DRIVEN 8-IN. CENTRIFUGAL.

Type "F," is made in three sizes: 4½, 5 and 8 in., and is furnished with a crank for hand-drive or a pulley for power-drive. The machine, mounted on ball-bearings, is exceptionally light-running and can be operated at high speed without any perceptible vibration.

Fig. 1 shows type F, 4½ and 5-in., hand-driven. In Fig. 2

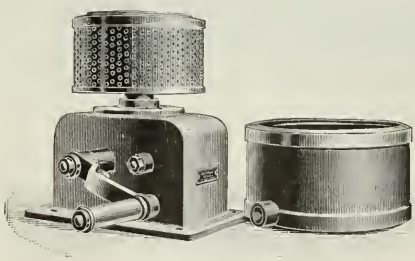


FIG. 4.—CASING REMOVED.

basket and casing are removed. Fig. 3 shows type B, 8-in., hand-driven, while Fig. 4 shows the same with casing removed.

There are no pulleys or belts in the internal mechanism, which consists of machine-cut gears, made with the greatest accuracy by special machinery in order to insure long wear and easy running.

The gears, lubrified by outside oilers, are encased in the base casting fully protected from dust, but can be easily reached for cleaning as no part has to be unscrewed.

The basket and also its casing are detached by simply lifting them off the machine and they can be replaced instantly.

The basket is made of bronze and brass and is perfectly balanced. For acid products it is tinned and its casing lined with porcelain. A hard rubber basket can be supplied too.

For all "cyclone" centrifugals the manufacturer guarantees that with a maximum of 70 revolutions of the crank or of the pulley per minute they will give all the results which shall be expected from a first-class laboratory centrifugal.

The general sales agent of the company is Mr. Victor J. Boulin, with offices at 123 Liberty Street, New York City.

Repair of a Large Rock-Crusher Shaft.

A repair of exceptional size was recently executed for the North River Stone Company, of Kingston, N. Y., on one of its rock-crusher shafts. The shaft in question was 18 ft. long and ranged from 12 in. to 22 in. in diameter. It was 14 in. in diameter at the point of fracture.

It was decided to make the repair by the thermit process, but owing to the fact that there were not adequate facilities at Kingston for aligning the pieces accurately, they were sent to the Jersey City shops of the Goldschmidt Thermit Company, where they could be placed in V blocks, mounted on a perfectly level bed plate and so lined up accurately, previous to welding them together.

There is no necessity to describe here again the well known thermit reaction of Dr. Hans Goldschmidt between aluminium and iron oxide, yielding a slag of alumina and as chief product a molten highly superheated low-carbon steel, available for welding. By mixing steel punchings with the thermit the carbon content of the thermit steel is increased, and the quality of the steel is also improved by the addition of other materials such as manganese, nickel or chromium.

The temperature attained by the reaction is approximately

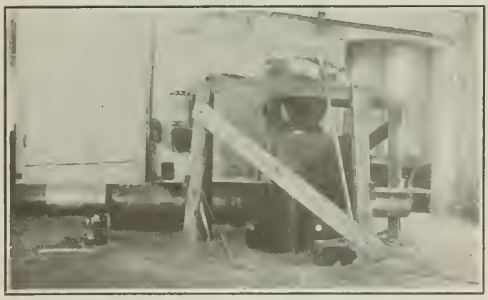


FIG. 1.—PREHEATING SHAFT PREVIOUS TO WELDING BOX AND CRUCIBLE IN POSITION.

5400 deg. Fahr. and it will readily be seen that by pouring this superheated steel into a mold surrounding the ends of the sections to be welded together it will dissolve the metal with which it comes in contact and amalgamate with it to form a single homogeneous mass when cool.

In welding the large shaft for the North River Stone Company a space of 2½ in. was provided between the ends of the sections to be united by cutting away the broken faces to this extent, after which a pattern of yellow wax was formed about the sections in the shape of a collar 14 in. wide and 2 in. thick at the middle point. In cross-section this collar resembled in shape a segment of a circle. A sheet-iron mold box was then placed in position and rammed with special molding material, consisting of a mixture of one-third fire clay, one-third ground fire brick and one-third fire sand. Ordinary molding material would not be sufficiently refractory to withstand the high temperature of the thermit steel.

Wooden patterns were used for two pouring gates, a riser directly over the top of the weld, and a small preheating opening at the bottom. The flame of a powerful gasoline compressed air torch was then directed into the preheating hole and melted out the wax, thus leaving the mold ready for pouring.

The preheating, however, was continued for a period of ten

FIG. 2.—VIEW AFTER WELD IS FINISHED, BUT BEFORE CUTTING OFF METAL LEFT IN POURING GATES AND RISERS.

hours so as to thoroughly bake the mold and bring the sections to be welded together to a bright red heat, at which time the torch was withdrawn, the preheating hole closed up and the thermit charge, held in two crucibles suspended over the pouring gates of the mold, ignited.

The thermit charge for this repair consisted of 1100 lb. of thermit, 275 lb. of steel punchings, 11 lb. pure carbonless manganese and 11 lb. of chromium thermit. These were all thoroughly mixed with the thermit in the crucibles.

The thermit reaction required about 35 seconds for completion, but the charge was held in the crucibles a few seconds longer so as to allow time for the slag to thoroughly separate from the steel. The slag, being lighter than the steel, floats to the top of the crucibles, while the steel sinks to the bottom. The crucibles are tapped from the bottom, so that the steel flows into the mold first, being led to the lowest point of the mold, then up between and around the sections to be welded into the riser. The slag does not enter the mold at all, but merely overflows.

It is important that the metal in the weld be allowed to cool slowly and in the case of this 14-ft. shaft the mold was left undisturbed for a period of 45 hours, after which it was broken up and the steel left in the riser and pouring gates removed by means of a cutting flame of oxygen and illuminating gas.

The entire repair was carried out in less than a week, including the time required for towing the shaft on a barge from Kingston to New York and back again. This alone took over two days.

The thermit steel collar or reinforcement was not machined off, as it did not interfere with the operation of the shaft and added considerably to the strength of the repair.

If the thermit process had not been available for this work, the owners would have been forced to wait several weeks for a new shaft, while the cost of a new shaft alone would have amounted to several times the cost of the weld.

Mr. Paul O. Abbé announces the opening of an office at room 1201, Johnston Building, 30 Broad Street, New York City, as a builder of jar and pebble mills in all styles and sizes and as agent for the sale of mixers, sifters, vacuum pumps, pressure blowers, and bolting cloth of various makes.

Frictionless Ink-Type Recording Instrument.

An illustrated description of Bristol's patented semi-transparent smoked chart recorder was printed in our issue of January, 1908 (Vol. VI, p. 35). Although these recorders are fundamentally simple in their construction and hundreds are in daily service, there has been a call for a recorder in which the record is made with ink. To meet the demand for a frictionless ink-type recording instrument which would parallel

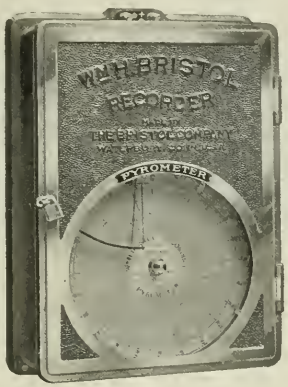


FIG. 1.—OUTSIDE VIEW OF RECORDER.

the smoked-chart recorder as nearly as possible in intrinsic simplicity, and accurately record fractions of millivolts and be adapted for use as a recording electric pyrometer, the instrument illustrated herewith has been developed and placed upon the market by the Bristol Company, of Waterbury, Conn.

These instruments have been thoroughly tested out in practical service for two years past, and are the result of several years of study and experience with an original design patented

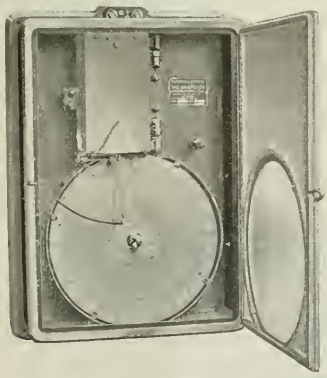


FIG. 2.—INTERIOR VIEW.

by Prof. Wm. H. Bristol on April 13, 1909, of a frictionless ink recorder using a hinged electrical movement carrying a retaining receptacle for marking fluid which extends over the path of the recording tip, and is provided with means for periodically making contact with the source of marking fluid and the chart.

Fig. 1 shows the recorder ready for operating.

Fig. 2 is an interior view showing the galvanometer movement case hinged to the back of the instrument and carrying the inking pad in front of the recording arm.

Fig. 3 shows the sensitive electrical movement swung to one side for convenience in removing the record and inserting a fresh chart. The capillary gold tube open at both ends is carried at the end of the recording arm at right angles to the surface of the chart. The inking pad is suspended from the case of the electrical movement and is curved to correspond with the arc covered by the motion of the end of the recording arm.

When the movement is swung back into its operating position, as shown in Fig. 2, the recording arm can swing free, accommodating itself to the position corresponding to the delicate current which is to be measured. The clock which revolves the chart at the desired speed also automatically presses the inking pad toward the chart every 10 seconds, bring-

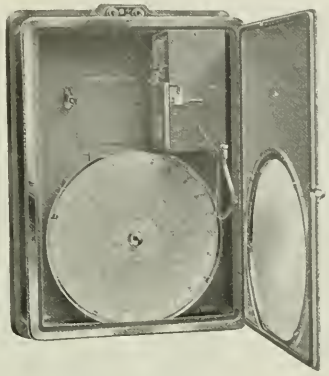


FIG. 3.—INTERIOR VIEW.

ing one end of the capillary tube into contact with the chart and the opposite end simultaneously into contact with the inking pad. A fine dot of ink is left on the chart and the capillary tube is replenished with ink from the pad. The recording arm thus carries a constant supply of ink, and its perfect balance, which is very important, is always maintained. The electrical movements used in these recorders are made especially for this purpose by the Weston Electrical Instrument Company.

Although the most important applications of these recording instruments have been for pyrometers, they have also been used for electrolytic research, recording voltmeters and recording shunt ammeters.

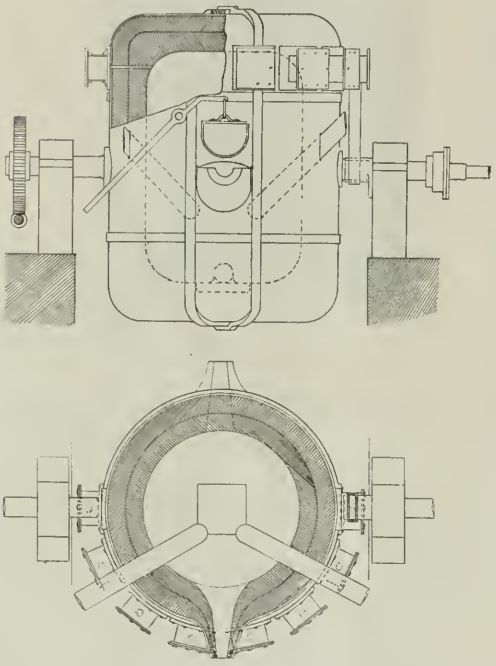
In the determination of sulphur in ores difficulty is frequently encountered in getting all the sulphur in solution before it separates out and collects in a globule of elementary sulphur. When this condition occurs it necessitates longer treatment to dissolve the sulphur. In order to overcome this obstacle, E. H. Westling proceeds as follows: The solvent used is nitric acid of 1.35 sp. gr. with a little dissolved potassium chlorate, at a temperature of about 60°C. The boiling point of this mixture is below the melting point of sulphur, and if the ore pulp is added gradually to the solution sulphur will not separate out but will be dissolved without difficulty. Rapid and complete decomposition is claimed for this procedure.

The Blum Zinc Oxide Process, which is protected by U. S. patent 993,019, consists in heating a charge made up of alternate layers of fuel and mixed zinc ore and fuel. It has been customary to carry out this process on a perforated grate, after the manner of the Wetherill process. Mr. Blum, however, proposes to use a pot like those used in the Huntington-Heberlein roasting system. The layers of alternate ore and fuel are 10 cm thick. The fuel is anthracite fine, both for mixing with the ore layer and for the fuel layer. The advantage claimed is large capacity as compared with Wetherill grates.

Electro-Bessemer Furnace.

In this issue, in the department of Recent Metallurgical and Chemical Patents, an interesting patent of Messrs. Walker and Gray, of the U. S. Steel Corporation, is mentioned, describing a steel refining furnace, which is first operated as an open-hearth and then as an electric furnace, thus obviating the necessity of transfer of the molten metal from one furnace to another.

The analogous idea for Bessemer converter practice is incorporated in the Electro-Bessemer furnace, designed by Messrs. Verdon Cutts & Hoult, 43 Edge Hill Road, Sheffield, England. This furnace was recently described in a paper by Mr. Verdon O. Cutts before the Staffordshire Iron and Steel Institute. After reviewing the different types of electric furnaces in general and those for iron ore reduction and steel refining in particular, Mr. Cutts described the Electro-Bessemer furnace as follows:



ELECTRO-BESSEMER FURNACE.

"The primary object which we had in view in designing this furnace was the reduction of working costs. We recognized the fact that the cost of electrical energy very considerably affects the cost of steel manufacture in the electric furnace. Our experience in important steel works both in this country and on the Continent has revealed a very high cost of generating electrical energy, and, in my opinion, this has proved sufficient at least to delay the adoption of the electric furnace in many steel works. It was, therefore, our object to produce a furnace in which electrical energy is only used where it offers unique advantages over other methods. A careful analysis of the subject shows that the value of the electrical process lies not so much in the making of the steel as in the finishing of the same. This has also been recognized by Héroult, Girod, and others, and accounts for their furnaces being now largely employed for the treatment of steel made in the ordinary Bessemer converter or open-hearth furnace. By our process the steel is made by the usual Bessemer method and finished electrically without it being necessary to pour the molten metal from one vessel to another during the refining operation. The loss of heat and expenditure of labor which are inseparably connected with the duplex process above referred to are thus avoided in the Electro-Bessemer furnace, resulting in a comparatively very high efficiency.

"Further, the Electro-Bessemer furnace is less dependent upon skilled manipulation than is the alternative combination of separate Bessemer and electric furnaces. . . .

"The many complicated designs which have been originated during the past few years have proved conclusively that the safest and shortest way to practical success is the utmost simplicity in construction, and the incorporation of old, well-known and successful methods with new processes. This we have kept well in mind, and the Electro-Bessemer process, as already suggested, combines in principle the ordinary Bessemer furnace and the electrical refining process. The result is that steel is produced cheaply and quickly as in the Bessemer furnace, and the disadvantage of lack of uniformity in steel so produced is entirely overcome by applying the electrical process of final refining.

"The Electro-Bessemer furnace is a closed vessel, having an opening in one side, through which the charge may be introduced and slags added and withdrawn. It is mounted on bearings, and is capable of being tilted within an angle of rather more than 180 degrees. One end of the furnace is fitted with blast tuyeres for blowing. At the other end of the vessel electrodes are arranged, entering through the furnace wall and being adjustable by any of the ordinary independent methods, either electrical or mechanical. A tapping hole is provided at this end of the furnace, which may be used for the removal of the finished product, if desired. Samplings may be taken from the charging opening, which opening might also be used, if desired, for pouring molten metal. The electrodes are fitted with cooling jackets of cold air or water.

"The process is simple in the extreme, being briefly as follows. The furnace is preheated in the ordinary way, and is then charged with molten metal direct from the cupola or blast-furnace mixer, the furnace being tilted into a horizontal position. It is then brought into a vertical position, the "Bessemer" portion containing the charge. The blast comes into action automatically as this position is gained, and the process of conversion proceeds as in the ordinary Bessemer converter. Meantime the "electric" portion of the furnace is heated by the hot gases, etc., resulting from the process, and for the ultimate removal of which gases ample provision is made. On the completion of this part of the process the furnace is tilted and the metal transferred to that end which is provided with electrodes. Current is switched on, and the refining operation continued to any desired extent. Both arc and resistance heating are obtained in the furnace.

"The Electro-Bessemer furnace may be either basic or acid lined; if it is basic lined it is possible to start with impure basic Bessemer pig-iron and to produce finished steel or better analysis than the best acid open-hearth steel. If the furnace has an acid lining, chemical refining of the blown metal does not take place, but the killing of the metal by heating in the 'electric' end of the furnace results in the production of particularly sound castings and ingots free from blow-holes.

"The utilization of the flame from the Bessemer blowing for the heating of the 'electric' hearth results in a considerable saving of electrical energy during the second stage.

"Having poured away the slag containing the bulk of the impurities at the end of the Bessemer blow (assuming a basic lining), new, clean, purifying slags can be added.

"Assuming that the phosphorus content is already low when the electrical operation commences, the energy consumption is less than 100 kw. hours per ton of steel produced. If, however, both desulphurization and dephosphorization are included, about 190 kw. hours are consumed."

Automatic Controllers for Motor-Driven Machinery.

The Electric Controller & Manufacturing Company, of Cleveland, O., has recently placed on the market a line of automatic controllers designed for the specific purpose of giving the utmost convenience in the control of motor-driven machinery.

It has been estimated that the output of many machines can

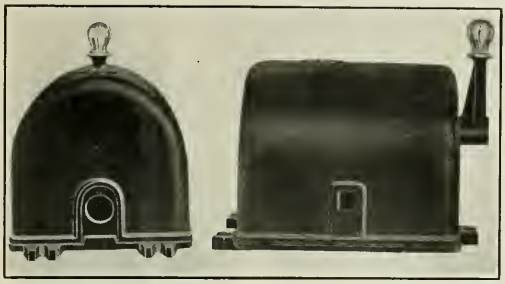


FIG. 1.—FRONT VIEW AND SIDE VIEW OF OPERATOR'S SWITCH.

be increased 20 per cent by the central grouping and convenient arrangement of all the operating levers. "Handiness of control" is recognized as being very important in securing the utmost production from a machine, and the automatic controllers described in this article were designed to provide this "handiness of control" for starting, stopping or reversing the motor and machine.

FIG. 2.—ACCELERATING UNIT.

The controller consists of a small operator's switch, shown in Fig. 1, and an accelerating unit which automatically accelerates the motor during starting and retards it during stopping. This accelerating unit is built in four different types, two of which are shown in Figs. 2 and 3.

The controllers are built in three types to secure: First, non-reversing and electric braking; second, reversing without electric braking; third, reversing and electric braking. In each type a variety of four different forms of accelerating units are

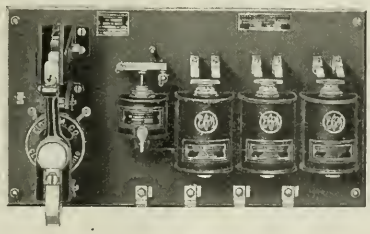


FIG. 3.—ACCELERATING UNIT.

offered. These accelerating units vary in their design from a simple train of accelerating switches up to a unit having a fused service switch, a train of accelerating switches, and complete circuit-breaker features.

The accelerating unit automatically accelerates or retards the motor through the action of series-wound accelerating switches which possess the remarkable characteristic of acting, not only as switches, but as current limit relays as well. When the current in the winding of one of these switches exceeds a predetermined value the switch locks open and cannot close until the current is reduced to the proper value.

When the operator's switch is thrown to the running position, current flows through the motor, through all of the starting resistance, and through the coil of the first series wound accelerating switch. As the motor accelerates the current drops, and when it reaches the correct value the first accelerating switch closes, cutting out a portion of the starting resistance. The succeeding accelerating switches operate similarly, ultimately cutting out all of the starting resistance and putting the motor across the line. By throwing the operator's switch to its original position, the motor is disconnected from the line and consequently stops. Different positions of the handle of

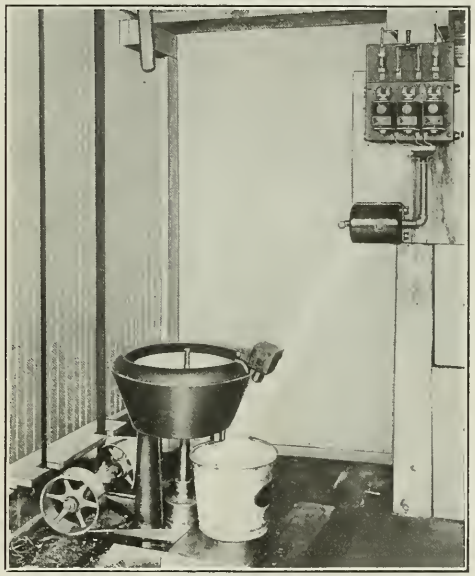


FIG. 4.—MOTOR-DRIVEN CENTRIFUGAL OIL EXTRACTOR EQUIPPED WITH AUTOMATIC CONTROLLER.

the different types of operator's switch provide for drifting, reversing, or rapid stopping by dynamic braking.

Electric braking is secured by a change of connections, accomplished by the operator's switch, which first inserts all the starting resistance in series with the armature. The motor is then quickly and evenly brought to rest by automatic electric braking, the accelerating switches, in this case, acting as retarding switches by cutting out, step by step, the resistance as the current, generated by the motor, decreases due to the slowing down of the motor.

The manufacturers claim the following important advantages for this automatic controller:

It limits the acceleration and retardation current at all times to a safe value.

It accelerates and retards the motor in the minimum safe amount of time, and automatically varies the time of acceleration and retardation, depending upon the load which the motor has to start and stop.

It provides the best conditions for good commutation.

By limiting the current, both in starting and stopping, it limits all mechanical strains on the motor and driven machinery.

It obviates the necessity of mechanical clutches on many motor-driven machine tools.

It adds very materially to the safety of an installation, since in case of accident the motor may be quickly stopped.

It inherently provides no-voltage protection, for if the voltage fails the switches drop open, and upon the return of volt-

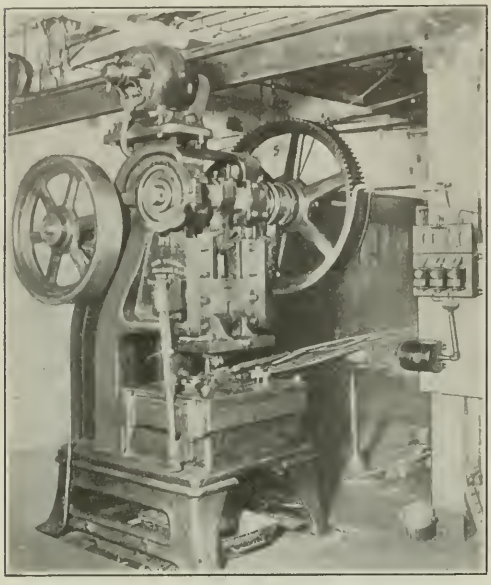


FIG. 5.—MOTOR DRIVEN PUNCH PRESS EQUIPPED WITH A TYPE H. N. B. FORM A. K. AUTOMATIC CONTROLLER.

age they automatically close in their regular method and sequence, again accelerating the motor to full speed.

Fig. 4 shows a motor-driven centrifugal oil extractor equipped with a type H N B, from A K automatic controller, Fig. 5 a motor-driven punch press equipped with the same type of controller. In both illustrations the automatic controller can be clearly seen on the right of the picture, the operator's switch being placed right below the accelerating units.

Census Report on Sulphuric, Nitric and Mixed Acids.

A preliminary statement of the general results of the thirteenth census of establishments engaged in the manufacture of sulphuric, nitric and mixed acids has just been issued by Director Durand, of the Bureau of the Census in the Department of Commerce and Labor. It was prepared under the direction of William M. Steuart, chief statistician for manufactures, Bureau of the Census. The figures are subject to such revision as may be necessary after a further examination of the original reports.

There were 42 establishments engaged in this industry in 1909 and 32 in 1904, an increase of 31 per cent.

The capital invested as reported in 1909 was $18,726,000, a gain of $5,964,000, or 47 per cent, over $12,762,000 in 1904. The average capital per establishment was approximately $446,000 in 1909 and $399,000 in 1904

The value of products was $9,884,000 in 1909 and $9,053,000 in 1904, an increase of $831,000, or 9 per cent. The average per establishment was approximately $235,000 in 1909 and $283,000 in 1904, a decrease of 6 per cent.

The cost of materials used was $5,386,000 in 1909, as against $4,973,000 in 1904, an increase of $413,000, or 8 per cent. In addition to the component materials which enter into the products of the establishment for the census year there are included fuel, rent of power and heat, and mill supplies.

The value added by manufacture was $4,498,000 in 1909 and $4,080,000 in 1904, an increase of $418,000, or 10 per cent. This item formed 46 per cent. of the total value of products in 1909 and 45 per cent. in 1904. The value added by manufacture represents the difference between the cost of materials used and the value of products after the manufacturing processes have been expended upon them. It is the best measure of the relative importance of industries

The miscellaneous expenses amounted to $916,000 in 1909 and $713,000 in 1904, an increase of $203,000, or 28 per cent. Miscellaneous expenses include rent of factory or works, taxes and amount paid for contract work, as well as such office and other expenses as cannot be elsewhere classified.

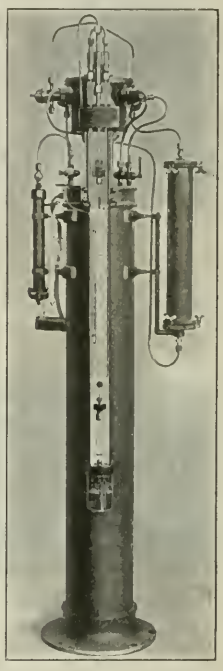


FIG. 1.—CONTINUOUS CARBON DIOXIDE AND TEMPERATURE RECORDER.

The salaries and wages amounted to $2,046,000 in 1909 and $2,062,000 in 1904, a decrease of $16,000, or 1 per cent.

The number of salaried officials and clerks was 330 in 1909 and 308 in 1904, an increase of 7 per cent; their salaries decreased from $556,000 in 1904 to $551,000 in 1909, or 1 per cent. The average yearly salary per official or clerk, therefore, decreased from $1,805 in 1904 to $1,670 in 1909.

The average number of wage earners employed during the year was 2252 in 1909 and 2447 in 1904, a decrease of 8 per cent; their wages decreased from $1,506,000 in 1904 to $1,495,000 in 1909, or 1 per cent. The average yearly wages per man, therefore, increased from $615 in 1904 to $664 in 1909.

The primary horse-power was 6494 in 1909 and 5416 in 1904, an increase of 20 per cent.

The average horse-power per establishment, considering all establishments, was approximately 155 horse-power in 1909 and 169 in 1904.

The total production of sulphuric acid, reduced to 50 deg. Baume, was 2,797,723 tons in 1909 and 1,869,446 in 1904, an increase of 50 per cent. Of this total 933,848 tons were reported in 1909 and 467,614 in 1904 by establishments making it as a chief product, an increase of 100 per cent.; 621,801 tons in 1909 and 433,377 in 1904 by establishments engaged primarily in the manufacture of other products, and 1,242,074 tons in 1909 and 968,455 in 1904 were made and consumed in establishments where manufactured.

The total production of nitric acid was 138,086,565 pounds in 1909 and 108,380,387 in 1904, an increase of 27 per cent. Of this total 8,396,326 pounds were reported in 1909 and 30,306,555 in 1904 by establishments making it as a chief product, a decrease of 72 per cent.; 18,929,620 pounds in 1909 and 15,927,526 in 1904 by establishment engaged primarily in the manufacture of other products; and 110,760,619 pounds in 1909 and 62,146,306 in 1904 were made and consumed in establishments where manufactured.

The total quantity of mixed acid manufactured in 1909 cannot

be shown because the amount made and consumed in establishments where manufactured was not reported; but the quantity manufactured by establishments making it as a chief product and by those reporting it as a by-product was 57,182,168 pounds in 1909, compared with 65,331,327 in 1904, a decrease of 12 per cent.

The production of establishments making mixed acid as a chief product was 45,361,626 pounds in 1909 and 42,812,894 in 1904, an increase of 6 per cent.

A comparative summary follows, giving the general statistics for the industry in 1904 and 1909:

SULPHURIC, NITRIC AND MIXED ACIDS—GENERAL SUMMARY, 1909 AND 1904.

	—Census.—		Per cent. of increase,
	1909	1904.	1904-1909.
Number of establishments	42	32	+ 31
Capital	$18,726,000	$12,762,000	+ 47
Cost of materials	5,386,000	4,973,000	+ 8
Salaries and wages	2,046,000	2,062,000	— 1
Salaries	551,000	556,000	— 1
Wages	1,495,000	1,506,000	— 1
Miscellaneous expenses	916,000	713,000	+ 28
Value of products	9,884,000	9,053,000	+ 9
Value added by manufacture (products less cost of materials)	4,498,000	4,080,000	+ 10
Employees:			
Number of salaried officials and clerks	330	308	+ 7
Average number of wage earners employed during the year	2,252	2,447	— 8
Primary horse-power	6,494	5,416	+ 20

Continuous Carbon Dioxide and Temperature Recorder.

In connection with the paper of Mr. Edward A. Uehling, printed elsewhere in this issue, a description of the Uehling continuous carbon-dioxide and temperature recorder should be of particular interest. It is made by the Uehling Instrument Company, of Passaic, N. J., and consists of a carbon dioxide and temperature measuring machine as shown in Fig. 1, to which are connected by small copper tubing standard type M Uehling pressure and vacuum recorders, properly calibrated for CO_2 or temperature.

Indicators of the standard type Z construction, which are based on the elementary U-tube principle may of course be connected to the machine in addition to the recorders. Thus, for instance, a standard equipment for a boiler plant would consist of the carbon dioxide and temperature machine shown in Fig. 1, which has at the front a CO_2 and a temperature indicator, and in addition to this machine an additional indicator located at the boiler front, and two type M recorders located in the engineer's office, one calibrated for CO_2, and the other for temperature.

Another style machine is shown in Fig. 2. This is a double combined machine, making two measurements of CO_2 and two measurements of temperature. It will be noted that at the front of the machine there are two indicators for temperature and two for CO_2. The full equipment comprises two other indicators placed at the front of the boilers, for the CO_2, and four standard recorders, two calibrated for temperature and two calibrated for carbon dioxide.

The two styles of machines illustrated and the two equipments above described are typical of the numerous arrangements which are possible when a continuous carbon dioxide and temperature machine of this type is used. Standard-style machines include those giving four measurements of CO_2 and no temperature measurements, or again, three measurements of CO_2 and one of temperature, or two measurements of CO_2, or else simply one measurement of CO_2. In every case the equipment can consist of as many recorders and indicators as desired, although the standard equipment comprises extra indicators at the boiler front for CO_2 and one standard recorder for each measurement of CO_2 and temperature.

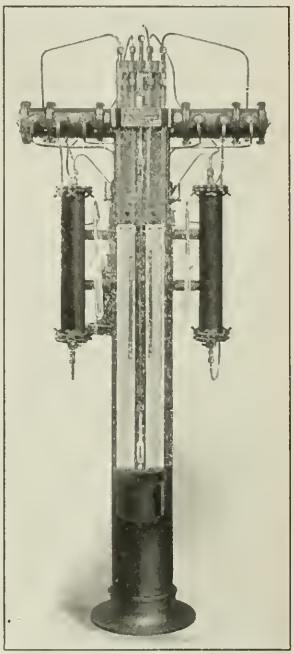


FIG. 2.—DOUBLE CO_2 AND TEMPERATURE RECORDER

This flexibility of equipment as also the ability to use standard recorders and indicators, and to obtain continuous measurement, all follow from the fact that the Uehling continuous carbon dioxide temperature machine operates on the pneumatic principle. By suction apparatus in the machines, a constant vacuum is maintained which draws the gas to be analyzed for CO_2 through two apertures Between the two apertures, the CO_2 contents of the gas are removed by caustic potash. The more CO_2 contained in the gas, the more is removed and the less gas has to pass through the last aperture, and therefore the higher will be the vacuum between the two apertures, so that the vacuum or suction in this chamber becomes a measure of the amount of CO_2 contained in the sample of gas drawn from the boiler. A vacuum indicator and recorder connected to this chamber will measure the per cent CO_2, and readings can be obtained if the CO_2 chart is properly calibrated.

The same principle is applied to measuring the temperature. Here, however, the two apertures through which the gas, or air, as it is in this case, passes, are not kept at the same temperature. In applying the pneumatic principle to the measurement of temperature, one aperture is located in a nickel tube which is placed in the hot gas whose temperature is to be measured, while the other aperture is kept at a constant lower temperature. As the air has a higher temperature when passing through the first aperture than through the second, the suction or vacuum in the chamber between these apertures will increase with increase of temperature, because the volume of air varies directly with the temperature (absolute). Therefore, the vacuum between the apertures needs only to be transmitted to the indicators or recorders by small copper tubing, in order to obtain any number of accurate and reliable measurements.

Water Power Withdrawals.—Over a million and a half acres of public land now stand withdrawn by the Government as being valuable for the possible development of water power. During October the United States Geological Survey recommended the withdrawal of 54,422 acres as having power possibilities and the restoral of 2600 acres in Montana and Wyoming because field examination by the Survey showed them not to be valuable for that purpose.

Welding.

At a meeting of the American Society of Mechanical Engineers held in New York, on Nov. 14, the general subject of autogenous welding was discussed in great detail. Mr. H. R. Cobleigh presented an introductory paper dealing with the general aspects of the subject, in which were described the origin and principles of each process and the apparatus used in each. He outlined the features of electric resistance welding, electric arc welding, thermit welding, oxy-acetylene welding and oxy-hydrogen welding. By means of numerous lantern slides the mechanical details of the apparatus employed in each system of welding were described. It was stated that each method possesses certain advantages and certain disadvantages, so that each system may be considered most advantageous for certain particular services. The oxy-acetylene flame produces a higher temperature than oxy-hydrogen, but operates under the disadvantage of a marked tendency to flash back. This tendency has been overcome by the use of special forms of torches, which require the gases to pass through porous material before being ignited. The oxy-hydrogen apparatus has a less tendency to flash back, but the temperature is not so high. The oxygen and hydrogen for the oxy-hydrogen apparatus can conveniently be produced by electrolytic dissociation of water. The acetylene used in the oxy-acetylene torch is produced from calcium carbide, according to the methods employed with illuminating acetylene. The acetylene is produced by dropping water upon the carbide, or by dropping the carbide into water, the latter method having certain advantages. The oxygen used in the oxy-acetylene flame can be produced either by chemical or electrolytic processes. In any event, it is most advantageous to convey the gases in highly compressed form.

Mr. G. E. Pelissier described the Goldschmidt thermit process of welding, which depends upon the chemical combination of aluminium with oxide of iron, and the resultant products being oxide of aluminium and iron. By the use of motion pictures the thermit welding of rails was shown in a striking and most interesting manner. This process has found numerous applications in heavy engineering, such as that in connection with locomotive and ship machinery repairing.

Mr. C. B. Auel gave much interesting information relating to the Bernardos, Slavianoff and Zerener processes of electric welding. These processes differ according to the material used for electrodes. In the first-named process one electrode is formed by the metal to be welded, and the other electrode is of copper. In the second process the two metallic parts to be joined form the electrodes, while in the third process there are two carbon electrodes, with the metal forming an intermediate contact between these two electrodes. Much attention was devoted by the author to the Bernardos process, which he considered to be the most universally applicable. However, the Slavianoff process produces a stronger joint. In this process use is made of an arc of a length of about ¼ in., an emf force of 25 volts and a current of 150 amp. In the Bernardos process the length of the arc would be about 1 in., the current from 160 to 700 amp and the emf from 45 to 50 volts at the arc.

Mr. W. H. Brown described a welder acting according to the Zerener process, which he has found highly advantageous on account of the economy in operation, the consumption being 18 kw as compared with 50 kw by other electrical processes.

The subject was discussed by Messrs. W. H. Spire, J. D. Mooney, Henry Cave, W. R. Noxon, W. H. Lavin, W. J. Fritz, J. F. Springer, E. B. Katte and Dr. Lieber.

Mr. Noxon called attention to the stoppage of leaks by means of oxy-acetylene welding in a pipe line in central Colorado, which carries water under a head of 1875 ft. The cost of welding this pipe line was only a small fraction of the cost which would have been involved in replacing the pipe line.

Mr. Lavin stated that the oxy-hydrogen process has been widely adopted for regular use in industrial establishments for welding purposes. He claimed as an advantageous feature the fact that the oxygen and hydrogen gases for the flame can be obtained in a high degree of purity by the electrolytic method.

Mr. Fritz showed numerous illustrations of the oxy-acetylene flame, which is now being employed for dismantling the ill-fated battleship Maine in the Havana harbor.

Mr. Springer attributed the high temperature of the oxy-acetylene flame to the concentration of heat units in the small space.

Dr. Lieber stated that the Blau gas which is being used in welding is compressed and liquified until it occupies only 1/400 part of its original volume. As compared with acetylene having 1500 heat units per cu. ft., the Blau gas has 1853 heat units.

New Detinning Works.

The Goldschmidt Detinning Company, of New York City, is erecting a new detinning plant in East Chicago, Ind., with a capacity of about 40,000 tons of tin scrap per year. This plant will have several novel and very interesting features.

Instead of using exclusively tin scrap, the company is making all preparations also for the use of old cans, provided they are obtainable in sufficient quantity and at sufficiently low cost. Further the process to be used will be entirely different from the one now employed by the company at their Chrome, N. J., plant. Instead of making anhydrous tetrochloride of tin as is done in Chrome, they will make pig tin in East Chicago.

The buildings for the plant will be completed within a few weeks and it is expected that the plant will be in operation in the Spring of 1912.

Fixation of Atmospheric Nitrogen in Norway.

Arrangements have recently been made, according to *Tidens Tegn*, Kristiania, Norway, between the Badische Anilin and Soda Fabrik and the Norsk Hydroelektrisk Kvaelstofaktieselskab, to the effect that the management of the large airsalpeter developments in Norway will be laid in the hands of the Norwegians, while it has up till now been in charge of the German company. Mr. Sam Eyde, joint inventor with Professor Birkeland of the nitric acid process bearing their name, is to be at the head of the undertaking.

The greater part of the capital interested in the new and old developments is French. The Germans have about one and one-third million dollars, and the Canadians two and two-third million dollars. Swedish as well as Norwegian capital is also quite heavily interested.

The paid-up capital is sufficient to finish the large developments of water power at Rjukan, which is going to be used exclusively in the airsalpeter industry.

NOTES

Unveiling of Bust of Wolcott Gibbs at Chemists' Club.—On the evening of Saturday, Nov. 25, the bust of the late Professor Wolcott Gibbs will be unveiled with appropriate exercises at the Chemists' Club in New York. The bust will be presented on behalf of the Gibbs family by Mr. Geo. Gibbs and will be accepted by Dr. Moore as president of the club. Prof. Charles E. Munroe will deliver the main address. There will be a reception afterwards and a fine musical program has been arranged. The ladies of the club members have been specially invited.

The American Platinum Works of Newark, N. J., have acquired the business of the Croselmire & Ackor Co.

The **Standard Silver Co., Ltd.**, of Toronto, Canada, are building a new five-story factory for the manufacture of silver-plated hollow ware (on Britannia metal, brass, German silver and copper).

The **Geary Water-Tube Boiler** is the subject of a neat and well illustrated little pamphlet of the manufacturers, the Oil City Boiler Works, of Oil City, Pa.

The Care of the Balance.—Messrs. Voland & Sons, makers of precision balances and weights in New Rochelle, N. Y., have just issued a neat little pamphlet on the "use and care of precision balances." It contains elementary but important instructions on the setting up of a balance, when received from the factory, how to place the parts in position, how to obtain the equilibrium, how to clean it, how to release the beam, where to put the balance, how to lubricate, when this should be necessary, etc. There are also some tables on weights and measures and decimal equivalents.

Dredging for Gold is the title of an interesting pamphlet distributed at the recent California meeting of the American Institute of Mining Engineers and containing a well illustrated description of the two newest gold dredges installed by the Natomas Consolidated, of San Francisco and Sacramento. They are of the California endless chain type, having digging buckets of 15 cubic feet capacity. They dig to a depth of 55 feet below the water line of the pond in which they float. They have a maximum theoretical capacity of about 650 cubic yards per hour and have, under favorable conditions, handled 500 cubic yards per hour of running time in continuous operations in formation extremely difficult to handle.

Experiments in slime filtration carried on by George J. Young at the Mackay School of Mines, Reno, Nev., lead him to believe, as stated in a paper presented to the American Institute of Mining Engineers, that: the vacuum filter is best adapted to the treatment of slime containing a large proportion of clayey material; the pressure filter does its best work on slime which contains much coarse and fine sand; while for material containing a small proportion of fine clayey material and much fine sand, either vacuum or pressure filters could be used with perhaps equally good results. Of the vacuum filters, the thin-cake continuous filters are a decided improvement over the thick-cake filters.

Electrolytic Production of Iron.—A recent British patent (28,226, Oct. 19, 1911) of Mr. S. O. Cowper-Coles is mentioned in London *Electrical Engineering*. It relates to an electrolytic process, in which the cathodes consist of thin sheets of electrolytic iron, and the anodes of bars of pig iron of any kind. The electrolyte can be sulphuric or hydrochloric acids, or a mixture of both, or a ferrous ammonium sulphate. Any thickness of iron may be deposited on the cathode, which can be removed from the cell and the iron taken off. To prevent the anodes becoming coated with carbon, graphite or other impurities, brushes are mounted on a spindle, and attached to the cathode. These revolve and keep the anode surface free.

Seger Cones.—In last year's report of the German Reichsanstalt numerous tests of Seger cones in ceramic kilns in practice are mentioned, which were accompanied by thermo-electric measurements. It was found that the Seger cones broke down in the ceramic kilns at considerably lower temperatures than in the electric furnaces of the Reichsanstalt. This may be due to the long time of heating (60 hours), and perhaps also to a chemical effect of the furnace gases. At any rate, this fact must be taken into consideration in using Seger cones for determining temperatures.

Asbestos is most widely used as a fireproof cloth for theater curtains. Other important uses are the making of firemen's clothing, and insulating materials for both steam and electric power plants.

Sealing Wires in Glass Globes.—A method of sealing wires into glass, which, while primarily intended for mercury-vapor lamps, may have wider use, is described in a recent British patent (No. 1880 of Oct. 12) of Mr. G. B. Burnside. As noted in London *Electrical Engineering*, the wire is placed through a small aperture in the glass or other vitreous substance and heated by a blowpipe flame until perfect cohesion between the two takes place. The seal is then removed from the flame, and when at a dull red heat it is cooled by gradually increasing immersions, of about two or three seconds' duration, in some sperm or similar oil or in wax. For currents above 15 amp. round tubular conductors, plugged with a conducting material, are employed. This form is very useful for mercury-vapor lamps or rectifiers, as the tubular shell is made of a substance that does not form an amalgam, and thus protects the copper from the mercury.

Personal

Mr. **Jacques Abady**, a director of the English firm of Alexander Wright & Co., was recently in this country on a two weeks' visit to their American branch, the Precision Instrument Company of Detroit, Mich. In addition to his business activities Mr. Abady is a barrister, the author of Abady's Gas Analyst's Manual, and other works on combustion, and one of the inventors of the Simmance-Abady combustion recorder and the Simmance-Abady vacuum and pressure gauges. His visit had to do with several new inventions which have passed the experimental stage and which will be shortly on the market for general use in steam and gas plants.

Mr. H. Standish Ball, late of McGill University, Montreal, has left for South Africa, where he will accept a position. His address is P. O. Box 2536, Johannesburg, Transvaal, South Africa.

Mr. R. B. Carnahan, Jr., Vice-President of the American Rolling Mill Company of Middletown, Ohio, has returned from a six months' business trip through Europe in the interest of their well-known American ingot iron.

Mr. Arthur H. Carpenter has returned to Denver after an extended stay in Nevada on professional work.

Mr. C. W. Carpenter, of New York, secretary of the Colorado Vanadium Company, recently made a trip of inspection to the company's mines in southwestern Colorado. He was accompanied by Mr. W. F. Bleecker, metallurgist of the company, and Mr. Kirby Thomas, consulting mining engineer.

Mr. J. V. N. Dorr, metallurgist, of Denver, has been appointed general manager of the Mogul Mining Company, Pluma, S. D., for which he has been consulting engineer for some time.

Mr. Arthur Lakes, Jr., of Denver, has gone to Ymir, B. C., to take charge of the mines and mill of the Ymir-Wilcox Development Company.

Mr. James C. Lawrence has resigned his position as chemical engineer in the U. S. Forest Service, to take up the practice of his profession, and has accepted the position as consulting engineer in charge of the operations of the Forest Products Chemical Company of Memphis, Tenn.

Mr. Warren B. Lippincott, who has been identified with the zinc industry for a number of years, has left the Colorado field and gone to Missouri. His partnership with J. M. McClave, of Denver, has been dissolved and his office will be at Joplin, Mo.

Mr. J. B. McIntosh, formerly of Salt Lake City, has removed his office to Los Angeles. He will have charge of the construction of the new copper smelter to be built for the United Verde Copper Company at Jerome, Ariz.

Mr. C. H. Repath, consulting engineer, of Los Angeles, Cal., has charge of the designing of the new copper smelter for the United Verde Copper Company at Jerome, Ariz.

Mr. B. F. Rice, vice-president and general manager of the McKesson Concentrator Company, of Colorado Springs, has opened an office in the Ideal Building, Denver.

Mr. Etienne A. Ritter, of Colorado Springs, has been examining the carnotite deposits in Utah.

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore Treatment (*Continued*).

495,637, April 18, 1893, Johannes Pfleger, of Frankfort-on-the-Maine, Germany.

Zinc. Electrodeposits zinc from solutions, or extracts pure zinc from zinc ore, hard zinc, zinc alloys, etc., by use of basic zinc salts obtained by dissolving oxide of zinc in concentrated solutions of sulphates or chlorides of zinc. When used as an electrolyte, all dissolved impurities, such as copper, are precipitated. Like impurities from soluble anodes cannot be dissolved, as basic copper chloride or sulphate is insoluble in the basic zinc solution. The basic zinc salt may be added to neutral solutions; or, with oxide of zinc in suspension, to acid salts. When used as electrolyte, formation of a spongy zinc deposit is prevented. To the solution of basic zinc salt may be added a good conducting neutral salt, such as chloride of sodium or magnesium, of sulphate of potash, and the solution circulated through the electrolytic bath. If necessary zinc oxide may be intermittently or continuously added.

495,715, April 18, 1893, Samuel R. Whitall, of Salt Lake City, Utah.

Gold and silver. Treats chloride, sulphide or arsenide ores containing, besides gold and silver, copper, iron and other impurities, by crushing to pass 16 to 40 mesh screen, then pouring over the dry ore a solution containing ½% to 1½% (of quantity of ore) potassium cyanide, with about 1/3 this quantity of sodium hyposulphite. Water is added in quantity sufficient to cover the ore and allowed to stand until precious metals are dissolved. Gold and silver are then precipitated by zinc or by electrolysis, platinum or carbon anodes being used and a cathode of wires or strips of metal, preferably iron, arranged in parallel lines on copper wires. Gold and silver will be deposited on wires as fine powder, which is collected, dried and melted into bars. Barium, calcium or sodium cyanides may be used instead of potassium cyanide.

If the ore is siliceous, it is first treated with caustic soda and steam under pressure to form a soluble silicate, thereby freeing the precious metals from their combinations with base metals. If the ore is sulphide, it is first given a chlorinating roast with salt and lixiviated with water. Copper or lead contained in the wash water is precipitated by any well-known means.

497,014, May 9, 1893, Frederick W. Cleghorn, of Salt Lake City, Utah, Assignor to Charles G. Brown, of same place.

Gold and silver. The ore is pulverized and screened through a No. 10 screen and saturated in a tank with solution of hyposulphite of soda and sulphuric acid while heated by steam. The sulphuric acid neutralizes impurities of solution. The gold and silver are precipitated as sulphides by calcium or sodium sulphide dried, roasted and cupelled.

The ore remaining in the tank is saturated with sulphuric acid, or muriatic acid, and salt. The gold and silver are precipitated from the solution as chlorides by metallic iron and passing an electric current through the solution. The remaining solutions may be used again.

501,997, July 25, 1893, Stephen H. Emmens, of London, England.

Iron, copper, etc. The apparatus comprises a shallow rectangular vat of paraffined wood. The sides and bottom of the vat are lined with carbon, which forms the anode. An inner rectangular wall of paraffined wood is placed on the carbon floor, leaving a space about four inches wide between it and the carbon side lining on all four sides. The space between the inner wall and the sides of the vat is filled with the ore or other material and a layer of such material about six inches deep is placed on the carbon floor. The electrolyte (such as an aqueous solution of ammonium sulphate or chloride) is poured into the vat until there is a depth of about four inches above the ore, but within the inner rectangular wall. The cathodes consist of a series of rectangular perforated plates suspended parallel with the floor and connected to cross bars extending across the vat, the cathodes being hung about two inches from the layer of ore. The electrolyte can pass underneath the inner wall and saturate the material between the inner wall and the wall of the vat, this arrangement preventing the short-circuiting between the cathode and the ore. The acid constituent of the electrolyte unites with the metal or metallic oxide to form a solution which is in turn decomposed, the metal being deposited and the acid set free to take up a fresh charge of metal from the ore. The perforations in the cathodes facilitate the escape of hydrogen and circulation of the fluid in the upper portion of the vat. When iron ore or scrap iron is being treated for the production of malleable iron, paraffin oil may be poured on the surface of the electrolyte to prevent oxidation. A screen of coarse canvas impregnated with paraffin oil may be supported beneath the cathodes to collect material falling from the cathodes. The apparatus is also applicable to reduction of copper ores, treatment of jewellers' sweepings, sulphides, and other metalliferous substances.

BOOK REVIEWS.

Die Metallurgie des Wolframs.—Praktisches Handbuch von Dr. **Hans Mennicke.** 416 pages, 39 illus. Price 15 Marks in paper cover, 17 Marks bound in cloth. Berlin: M. Krayn.

This book is intended primarily as a reference book. To many it will seem almost incredible that the metallurgy of tungsten requires so bulky a volume, yet the information contained in these pages is for the most part very concise. Although but a few decades ago tungsten was classed as a rare metal, it has in recent years taken a very prominent place among the useful metals. The author has covered every article and patent pertaining to tungsten that have appeared in the last twenty years.

Among the chapter headings are: Ores, Reduction Methods, By-products including scandium, titanium, tantalum and niobium, Manufacture of Sodium Tungstate, Tungstic Acid, Alloys and Compounds, Tungsten Filaments, Fire-proofing Cotton and other Fabrics, and Analytical Methods. Although the chapters on tungsten steel and on tungsten-filament manufacture are comparatively brief and the information based largely on patent literature, the chapters on the production of tungsten powder are particularly well treated, the author having had considerable experience in this line. The book will be welcomed not only by research students but also by steel metallurgists and incandescent-lamp manufacturers.

The Soil Solution. By **Frank K. Cameron.** 125 pages. Price $1.25. Easton, Pa.: The Chemical Publishing Company.

This is an interesting elementary chemical text-book on the nutrient medium for the growth of plants in soil, written from the physico-chemical standpoint. As a rule, books on this subject are either intensely technical and difficult, or they are popular in character so as to aid the practising farmer or horticulturist in raising crops for the market.

This book occupies a middle position. It is neither so technical as to appeal only to specialists, on the one hand, nor so popular and practical as to appeal solely to business men on the other. The book will interest the scientific student of agriculture, the student of applied chemistry and the scientifically minded members of the public at large.

Its thirteen chapters deal with the following topics: The soil, soil management, soil analysis, plant-food theory, dynamic soil phenomena, film water, mineral constituents, absorption by soils, plant growth and concentration, supply and removal of mineral nutrients, organic constituents, fertilizers and alkali.

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